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THE MINERAL INDUSTRY

ITS

STATISTICS, TECHNOLOGY AND TRADE

DURING

1908

FOUNDED BY RICHARD P. ROTHWELL

EDITED BY

WALTER RENTON INGALLS

Editor of the Engineering and Mining Journal; Member American Institute of Mining Engineers; Member Institution of Mining and Metallurgy; Member Mining and Metallurgical Society of America; Member American Chemical Society; Member Society of Chemical Industry, Etc.

VOLUME XVII

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THE STATE GEOLOGICAL SURVEYS.

Of the 45 States of the Union, 33 have organized geological surveys, these comprising nearly all of the States in which the mining industry is important. The organization of a geological survey in Massachusetts is under consideration. Certain States which have no geological survey have officials who give attention to the mining industry. Thus, California has a State mineralogist, while Tennessee has a State mine inspector who collects statistics of mineral production. The States having organized geological surveys, together with the names and addresses of the respective State geologists, as of Jan. 1, 1909, are given in the following list:

STATE GEOLOGISTS.

State.	Name and Address.	State.	Name and Address.
Alabama	Eugene A. Smith, University	Missouri	H. A. Buehler, Rolla
Arizona	Wm. P. Blake, Tucson	Nebraska	E. H. Barbour, Lincoln
Arkansas	A. H. Purdue, Fayetteville	New Jersey	H. B. Kümmel, Trenton
Connecticut	Wm. N. Rice, Hartford	New York	John M. Clark, Albany
Colorado	R. D. George, Boulder	N. Carolina	Joseph Hyde Pratt, Chapel Hill
Florida	E. H. Sellards, Tallahassee	N. Dakota	A. G. Leonard, Grand Forks
Georgia	S. W. McCallie, Atlanta	Ohio	J. A. Bownocker, Columbus
Illinois	H. Foster Bain, Urbana	Oklahoma	Charles N. Gould, Norman
Indiana	W. S. Blatchley, Indianapolis	S. Carolina	Earle C. Sloan, Charleston
Iowa	Samuel Calvin, Des Moines	S. Dakota	E. C. Perisho, Vermillion
Kentucky	C. J. Norwood, Lexington	Vermont	G. H. Perkins, Burlington
Kansas	Erasmus Haworth, Lawrence	Virginia	Thos. L. Watson, Charlottesville
Louisiana	G. D. Harris, Ithaca, N. Y.	Washington	Henry Landes, Seattle
Maine	L. A. Lee, Brunswick	West Virginia	I. C. White, Morgantown
Maryland	William Bullock Clark, Baltimore	Wisconsin	E. A. Birge, Madison
Michigan	Alfred C. Lane, Lansing	Wyoming	H. C. Beeler, Cheyenne
Mississippi	A. F. Crider, Jackson		

STATE MINE INSPECTORS, COMMISSIONERS, ETC.

State.	Name and Address.
Arkansas	G. B. Tucker, Commissioner, Bureau of Mines, Manufactures and Agriculture, Little Rock
California	L. E. Aubury, State Mineralogist, San Francisco
Colorado	T. J. Dalsell, Commissioner of Mines, Denver
Idaho	F. Cushing Moore, State Mine Inspector, Boise
Michigan	J. L. Nankervis, Commissioner of Mineral Statistics, Calumet
Missouri	J. W. Marstellar, Secretary, Bureau of Mines and Mines Inspection, Jefferson City
Montana	William Walsh, State Mine Inspector, Helena
New Mexico	J. E. Sheridan, Mine Inspector for the Territory of N. M., Silver City
Ohio	George Harrison, Chief Inspector of Mines, Columbus
Pennsylvania	James Roderick, Chief, Department of Mines, Harrisburg
South Dakota	Nicholas Treweek, Jr., State Mine Inspector, Lead
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VALUES OF FOREIGN COINS.

ESTIMATE BY DIRECTOR OF THE MINT, JAN. 1, 1909.

COUNTRY.	Standard	Monetary Unit.	Value in Terms of U.S. Gold Dollar.	Coins.
Argentine Republic	Gold	Peso	$0.965	Gold: argentine ($4.824) and ½ argentine. Silver: peso and divisions.
Austria-Hungary	Gold	Crown	.203	Gold: 10 and 20 crowns. Silver: 1 and 5 crowns.
Belgium	Gold	Franc	.193	Gold: 10 and 20 francs. Silver. 5 francs.
Bolivia	Silver	Boliviano	.365	Silver: boliviano and divisions.
Brazil	Gold	Milreis	.546	Gold: 5, 10, and 20 milreis. Silver: ½, 1, and 2 milreis.
British Possessions, N. A. (except Newf'nd).	Gold	Dollar	1.000	
Central Amer. States—				
Costa Rica	Gold	Colon	.465	Gold: 2, 5, 10, and 20 colons ($9.307). Silver: 5, 10, 25, and 50 centimos.
British Honduras	Gold	Dollar	1.000	
Guatemala, Honduras, Nicaragua, Salvador	Silver	Peso	.365	Silver: peso and divisions.
Chile	Gold	Peso	.365	Gold: escudo ($1.825), doubloon ($3.6 0), and condor ($7.300). Silver: peso and divisions.
China	Silver	Tael. Amoy	.596	
		Tael. Canton	.597	
		Tael. Chefoo	.572	
		Tael. Chin Kiang	.584	
		Tael. Fuchau	.553	
		Tael. Haikwan (customs)	.609	
		Tael. Hankow	.560	
		Tael. Kiaochow	.590	
		Tael. Nankin	.592	
		Tael. Niuchwang	.561	
		Tael. Ningpo	.575	
		Tael. Peking	.583	
		Tael. Shanghai	.546	
		Tael. Swatow	.553	
		Tael. Takau	.602	
		Tael. Tientsin	.580	
		Dollar. Hongkong	.393	
		Dollar. British	.393	
		Dollar. Mexican	.376	
Colombia	Gold	Dollar	1.000	Gold: condor ($9.647) and double condor. Silver: peso.
Denmark	Gold	Crown	.268	Gold: 10 and 20 crowns.
Ecuador	Gold	Sucre	.487	Gold: 10 sucres ($4.8665). Silver: sucre and divisions.
Egypt	Gold	Pound (100 piasters)	4.943	Gold: pound (100 piasters), 5, 10, 20, and 50 piasters. Silver: 1, 2, 5, 10, and 20 piasters.
Finland	Gold	Mark	.193	Gold: 20 marks ($3.859), 10 marks ($1.93).
France	Gold	Franc	.193	Gold: 5, 10, 20, 50, and 100 francs. Silver: 5 francs.
German Empire	Gold	Mark	.238	Gold: 5, 10, and 20 marks.
Great Britain	Gold	Pound sterling	4.866½	Gold: sovereign (pound sterling) and ½ sovereign.
Greece	Gold	Drachma	.193	Gold: 5, 10, 20, 50, and 100 drachmas. Silver: 5 drachmas.
Haiti	Gold	Gourde	.965	Gold: 1, 2, 5, and 10 gourdes. Silver: gourde and divisions.
India (British)	Gold	Pound sterling*	4.866½	Gold: sovereign (pound sterling). Silver: rupee and divisions.
Italy	Gold	Lira	.193	Gold: 5, 10, 20, 50, and 100 lire. Silver: 5 lire.
Japan	Gold	Yen	.498	Gold: 5, 10, and 20 yen. Silver: 10, 20, and 50 sen.

NOTE.—The coins of silver-standard countries are valued by their pure silver contents, at the average market price of silver for the three months preceding January 1, 1909.

* The sovereign is the standard coin of India, but the rupee ($0.3244½) is the current coin, valued at 15 to the sovereign.

COUNTRY.	Standard	Monetary Unit.	Value in Terms of U.S. Gold Dollar.	Coins.
Liberia	Gold	Dollar	1.000	
Mexico	Gold	Peso†	.498	Gold: 5 and 10 pesos. Silver: dollar‡ (or peso) and divisions.
Netherlands	Gold	Florin	.402	Gold: 10 florins. Silver: 2½, 1 florin, and divisions.
Newfoundland	Gold	Dollar	1.014	Gold: 2 dollars ($2.027).
Norway	Gold	Crown	.268	Gold: 10 and 20 crowns.
Panama	Gold	Balboa	1.000	Gold: 1, 2½, 5, 10, and 20 balboas. Silver: peso and divisions.
Persia	Silver	Kran	.067	Gold: ¼, 1, and 2 tomans ($3.409). Silver: ½, ¼, 1, 2, and 5 krans.
Peru	Gold	Libra	4.866⅔	Gold: ½ and 1 libra. Silver: sol and divisions.
Philippine Islands	Gold	Peso	.500	Silver peso: 10, 20, and 50 centavos.
Portugal	Gold	Milreis	1.080	Gold: 1, 2, 5, and 10 milreis.
Russia	Gold	Ruble	.515	Gold: 5, 7½, 10, and 15 rubles. Silver: 5, 10, 15, 20, 25, 50, and 100 copecks.
Spain	Gold	Pesata	.193	Gold: 25 pesetas. Silver: 5 pesetas.
Straits Settlements	Gold	Pound sterling§	4.866⅔	Gold: sovereign (pound sterling). Silver: dollar and divisions.
Sweden	Gold	Crown	.268	Gold: 10 and 20 crowns.
Switzerland	Gold	Franc	.193	Gold: 5, 10, 20, 50, and 100 francs. Silver: 5 francs.
Turkey	Gold	Piaster	.044	Gold: 25, 50, 100, 250, and 500 piasters.
Uruguay	Gold	Peso	1.034	Gold: peso. Silver: peso and divisions.
Venezuela	Gold	Bolivar	.193	Gold: 5, 10, 20, 50, and 100 bolivars. Silver: 5 bolivars.

† Seventy-five centigrams fine gold.
‡ Value in Mexico, $0.498.
§ The current coin of the Straits Settlements is the silver dollar issued on Government account, and which has been given tentative value of $0.567758⅓.

CHRONOLOGY OF MINING IN NORTH AMERICA IN 1908.

January 2—Canadian branch of the Royal Mint opened at Ottawa.

January 4—President Roosevelt, in a letter to Governor Sparks, announced that U. S. troops would remain at Goldfield, Nev., until the legislature could take steps to preserve order.

January 12—Report of commission appointed by President Roosevelt to inquire into labor troubles at Goldfield, Nev., was rather unfavorable to the mine owners and Governor Sparks.

January 15—Utah Mine Owners' Association organized to secure better treatment from smelters.

January 16—Nevada legislature unanimously asked President Roosevelt to keep troops at Goldfield until arrangements could be made to preserve order.

January 26—Work resumed by Fairmont Coal Company in Monongah mine, W. Va., wrecked by the explosion of Dec. 6, 1907.

January 27—Nevada legislature passed a law establishing a State police organization.

February 1—At a conference of 75 steel manufacturers at New York it was decided that the prevailing prices of steel were justified, in spite of much sentiment in favor of a reduction of prices.

February 2—Secretary Garfield, of the Department of the Interior, in his annual report recommended a system of leases to preserve the coal lands of the United States; also asked for increase of appropriation for the Geological Survey to investigate means for minimizing accidents and dangers in mining.

February 3—A number of the smaller steel plants in the Pittsburg district prepared to resume operations following the settlement of the price question at the New York conference.

February 5—Three thousand miners in 10 mines owned by the Pittsburg Coal Company, situated in the Youghiogheny valley, struck because they were asked to use other than black powder.—Rail manufacturers and railroads adopted a heavier and improved section for steel rails.

February 6—The cases in New Mexico against Phelps, Dodge & Co., alleging land frauds, were compromised and indictments dropped.

February 8—Pipe line between Port Arthur, Tex., and Tulsa, Okla., 582 miles long, completed, to go into regular operation in April.

February 10—Explosion of gas in the colliery of the Moody Coal Company, near Central City, Ky., blew off the top of the shaft and killed nine men.

February 13—Heavy sales of copper at large cuts in price.

February 19—Sales of copper for export made at prices as low as 12c., immense transactions having been made on the decline from 13c.—Several zinc smelters in Kansas prepared to close down owing to unfavorable conditions in ore and metal market.

February 25—Officials of the United States Smelting Company, at Salt Lake City, announced the discovery of a solution of the smelter smoke problem by the neutralization of sulphuric anhydride by means of zinc oxide and then filtering the smoke.

March 3—Anaconda Copper Company resumed operations in Montana.—North Butte and Butte Coalition companies resumed operations.

March 4—Calumet & Hecla and Wolverine companies reduced dividends.

March 7—Federal troops left Goldfield and Nevada State police assumed responsibility of maintaining order.

March 13—Furnaces at Washoe plant, Aanaconda, Mont., started up.

March 16—De Beers Mining Company, South Africa, decided to close its Dutoitspan diamond mine.

March 17—Premier Diamond Company refused to renew agreement with the diamond syndicate for sale of its output.

March 20—Steel men in conference again voted to maintain prices.

March 27—Federal troops ordered to Douglas Island, Alaska, to protect property threatened by 800 striking miners at the Treadwell mine.—Stock of Yukon Gold Company offered to the public.

March 28—Explosion in mine No. 1 of the Union Pacific Coal Company at Hanna, Wyoming, killed 70 miners.—Yukon Gold stock sold at 8.

March 31—Yukon Gold sold at 5.

April 1—Coal miners to the number of 250,000, belonging to the United Mine Workers of America, suspended work pending the adoption of a new wage agreement with the operators.

April 7—The United States Smelting, Refining and Mining Company secured a modification of the perpetual injunction against its Salt Lake smeltery and permission to run three years to demonstrate the efficiency of its bag-house plant to prevent damage by smeltery fumes.

April 16—Foundry-iron producers met in New York and under the leadership of U. S. Steel Corporation officials agreed to maintain prices.

April 19—Half the first unit of the Steptoe Valley concentrator, Ely, Nev., began operations.

April 22—Judge Burns, of the U. S. Circuit Court for southern Texas, gave a decision in the zinc-ore case, upholding the position of the importers and sustaining the previous decision of the U. S. Board of General Appraisers in all particulars.

May 4—Southern pig-iron manufacturers agreed not to reduce prices. —Drop in Montgomery-Shoshone stock explained by an official statement that the ore at depth had proved disappointing.

May 8—The collapse of pig-iron selling association in the eastern United States officially announced.

May 12—The British Columbia Copper Company resumed operations after six months' idleness.

May 13—Conference at the White House of governors and other prominent men to consider conservation of national resources.

May 21—The bill to establish a bureau of mines in the United States Department of the Interior passed by the House.—The Anaconda Copper Company reported a deficit for 1907 of $2,921,780.

May 22—Steel manufacturers again agreed to maintain prices.

May 28—Arguments in the Calumet & Hecla-Osceola case concluded at Grand Rapids, Michigan.

June 1—Amalgamated Copper Company reported to its stockholders a loss of $7,500,000 in net income for the last fiscal year.

June 2—United States Steel Corporation reduced the price of bar steel from $32 to $28 per ton.

June 3—A deputation from the Mexican Chamber of Mines called upon Minister Molina and entered formal protest against the provision in the proposed new mining law restricting the ownership of Mexican mines by foreigners.

June 4—The price of bar iron reduced $3 per ton.

June 9—Conference of steel manufacturers decided to cut prices of all steel-finished products except rails. A reduction in the price of ore also agreed upon.

June 18—Bar iron sold on "open market" basis. Bar steel and merchant steel sold at lower prices.

June 19—Two thousand miners in Colorado coalfields stopped work, 30 mines being affected.

June 20—Explosion in Elsworth No. 1 mine of the Pittsburg Coal Company, Monongahela City, Penn., killed several miners and injured others.

June 22—Eleven men killed in an explosion of firedamp in the Loire colliery at St. Etienne, France.—Operations begun at the Steptoe Valley smelting works.

June 23—Fires ordered lighted in 1000 idle ovens at Connellsville.

July 1—A new wage scale with slight reductions adopted for the year in the sheet and tin-plate industry.

July 2—The steel syndicate in Germany withdrew from the price agreement.—International Tube-makers Association dissolved.—An explosion of gas in the Rikovsky coal mine, Yusovo, European Russia, killed about 200 men.—The courts of Venezuela decided that the New York & Bermudez Asphalt Company must pay its $15,000,000 fine.

July 9—Converter plant at the Steptoe smelter blown in.

July 11—The smelting plant of the Cananea Consolidated Copper Company resumed operations with numerous improvements.

July 15—Explosion in a coal mine of the Susquehanna Coal Company, at Williamstown, Penn., killed 12 men and seriously injured eight.

July 20—It was announced that on Sept. 1, the Mexican government would put into effect the advance in freight rates on ores authorized several months previously, in spite of the protests of smelter and mining interests.—Chlorination mills in Cripple Creek, Colo., increase treatment charges on ores.

July 22—The De Beers company decided to shut down the De Beers diamond mine, Kimberley, South Africa, on July 31.

July 23—The first cargo of 12,000 tons of iron ore arrived at the docks at Gary, Indiana, the new plant of the U. S. Steel Corporation.

July 25—The Mexican cabinet referred the proposed anti-foreign law to President Diaz with power to accept, enforce or reject the measure.

August 11—Explosion of firedamp in the Dodweiler colliery at Saarbrücken, Germany, killed 12 men and seriously injured eight others.

August 17—Indiana coal miners' strike affected 10,000 men. Strikes in Ohio affected 28 mines.—Low water in the Ohio river closed the coal mines along the Monongahela river.

August 18—Explosion in the Maypole colliery at Wigan, England, destroyed headframe and fans and killed 76 miners.

August 26—Fire in the Hailey-Ola coal mine No. 1, near Haileyville, Okla., killed 30 miners.

August 27—The Alvarado Consolidated Mines Company, a Maine corporation, with $10,000,000 capital, took over the lease of the Palmillo mine at Parral, Chihuahua, from the Coram syndicate of Boston.

September 4—Fire destroyed the town of Rawhide, Nevada.

September 8—THE MINERAL INDUSTRY, Vol. XVI, published.

September 10—The United States Circuit Court for the Eastern District of Pennsylvania decided that the "commodity clause" affecting the transportation by railroads of coal owned by them is unconstitutional.

September 14—It was announced that all properties of the Tennessee Coal, Iron and Railroad Company would be in operation on October 1.

September 16—The Montana State Railway Commission ordered cuts in freight rates on ore shipments from all parts of the State.

September 18—Vice-President Ramon Corral of Mexico declared semi-officially that the section of the new mining law obnoxious to foreigners would not be passed.

September 24—Trustees in bankruptcy offered for sale the property of the Consolidated Arizona Smelting Company, including the smelter at Humboldt, the Blue Bell mine and stock in the De Soto Mining Company.

September 30—Agreement among European aluminum producers was dissolved.

October 1—The director of the U. S. Mint announced the resumption of the purchase of about $125,000 worth of silver per week.

October 4—Judge Knappen, of the U. S. Court at Grand Rapids, Mich., decided that the Calumet & Hecla company had a right to vote its stock in the Osceola company, a right disputed by A. S. Bigelow.

October 14—Portion of Bisbee, Ariz., destroyed by fire.—Fire in the König coal mine, Königshütte, Silesia, destroyed a number of lives.

October 15—Article 144 of proposed new Mexican mining law, commonly described as the anti-foreign law, decisively beaten.

October 16—Balaklala smelter, California, began operations.

October 17—Sylvanite, new gold camp near Hachita, N. M., organized.

October 19—U. S. Smelting, Mining and Refining Company purchased control of the Bullion-Beck & Champion company.

October 24—Receiver appointed for the Greene Gold-Silver Company found practically no assets.

October 27—The quarterly statement of the U. S. Steel Corporation showed a gain of nearly $7,000,000 in net earnings as compared with that of the last preceding quarter.

November 5—Very large sales of copper at advancing prices.

November 7—Excitement in copper market, with violent advance.

November 10—Tariff hearings began before Ways and Means Committee.—Property of Consolidated Arizona Smelting Company bought by Hooley, Learned & Co. at trustees' sale in Arizona, as preliminary to reorganizing the company by the stockholders and creditors.—Western Pacific railway open for passenger traffic from Salt Lake City to Ely, Nevada.

November 12—Firedamp explosion at Radbod colliery, Hamm, Westphalia, Germany, killed 300 men and wrecked mine.—Extraordinary buying of U. S. Steel common at highest prices in history.—January Mining Company sued Bunker Hill & Sullivan for $7,300,000 for ore taken from disputed vein.

November 13—U. S. Steel Corporation announced the intention to spend several millions on plant for production of steel car wheels.

November 14—El Oro, Mexico, partially destroyed by fire.

November 15—Collapse of attempts to reorganize the four pig-iron syndicates of Germany, and demoralization in the German pig-iron market.

November 16—Four plants of the Illinois Steel Company resumed operations.—Announced that three blast furnaces at Gary, Ind., would be blown in Dec. 15, and that steel making would begin Jan. 1.

November 17—Reported that Mexican Petroleum Company had merged with Waters-Pierce Oil Company, to fight S. Pearson & Sons.—Stockholders of Virginia-Carolina Chemical Company approved issue of $15,000,000 15-year 5 per cent. bonds.

November 19—Utah Consolidated announced contract with a new smelting company and abandonment of its own plan to build works.

November 23—Andrew Carnegie declared that American steel works can produce more cheaply than all competitors and require no tariff protection.

November 28—Explosion in Marianna mine of Pittsburg-Buffalo Coal Company killed 130 men.

December 3—Experimental station of the U. S. Geological Survey at Pittsburg, Penn., formally opened by Secretary Garfield.

December 8—National Conservation Commission made its first report.

December 10—President Roosevelt withdrew from entry all public lands containing phosphate rock in Wyoming, Idaho and Utah.—Conference of steel manufacturers at New York, but no action as to prices.

December 11—Announcement that Phelps, Dodge & Co., Incorporated, capital $50,000,000, is to take over all the interests of the firm of the same name.

December 15—Mexican Congress adjourned until April without taking action on new mining regulations.

December 17—Announcement that Cole & Ryan would organize new company to compete with American Smelting and Refining Company.

December 18—Copper producers met at New York to consider revival of statistical association.

December 21—Andrew Carnegie declared before the Ways and Means committee that the American iron and steel industry no longer needs protection.—Incorporation of International Smelting and Refining Company (the Ryan-Cole company), with capital of $50,000,000, all common stock.—Manufacture of steel begun at Gary, Ind.

December 26—President Roosevelt appointed commissioners to consult with Canada and Mexico regarding the conservation of natural resources.

December 28—Earthquake in southern Italy and Sicily destroyed thousands of lives and millions of dollars' worth of property.

INTRODUCTION.

In the preparation of the statistics for this volume, the figures previously reported for 1907, and in some cases for earlier years, have been revised in the light of later and more minute investigation, in accordance with our regular practice; therefore it is important for all who have occasion to refer to them to observe the caution to *use always the figures in the latest volume of* THE MINERAL INDUSTRY. There are no statistical reports of this nature that are absolutely correct, owing to the practical impossibility of obtaining accurate data from all the producers in some extensive and greatly subdivided industries, the absence of records on the part of many producers, which prevents them from making returns, the unwillingness of a few to give their figures, and the confusion as to the stage in which many products are to be reported. The last difficulty is especially likely to lead to errors in values, some producers estimating the worth of their product at the pit's mouth, and others reporting it in a more or less advanced state of completion, including thus not only the cost of carriage, but also the cost of manipulation. These difficulties appear not only in our statistics, but also in those reported by various governments. In our own work we make a practice of going backward and correcting figures previously reported, whenever mistakes are discovered by subsequent investigation. In estimating values, we are disposed to use actual market prices rather than the values reported by the producers themselves, which are apt to be misleading for the reasons mentioned above.

For many of the statistics relating to the mineral production of the United States in 1908 and previous years, we are indebted to the U. S. Geological Survey; for the production of gold and silver in the United States to Frank A. Leach, Director of the Mint, and for the statistics of American imports and exports to O. P. Austin, chief of the bureau of statistics of the Department of Commerce and Labor. Acknowledgment is due also to various State geological surveys and statistical bureaus for information incorporated in this volume. In the text and footnotes to the various tables, we have generally credited such information to the proper sources, but this acknowledgment may stand for any unintentional oversight. The same acknowledgment is due with respect to the foreign statistics, which we state generally as officially reported by the respective governments, when such reports are available.

In publishing this volume of THE MINERAL INDUSTRY at a comparatively early date in the year following that to which it especially relates, the editorial work upon it having been completed at the end of May, and the last proof-reading having been done in June, it has been impossible to collect statistics for all the substances of mineral production in the United States, but the omissions are generally in the cases of those of minor importance. In many instances it has been possible to make use of the statistics collected by the U. S. Geological Survey for substances whereof an independent investigation in behalf of THE MINERAL INDUSTRY was not made. Many of the statistics of the U. S. Geological Survey are now published with commendable promptness and with an accuracy that makes it unnecessary to enter into the same duplication as formerly. The statistics for foreign countries are given in all cases for the latest year available. In several cases our work has been greatly facilitated by the courtesy of the statisticians of foreign governments in sending us their reports for 1908 in manuscript in advance of the regular publication in print. This has been highly helpful in enabling us to present this complete summary of 1908 so soon after the close of the year. In the preparation of THE MINERAL INDUSTRY there is always a temptation to postpone the conclusion of the editorial work pending the receipt of missing figures, but it has been considered that May 31 is a reasonable date at which to draw the line, with a view to combining the maximum of completeness with the maximum of promptness. Some of the statistics reported in this volume are preliminary, and subject to revision. It is our belief, however, that statistics of reasonable commercial accuracy, promptly published, are of greater value to technology and trade than are statistics, correct to the last unit, which are published a year or two late.

The list of contributors to Vol. XVII is noteworthy for its length and the high standing of all. The reviews of industrial progress are uniformly prepared by recognized experts in each line, while the records of current events are supplied by thoroughly informed reporters. Thus, the progress in the mining industry of each mineral is to a large extent described by the State geologists, or members of the State geological surveys, nearly all of whom are contributors to Vol. XVII, as they were to Vol. XVI. These contributors give an official character to the publication, which constitutes a valuable and appreciated medium for collating and bringing to general attention the statistical researches and commercial investigations in the several States. This is work that is most properly done by the State geological surveys, and THE MINERAL INDUSTRY affords them a means for the coördinated presentation of reports that otherwise might remain buried in local libraries.

PRODUCTION OF ORES AND MINERALS IN THE UNITED STATES.

Products.	Measures.	1907		1908	
		Quantity.	Value.	Quantity.	Value.
Antimony ore	Sh. T.	210	$28,432	360	$19,800
Asbestos	Sh. T.	950	11,700	1,350	24,000
Asphaltum	Sh. T.	(w) 223,861	2,826,489	(w)	.....
Barytes	Sh. T.	65,579	251,308	38,546	130,409
Bauxite	Lg. T.	(w) 97,776	480,330	(w) 52,167	263,968
Borax	Sh. T.	48,444	(w)	33,500	117,500
Chrome ore	Lg. T.	335	5,020	280	5,600
Coal, anthracite	Sh. T.	86,341,832	172,871,146	80,329,578	159,122,961
Coal, bituminous	Sh. T.	393,692,113	468,932,577	337,929,632	387,678,972
Diatomaceous earth	Sh. T.	(w)	104,406	(w)	.....
Emery	Sh. T.	(w) 1,069	12,394	790	10,360
Feldspar	Sh. T.	(w) 92,799	558,044	59,180	319,570
Flint	Sh. T.	(w) 75,561	407,499	64,220	318,000
Fluorspar	Sh. T.	36,350	202,736	39,389	212,647
Fuller's Earth	Sh. T.	34,039	323,375	30,517	270,685
Garnet	Sh. T.	6,723	209,895	2,530	78,090
Graphite, amorphous	Sh. T.	(w) 26,962	138,381	(w) 14,443	75,250
Graphite, crystalline	lb.	4,586,149	149,548	3,433,039	149,763
Gypsum	Sh. T.	(w) 1,751,748	4,942,264	(w) 1,721,639	4,138,560
Iron ore	Lg. T.	52,955,070	117,560,255	32,789,987	60,821,976
Limestone flux	Lg. T.	15,722,801	7,480,121	9,563,158	4,720,485
Magnesite	Sh. T.	6,405	57,720	8,967	52,342
Manganese ore (*d*)	Lg. T.	(w) 5,604	63,369	(w) 6,344	64,659
Mica sheet	lb.	(w) 1,060,182	349,311	(w) 972,964	234,021
Mica scrap	Sh. T.	(w) 3,025	42,900	(w) 2,417	33,904
Molybdenum ore	Sh. T.	(w)	.....	15	6,000
Monazite	lb.	(w) 547,948	65,754	(w) 422,646	50,718
Petroleum, crude	Bbl. (*i*)	164,347,930	123,260,948	184,711,413	136,347,831
Phosphate rock	Lg. T.	2,251,459	10,450,522	2,375,031	15,040,882
Pumice	Sh. T.	(w) 8,112	33,818	(w) 10,569	39,287
Pyrite	Lg. T.	261,871	851,346	206,471	744,463
Quartz, crystalline	Sh. T.	(w) 22,977	157,094	(w)	.....
Salt	Bbl. (*k*)	(w)29,704,128	7,429,551	(w)28,745,319	7,486,804
Sand, glass	Sh. T.	1,187,296	1,250,067	(w)	.....
Slate roofing	Squares (*f*)	(w) 1,277,554	4,817,769	(w) 1,333,171	6,316,817
Sulphur	Lg. T.	307,806	6,427,025	307,750	6,795,120
Talc, common	Sh. T.	(w) 72,010	905,047	(w)	.....
Talc, fibrous	Sh. T.	59,000	501,500	70,739	697,390
Tin ore	Sh. T.	63	15,209	50	12,500
Tungsten ore	Sh. T.	1,468	715,031	497	126,281
Whetstones and oilstones	.....	.....	(w) 264,188	.....	(w)
Zinc ore	Sh. T.	902,923	(w)	838,377	(w)

Additional details will be found under the respective captions farther on in this volume. (*d*) Does not include manganiferous iron ore. (*e*) Estimated. (*f*) One square covers 100 square feet. (*i*) Barrels of 42 gallons. (*k*) Includes salt used in manufacture of alkali; the barrel of salt weighs 280 lb. (w) Figures reported by the United States Geological Survey. (w) Statistics not collected.

PRODUCTION OF METALS IN THE UNITED STATES. (*z*)

Products.	Measures.	1907		1908	
		Quantity.	Value.	Quantity.	Value.
Aluminum	lb.	26,000,000	$10,920,000	13,000,000	$4,095,000
Antimony	lb.	5,794,000	859,830	1,600,000	128,000
Copper	lb.	879,241,766	181,660,141	948,196,490	127,058,329
Ferromanganese (*q*)	Lg. T.	339,348	21,887,946	152,018	6,460,765
Gold, fine	Troy oz	4,314,742	89,191,726	4,659,562	96,313,256
Iron, pig	Lg. T.	25,442,013	580,077,896	15,784,000	267,540,378
Lead	Sh. T.	350,130	37,288,845	314,067	26,381,628
Nickel	lb.	*Nil.*	.....	500,000	250,000
Platinum	Troy oz.	357	7,443	510	11,655
Quicksilver	Flasks (*o*)	20,932	868,678	17,969	805,690
Silver, fine	Troy oz.	56,514,700	36,919,358	51,798,053	27,382,523
Sodium	Sh. T.	2,000	1,000,000	2,000	1,000,000
Tin	Sh. T.	(v) 1,662	914,404	1,200	707,160
Zinc (*y*)	Sh. T.	249,612	29,763,735	210,511	19,897,500

Additional details will be found under the respective captions farther on in this volume. (*e*) Estimated. (*o*) Flasks of 75 lb. (*q*) Includes spiegeleisen, although the value is given as for ferromanganese. (*v*) Recovered from scrap metal. (w) Statistics not available. (*z*) Includes only metal produced from domestic ores except in case of zinc. (*y*) Includes zinc from foreign ore.

PRODUCTION OF SECONDARY MINERALS AND CHEMICALS IN THE UNITED STATES.

Products.	Measures.	1907		1908	
		Quantity.	Value.	Quantity.	Value.
Alum	Sh. T.	(u) 10,404	$361,900	(u) 7,700	$236,710
Aluminum sulphate	Sh. T.	(u) 106,821	2,008,046	(u) 97,255	1,835,213
Alundum	lb.	6,751,444	405,086	3,160,000	189,600
Ammonium sulphate	Sh. T.	89,000	5,511,770	87,600	5,247,240
Arsenic	lb.	2,020,000	101,000	2,603,505	99,193
Borax, refined	Sh. T.	(w)	.....	17,115	1,537,688
Boric acid	Sh. T.	2,672	23,547	3,067	406,767
Bromine	lb.	1,062,000	138,060	1,149,000	103,410
Calcium chloride	Sh. T.	45,000	450,000	32,000	224,000
Carborundum	lb.	7,532,670	451,960	4,907,170	294,430
Cement, nat. hyd.	Bbl. (g)	(u) 2,887,700	1,467,302	(u) 1,621,862	808,509
Cement, portland	Bbl. (h)	(u)48,785,390	53,992,551	(u)51,002,612	43,472,679
Cement, pussolan	Bbl. (i)	(u) 557,252	443,998	(u) 151,451	95,468
Cement, slag	Bbl.	2,129,700	2,449,155	4,535,300	3,628,240
Cobalt oxide	lb.	*Nil.*	.....	100,000	140,000
Coke	Sh. T.	40,595,238	107,445,050	23,496,666	55,595,792
Copper sulphate (c)	lb.	34,867,650	2,804,228	37,654,961	1,833,796
Copperas	Sh. T.	26,771	294,481	35,334	388,674
Crushed steel	lb.	840,000	58,800	630,000	44,100
Graphite, artificial	lb.	6,924,000	483,717	7,385,511	502,667
Lead, white	Sh. T.	111,409	12,254,990	116,628	10,515,315
Lead, sublimed, white	Sh. T.	8,700	1,026,600	9,100	973,000
Lead, red	Sh. T.	13,370	1,778,717	11,358	1,156,282
Lead, orange, mineral	Sh. T.	815	123,917	393	43,157
Litharge	Sh. T.	14,769	1,624,553	12,254	1,231,206
Manganese residuum	Lg. T.	93,413	280,239	110,225	220,450
Mineral wool	Sh. T.	9,008	81,769	9,197	77,228
Zinc oxide (m)	Sh. T.	85,390	7,731,100	65,100	5,876,342

Additional details will be found under the respective captions further on in this volume. (c) Does not include sulphate made from metallic copper. (e) Estimated. (g) Barrels of 265 lb. (h) Barrels of 380 lb. (m) Includes a small quantity made from spelter. (i) Barrels of 330 lb. (u) Figures reported by the United States Geological Survey. (w) Statistics not collected.

PRODUCTION OF THE RARE METALS IN THE UNITED STATES.

Products.	Measures.	1907		1908	
		Quantity.	Value.	Quantity.	Value.
Bismuth	lb.	10,000	$12,500	5,200	$6,600
Cadmium	lb.	15,000	18,750	10,000	9,300
Calcium	lb.	350	613	350	525
Magnesium	lb.	*Nil.*	.....	500	500
Vanadium	lb.	41,000	102,500	64,800	162,000

Additional details will be found under the respective captions farther on in this volume. (e) Estimated. (z) Statistics not available.

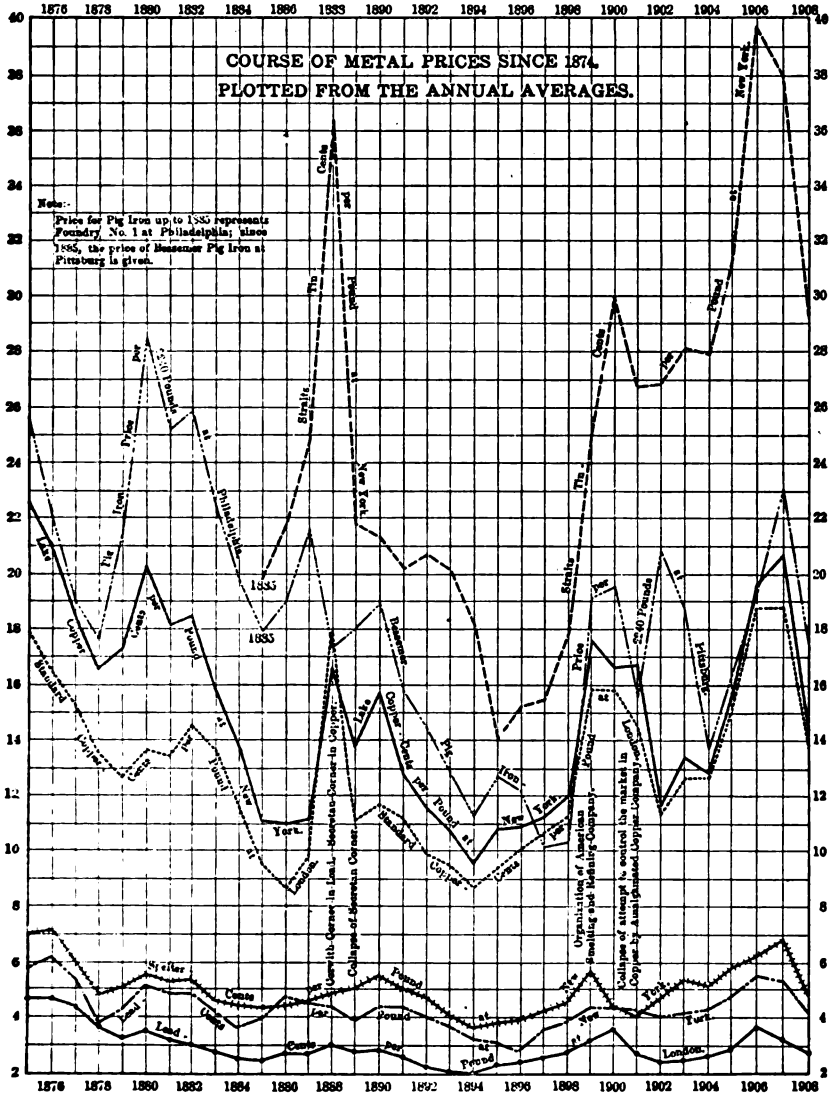
SUMMARY.

Alum and Aluminum Sulphate.—The 1908 production of both of these salts decreased from the figures for 1907. The output of the respective years was: 1907, alum, 10,404 short tons, valued at $34.78 per ton; aluminum sulphate, 106,821 short tons, valued at $18.80 per ton; 1908, alum, 7700 short tons at $30.74 per ton and aluminum sulphate, 97,255 short tons, worth $18.87 per ton.

Aluminum.—The production of aluminum in the United States in 1908 is estimated at 13,000,000 lb. against 26,000,000 lb. in 1907. These statistics are estimated upon the basis of known furnace capacity. The output in 1908 was valued at $4,095,000 as against $10,920,000 in 1907.

Alundum.—In 1908 3,160,000 lb. were produced, valued at $189,600, against 6,751,444 lb. valued at $405,086 in 1907.

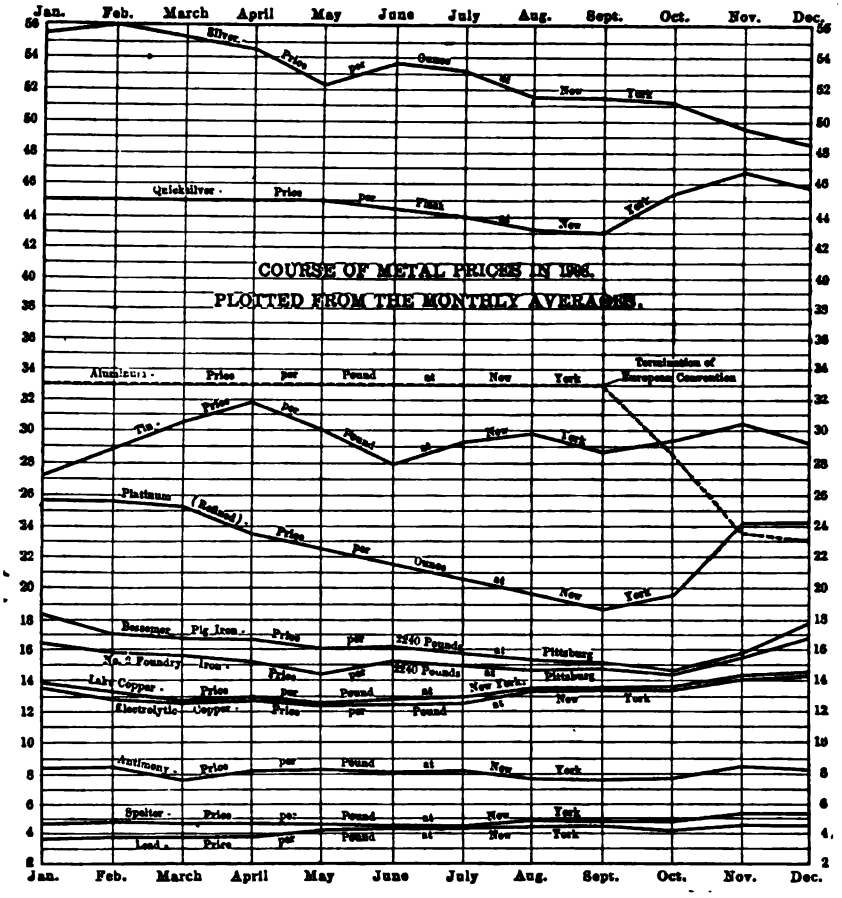
Ammonia and Ammonium Sulphate.—The production in 1908 is esti-



mated at 87,600 short tons valued at $5,247,240 against 89,000 short tons valued at $5,511,770 in 1907. These figures include the production of all forms of ammonia expressed in equivalent sulphate.

Antimony.—Only 1,600,000 lb. of antimony worth about $128,000 was produced in the United States in 1908. In 1907 the production was 5,794,000 lb. worth $859,830.

Antimony Ore.—The production in 1908 was 360 short tons ($19,800) as compared with 210 short tons ($28,432) in 1907.



Arsenic.—The production of arsenious acid increased from 2,020,000 lb. valued at $101,000 in 1907 to 2,603,505 lb. valued at $99,193 in 1908.

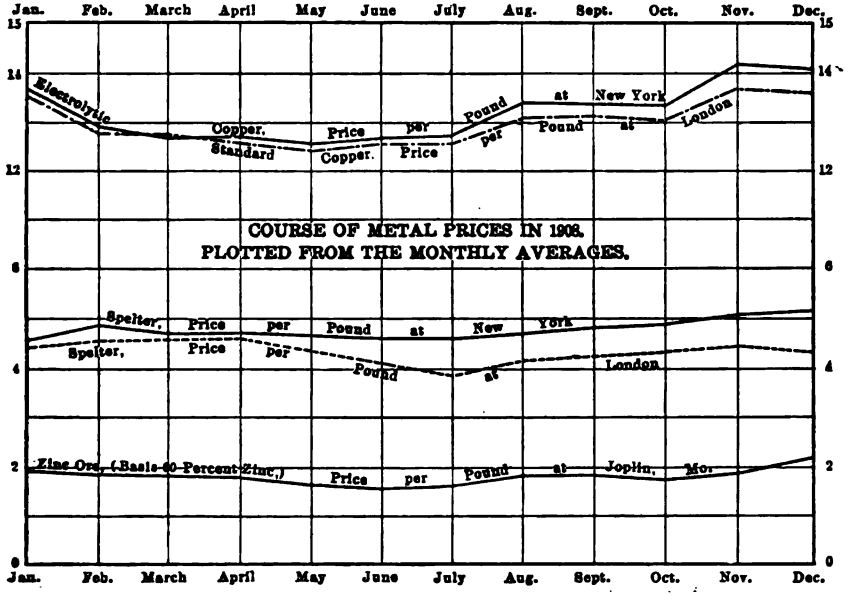
Asbestos.—The production of asbestos in 1908 was 1350 short tons ($24,000), which is an increase of 400 tons in quantity and $12,300 in value over 1907.

Barytes.—Production of barytes fell off badly in 1908, amounting to only 38,546 short tons ($130,409) as compared with 65,579 short tons ($251,308) for 1907. Kentucky produced 11,051 short tons in 1908.

Bauxite.—The 1908 production of bauxite amounted to 52,167 long tons ($263,968) against 97,776 tons ($480,330) in 1907.

Bismuth.—The production in 1908 was not as great as that in the initial year in this industry in the United States. Only 5,200 lb., valued at $6600, were produced against 10,000 lb., valued at $12,500, in 1907.

Borax.—The 1908 production of crude borax in the United States was 33,500 short tons valued at $117,500 against 48,444 short tons for 1907. In 1908 3067 short tons of boric acid ($406,767) and 17,115 short tons of refined borax ($1,537,688) were produced.



Bromine.—In 1908 there were produced 1,149,000 lb. of bromine ($103,410) against 1,062,000 lb. ($138,060) in 1907.

Cadmium.—The production of cadmium in the United States in 1908 was 10,000 lb. valued at $9300. This was the second year that cadmium was produced in this country, 15,000 lb. valued at $18,750 having been made in 1907.

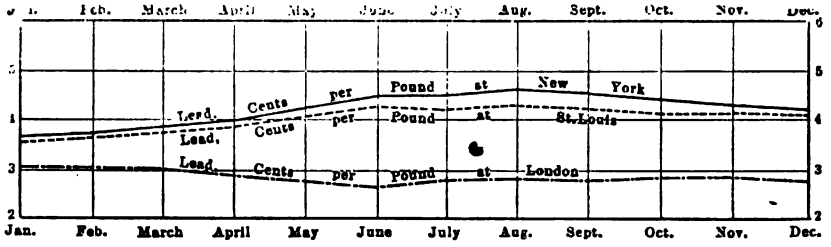
Calcium.—The production of calcium in 1908 was 350 lb. valued at $525. The first metallic calcium was produced in the country in 1907 when 350 lb. valued at $613 were made.

Calcium Chloride.—In 1908 the United States produced 32,000 short tons of calcium chloride valued at $224,000, against 45,000 short tons valued at $450,000 in 1907.

Carborundum.—The output of carborundum in the United States in 1908 was 4,907,170 lb. valued at $294,430. In 1907 the production was 7,532,670 lb. valued at $451,960.

Cement.—The total production of all kinds of cement in the United States during 1908 was 52,775,925 bbl. valued at $44,376,656. This output was divided as follows: Portland cement, 51,002,612 bbl., $43,472,679; natural cement, 1,621,862 bbl., $808,509; puzzolan cement, 151,451 bbl., $95,468. There was a heavy decrease of production in 1908 as compared with 1907 in most of the eastern and southern States, the loss being greatest in Pennsylvania. The average price of the entire portland cement output in 1908 was only 85c. per bbl.—36c. below the average in 1907. The 1908 production of slag cement was 4,535,300 bbl., against 2,129,700 bbl. for 1907.

Chrome Ore.—In 1908 the production of chrome ore was 280 long tons ($5600), a decrease from the 1907 output of 335 tons ($5620).



[COURSE OF LEAD PRICES DURING 1908 PLOTTED FROM MONTHLY AVERAGES.

Coal.—The production of coal in 1908 amounted to 418,259,210 short tons ($546,801,933) as compared to 480,033,945 short tons ($640,803,723) in 1907. Of the 1908 production 80,329,578 short tons were anthracite, with an average value of $1.98 per ton at the pit mouth, and 337,929,632 short tons bituminous coal, worth $1.13 per ton at the mines. The average for anthracite in 1907 was $1.99 and for the bituminous $1.19, the production being 86,341,832 tons of the former and 393,692,113 tons of the latter.

Cobalt Oxide.—In 1908 the North American Lead Company of Missouri began producing cobalt oxide and turned out about 100,000 lb. worth $140,000.

Coke.—In 1908 the United States produced 23,496,666 short tons of coke worth $55,595,792 ($2.36 per ton), while in 1907 the production amounted to 40,595,238 short tons worth $107,445,050 ($2.65 per ton).

Copper.—The production of copper in 1908 was larger than that in either 1906 or 1907. The total production of the smelteries for 1908

was 948,196,490 lb. The 1908 output, however, was valued at only $127,058,329 against $181,660,141 for that of 1907, which amounted to 879,241,766 lb. Arizona led the States in 1908 with a production of 290,167,795 lb. Montana and Michigan were close with outputs of 252,558,330 lb. and 222,267,444 lb. respectively. The United States produced approximately 55 per cent. of the world's output for 1908.

Copperas.—There was an increased output of copperas in 1908 as compared with 1907, amounting to 8563 short tons in quantity and $94,193 in value. The total 1908 production was 35,334 short tons valued at $388,674.

Copper Sulphate.—The production in 1908 was 37,654,961 lb. against 34,867,650 lb. in 1907. These figures do not include that which is produced by the treatment of metallic copper. They represent chiefly the by-product of the electrolytic copper refiners, but include a small production that is obtained directly from matte.

Crushed Steel.—In 1908 the production of crushed steel was 630,000 lb. ($44,100), against 840,000 lb. ($58,800) in 1907. Crushed steel is used as an abrasive. It is produced by only one concern in the United States.

Emery.—The United States produced 790 short tons of emery ($10,360) in 1908, against 1069 tons ($12,294) in 1907.

Feldspar.—In 1908 the United States produced 59,180 short tons ($319,570) of feldspar. The 1907 production was much larger, amounting to 92,799 short tons ($558,944). Maine, Pennsylvania and Virginia were the leading producers.

Ferromanganese.—In 1908 there were 40,642 long tons of ferromanganese and 111,376 long tons of spiegeleisen produced in the United States, which, together with a few tons of ferrophosphorus, brought the total output up to more than 152,018 long tons. The 1907 production was 339,348 long tons.

Flint.—Less flint was quarried in the United States in 1908 than in 1907 by 11,341 short tons ($99,699). The 1908 production was 64,220 short tons worth $318,000.

Fluorspar.—The output in 1908 was 39,389 short tons ($212,647), against 36,350 tons ($202,736) in 1907. Kentucky contributed about 33 per cent. of the total output of the United States.

Garnet.—In 1908 the production of garnet amounted to 2530 short tons ($78,090), against 6723 tons ($209,895) in 1907.

Gold.—In 1908 the output of the United States was 4,659,562 oz. ($96,313,256) as against 4,314,742 oz. ($89,191,726) in 1907. Colorado, Alaska and California led the producing States, in the order named.

Nevada was the only other State with a production over $10,000,000 and its production fell off badly from that for 1907.

Graphite.—The production of crystalline graphite was 3,433,033 lb. worth $149,763. In 1907 the production was 4,586,149 lb. valued at $149,548. The 1908 production of amorphous graphite amounted to 1443 short tons valued at $75,250; in 1907 the output was 26,962 short tons valued at $138,381. Despite depressed trade conditions the output of artificial graphite continued to increase. The total production was 7,385,511 lb. worth $502,667, against 6,924,000 lb. worth $483,717 in 1907.

Gypsum.—The output of gypsum in 1908 was 1,721,829 short tons ($4,138,560) against 1,751,748 tons ($4,942,264) in 1907. In 1908 Michigan produced 250,000 tons valued at over $700,000.

Iron Ore.—In 1908 the production of iron ore fell off badly, amounting to only 33,789,987 long tons valued at $60,821,976, as against 52,955,070 tons valued at $117,560,255 in 1907. The total value for 1908 is figured on an average of $1.80 per long ton of ore, delivered at lake ports, from the Lake Superior field, or f.o.b. in the other regions. Of the 1908 production 33,789,987 tons came from the Lake Superior region, 26,014,987 long tons from the South and 1,875,000 long tons from other States. All districts produced less in 1908 than in the previous year, but the South fell off less than either the Lake Superior field or New York and the central States section.

Lead.—In 1908 the total lead production of the United States was 314,067 short tons worth $26,381,628. Missouri contributed 123,613 tons to the total, Idaho 98,394 tons, Utah 43,995 tons and Colorado 26,707 tons. The 1907 production was 350,130 short tons worth $37,288,845.

Lead Products.—The domestic output of lead products during 1908 was as follows: White lead, 116,628 short tons worth $10,515,315 against 111,409 short tons worth $12,254,990 in 1907; sublimed white lead, 9100 short tons worth $973,000 against 8700 short tons worth $1,026,600 in 1907; red lead, 11,358 short tons worth $1,156,282 against 13,370 short tons worth $1,778,717 in 1907; orange mineral, 393 short tons worth $43,157 against 815 short tons worth $123,917 in 1907; litharge, 12,254 short tons worth $1,231,206 against 14,769 short tons worth $1,624,553 in 1907.

Limestone Flux.—The amount of limestone quarried in the United States in 1908 for use as a flux is calculated as 9,563,158 long tons, valued at $4,720,485. This was used as follows: 8,658,558 tons as iron flux; 324,600 tons for lead and 580,000 tons for copper smelting. The amount used as flux in 1907 was 15,722,801 tons, valued at $7,480,121.

Magnesite.—In 1908 the amount of magnesite produced was 8967 short tons valued at $52,342. This is larger by 2562 tons than the 1907

production, but the value of the output is lower by $5378. California contributed all of the production each year.

Magnesium.—The output of this metal in the United States amounted to 500 lb., worth $500.

Manganese Ore.—The 1908 production was 6344 long tons worth $64,659 against 5604 tons worth $63,369 in 1907. In addition to this there was 51,554 long tons of manganiferous silver ore valued at $123,407 produced in 1908.

Manganese Residuum.—There was 110,225 long tons ($220,450) of manganese residuum produced in the United States in 1908, against 93,413 long tons ($280,239) in 1907.

Mica.—The production of scrap and sheet mica in the United States in 1908 was as follows: Scrap, 2417 short tons, value $33,904; sheet, 972,964 lb., value $234,021. The 1907 production was 3025 short tons ($42,800) of scrap and 1,060,182 lb. ($349,311) of sheet mica.

Mineral Wool.—The production of mineral wool in 1908 was 9197 short tons valued at $77,228 as against 9008 short tons valued at $81,769 in 1907. The chief producers continued to be the U. S. Mineral Wool Company of New York and the Columbia Mineral Wool Company of Chicago.

Molybdenum Ore.—In 1908 the United States produced 15 tons of molybdenum ore worth approximately $6000.

Monazite.—The production in 1908 amounted to 422,646 lb. of concentrated 90 per cent. sand worth $50,718. North Carolina fell off badly in its production but in South Carolina the industry gained materially.

Nickel.—1908 was the first year in which nickel was produced from domestic ores in the United States. The output of the one producing company, the North American Lead Company of Fredericktown, Mo., was about 500,000 lb. worth $250,000.

Petroleum.—The total production was 184,711,413 bbl. of 42 gal. valued at $136,347,831 as against 164,347,930 bbl. valued at $123,260,948 in 1907. In 1908 Oklahoma was the largest producer, California and Illinois following in order. The Lima and Appalachian fields both showed a decreased output for the year.

Phosphate Rock.—The production in the United States in 1908 was 2,375,031 long tons ($15,040,882) as compared with 2,251,459 tons ($10,450,522) in 1907. The chief producing States were Florida, Tennessee and South Carolina, in the order named.

Pig Iron.—The average price of pig fell off from $22.80 per long ton in 1907 to $16.95 in 1908, and the production decreased from

AVERAGE MONTHLY PRICES OF CHEMICALS, EARTHS, MINERALS, ETC., IN 1908.

(IN CARLOAD LOTS AT NEW YORK)

MATERIAL	UNIT.	JAN.	FEB.	MAR.	APR.	MAY.	JUNE.	JULY.	AUG.	SEPT.	OCT.	NOV.	DEC.	YEAR.
Abrasives:														
Corundum	Lb.	$0.07	$0.07	$0.07	$0.07	$0.07	$0.07	$0.07	$0.07	$0.07	$0.17	$0.07	$0.07	$0.07
Emery, Turkish flour	Lb.	0.0175	0.0175	0.0175	0.0175	0.0175	0.0175	0.0175	0.0175	0.0175	0.0175	0.0175	0.0175	0.0175
Emery, Naxas flour	Lb.	0.0175	0.0175	0.0175	0.0175	0.0175	0.0175	0.0175	0.0175	0.0175	0.0175	0.0175	0.0175	0.0175
Emery, Chester flour	Lb.	0.0150	0.0150	0.0150	0.0150	0.0150	0.0150	0.0150	0.0150	0.0150	0.0150	0.0150	0.0150	0.0150
Emery, Peekskill flour	Lb.	0.0125	0.0125	0.0125	0.0125	0.0125	0.0125	0.0125	0.0125	0.0125	0.0125	0.0125	0.0125	0.0125
Garnet	Sh. T.	30.00	30.00	30.00	30.00	30.00	30.00	30.00	30.00	30.00	30.00	30.00	30.00	30.00
Boric acid	Lb.	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07
Borax, sacks	Lb.	0.0425	0.0425	0.0425	0.0425	0.0425	0.0425	0.0425	0.0425	0.0425	0.0425	0.0425	0.0425	0.0425
Alum, lump	100 Lb.	1.75	1.75	1.75	1.75	1.75	1.75	1.75	1.75	1.75	1.75	1.75	1.75	1.75
Aluminum sulphate	100 Lb.	1.25	1.10	1.10	1.10	1.10	1.10	1.10	1.10	1.10	1.10	1.10	1.10	1.112
Ammonium sulphate	100 Lb.	3.075	3.075	3.075	3.075	3.075	3.075	3.075	3.075	3.075	3.075	3.075	3.075	3.075
Arsenic, white	Lb.	0.0475	0.05125	0.0500	0.0425	0.0350	0.03375	0.0350	0.0350	0.0325	0.03125	0.03125	0.03125	0.0381
Gibsonite	Sh. T.	35.00	35.00	35.00	35.00	35.00	35.00	33.00	32.00	32.00	32.00	32.00	32.00	33.58
Barytes, American ground	Sh. T.	17.50	17.50	17.50	17.50	17.50	17.50	17.50	17.50	17.50	15.75	15.75	15.75	17.06
Chrome ore	Lg. T.	17.50	17.50	17.50	17.50	17.50	17.50	17.50	17.50	17.50	17.50	17.50	17.50	17.50
Cobalt oxide	Lb.	2.50	2.50	1.45	1.45	1.45	1.45	1.45	1.45	1.45	1.40	1.40	1.40	1.61
Copperas, bulk	100 Lb.	0.55	0.55	0.55	0.55	0.55	0.55	0.55	0.55	0.55	0.55	0.55	0.55	0.55
Fluorspar, lump	Lg. T.	9.00	9.00	9.00	9.00	9.00	9.00	9.00	9.00	9.00	9.00	9.00	9.00	9.00
Fuller's earth	100 Lb.	0.800	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.70	0.80	0.80	0.80	0.80
Magnesite, Greece, crude	Lg. T.	9.00	9.00	9.00	9.00	9.00	9.00	9.00	9.00	9.00	9.00	9.00	9.00	9.00
Magnesite, cr ned	Sh. T.	35.00	35.00	35.00	35.00	35.00	35.00	30.50	30.50	30.50	30.50	30.50	30.50	32.75
Nitrate of so	100 Lb.	2.40	2.42	2.375	2.3125	2.275	2.25	2.2375	2.1875	2.1625	2.1175	2.15	2.175	2.255
Litharge	Lb.	0.0625	0.0625	0.0625	0.0625	0.0625	0.0625	0.0625	0.0625	0.060	0.060	0.060	0.060	0.0617
Red lead	Lb.	0.065	0.065	0.065	0.065	0.065	0.066	0.066	0.066	0.065	0.065	0.065	0.065	0.0652
White lead	Lb.	0.0563	0.0563	0.0563	0.055	0.055	0.055	0.0563	0.0563	0.055	0.0538	0.0538	0.0538	0.0552
Zinc white	Lb.	0.0513	0.0513	0.0513	0.0513	0.0513	0.0513	0.0513	0.0513	0.0513	0.0513	0.0513	0.0513	0.0513
Phosphate Rock:														
*Florida hard rock	Lg. T.	10.25	10.25	10.25	10.25	10.25	9.50	9.25	10.00	10.00	10.00	10.00	10.00	10.00
Florida land pebble, 68 per cent	Lg. T.	5.25	5.25	5.25	4.00	4.00	4.00	4.00	4.25	4.25	4.25	4.25	4.25	4.42
†Tennessee, 78 to 80 per cent	Lg. T.	7.00	7.00	7.00	7.00	7.00	6.00	6.00	6.00	6.00	6.00	6.00	6.00	6.42
Tennessee, 75 per cent	Lg. T.	6.25	6.25	6.25	4.50	4.50	4.50	4.50	5.00	5.00	5.00	5.00	5.00	5.15
Tennessee, 68 to 72 per cent	Lg. T.	5.25	5.25	5.25	4.00	4.00	4.00	4.00	4.00	4.00	4.00	4.00	4.00	4.31
‡South Carolina land rock	Lg. T.	6.75	6.75	6.75	6.75	6.75	6.75	6.75	6.75	6.75	6.75	6.75	6.75	6.75
South Carolina river pebble	Lg. T.	7.00	7.00	7.00	7.00	7.00	7.00	7.00	5.00	5.00	5.00	5.00	5.00	6.17
Potassium cyanide	Lb.	0.18	0.18	0.18	0.18	0.18	0.18	0.18	0.18	0.180	0.18	0.18	0.18	0.18
Kainite, bulk	Lg. T.	8.50	8.50	8.50	8.50	8.50	8.50	8.50	8.50	8.50	8.50	8.50	8.50	8.50
Pyrites:														
Dom., Non-arsenical furnace	Unit of S	0.1125	0.1125	0.1125	0.1125	0.1125	0.1125	0.1125	0.1125	0.1125	0.1125	0.1125	0.1125	0.1125
Dom., Non-arsenical fines	Unit of S	0.1025	0.1025	0.1025	0.1025	0.1025	0.1025	0.1025	0.1025	0.1025	0.1025	0.1025	0.1025	0.1025
Imp., Non-arsenical furnace	Unit of S	0.1325	0.1275	0.1275	0.1275	0.1275	0.1275	0.1275	0.1275	0.1275	0.1275	0.1250	0.1250	0.1271
Imp., Arsenical furnace	Unit of S	0.1250	0.1225	0.1225	0.1225	0.1225	0.1225	0.1225	0.1225	0.1225	0.1225	0.1200	0.1200	0.1221
Imp., Arsenical fines	Unit of S	0.0875	0.0875	0.0875	0.0875	0.0875	0.0875	0.0875	0.0875	0.0875	0.0875	0.0875	0.0875	0.0875
Imp., Non-arsenical fines	Unit of S	0.1075	0.1075	0.1075	0.1075	0.1075	0.1075	0.1075	0.1075	0.1075	0.1075	0.1075	0.1075	0.1075
Soda, caustic, 76/77	100 Lb.	1.800	1.800	1.800	1.800	1.800	1.800	1.800	1.800	1.800	1.800	1.800	1.800	1.800
Saltcake, bulk	100 Lb.	0.55	0.55	0.40	0.40	0.40	0.40	0.40	0.40	0.40	0.40	0.40	0.40	0.425
Sodium bromide	Lb.	0.16	0.16	0.16	0.16	0.16	0.16	0.16	0.16	0.15	0.135	0.135	0.135	0.153
Sodium cyanide	Lb.	0.18	0.18	0.18	0.18	0.18	0.18	0.18	0.18	0.18	0.18	0.18	0.18	0.18
Sodium sulphate	100 Lb.	0.60	0.60	0.60	0.60	0.60	0.60	0.60	0.60	0.60	0.60	0.60	0.60	0.60
Sulphur	Lg. T.	22.00	22.00	22.00	22.00	22.00	22.00	22.00	22.00	22.00	22.00	22.50	22.50	22.08
Talc, dom	Sh. T.	20.00	20.00	20.00	20.00	20.00	20.00	20.00	20.00	20.00	20.00	20.00	20.00	20.00
Zinc chloride, gran	Lb.	0.0475	0.04750	0.04750	0.0475	0.0475	0.0475	0.0475	0.0475	0.0475	0.0475	0.0475	0.0475	0.0475
Zinc dust	Lb.	0.0569	0.0569	0.0569	0.0569	0.0569	0.0625	0.050	0.050	0.0475	0.0475	0.0475	0.0475	0.0523
Zinc sulphate	Lb.	0.02375	0.0215	0.0215	0.0215	0.0215	0.0215	0.02125	0.02125	0.02125	0.02125	0.02125	0.02125	0.0216

* F.o.b. Florida or Georgia ports. † F.o.b. Mt. Pleasant. ‡ On vessel, Ashley river, S. C.

NOTE—These quotations were for ordinary wholesale lots in New York unless otherwise specified, and were generally subject to the usual trade discounts. In the cases of some of the important minerals, such as phosphate rock, pyrites and sulphur, in which there are well established markets, the quotations fully represent the latter. But in the cases of some of the minor mineral products, the quotations represent what dealers asked of consumers and not what producers realized in selling their outputs as matters of private contract.

25,442,013 long tons, value $580,077,896, to 15,784,000 long tons, value $267,540,378.

Platinum.—There was an increase from 357 oz. valued at $7,443 for 1907, to 510 oz. valued at $11,655 in 1908.

Pumice.—The 1908 production was 10,569 short tons worth $39,287; an increase of 2457 tons in quantity and $5469 in value over 1907.

Pyrite.—The United States production of pyrite in 1908 decreased to 206,471 long tons ($744,463) from 261,871 tons ($851,346) in 1907. A number of the smaller companies failed to operate. Virginia, the largest State producer, furnished 117,440 long tons worth $389,808.

Quicksilver.—Only 17,969 flasks, of 75 lb. each, valued at $805,690 were produced in the United States in 1908. As usual, California furnished the greater part of the total output. In 1907 the production was 20,932 flasks valued at $868,678.

Salt.—In 1907, 29,704,128 bbl. of 280 lb. valued at $7,439,551 were produced, while in 1908, the output was 28,745,319 bbl. valued at $7,486,894.

Silicon.—In 1908 the United States produced 600 long tons of this metal worth $72,000. Statistics for earlier years are not available, as the industry was in more or less an experimental stage.

Silver.—The 1908 production of fine silver in the United States amounted to 51,798,053 oz. ($27,382,523) against 56,514,700 oz. ($36,919,358) in 1907. The low price for silver in 1908 curtailed the output. The three largest producers among the States were Montana, Colorado and Nevada, in the order named.

Sulphur.—The 1908 domestic production was about the same as that in 1907, viz: 307,750 long tons ($6,795,120) against 307,806 tons ($6,427,025). Louisiana as usual produced nearly all of this. Utah and Wyoming also contributed to the total.

Sodium.—The production in 1908 was about the same as that for the previous year, viz., 2000 short tons valued at $1,000,000.

Talc.—New York produced 70,739 short tons ($697,390) of fibrous talc in 1908, against 59,000 tons ($501,500) in 1907. Virginia produced 18,623 short tons ($335,068) of talc and soapstone.

Tin Ore.—Only 50 tons of tin ore valued at $12,500 were produced in the United States in 1908. In 1907 the output was 63 tons valued at $15,209.

Tungsten Ore.—The production of this mineral fell off heavily in 1908, amounting to only 497 short tons ($126,281) against 1468 short tons ($715,031) in 1907. Practically all came from Colorado.

Vanadium.—The output of this metal in the United States increased in 1908; the production was 64,800 lb. ($162,000), while in 1907 it amounted to 41,000 lb. ($102,500).

Zinc.—The output of virgin spelter in 1908 was 210,511 short tons, against 249,612 short tons in 1907. The 1908 production was valued at only $19,897,500, while that of 1907 was stated as worth $29,763,735.

Zinc Ore.—The 1908 production of zinc ore in the United States was 838,377 short tons against 902,923 short tons in 1907.

Zinc Oxide.—The total 1908 production of zinc oxide (including a small quantity made from spelter) was 65,100 short tons worth $5,876,342 as comapred to 85,390 short tons worth $7,731,100 in 1907.

METALLURGICAL PRODUCTION—PRICES—CONSUMPTION.

The statistics presented in the preceding tables and summary cover, with but few exceptions, only the production of metals from domestic ore. In addition thereto the United States produces a large amount of several metals derived from foreign ores, especially copper, lead, spelter, nickel, antimony, and the three precious metals—gold, silver, and platinum. Aside from gold and silver, data as to the total metallurgical production will be found in the following pages.

In the following pages also will be found much data as to the domestic consumption of metals and mineral substances. In general this figure is computed from the production plus imports, less exports; but in some important cases it has been possible to take into account the stocks on hand at the beginning and end of the year. In the cases of lead and spelter a statistical investigation, covering a series of years, has been made on the basis of reports received directly from the consumers.

In each chapter of this book all available information as to market conditions and prices has been given. For all of the important metals averages, monthly or annual, are given for a long series of years. Unfortunately, such data are not available for all of the mineral substances. The monthly averages for many of these substances in 1908, not including the metals, are given in the table on a previous page.

ALUMINUM.[1]

Aluminum in 1908 became for the first time a competitive metal; not that it was not previously an article of large trade, but its price was controlled by agreement and the metal was not subject to the fluctuating conditions of open buying and selling like copper, lead, tin, and the others. The history of aluminum in 1908 is in some respects spectacular. In 1906-07 aluminum was almost unobtainable. At one time the Aluminum Company of America had its wire-drawing capacity contracted for several months in advance. With the industrial setback in the last quarter of 1907, the demand for aluminum fell off along with that for the other metals. This led the producers in Europe, still bound together by the terms of a convention, to reduce their price from 33c. per lb. to 22c. At about the same time the American producer reduced its price from 38c. to 33c. Previous to the panic in 1907, the American price was 42c.

Even as early as the first quarter of 1908, foreign producers began to offer ingot metal in the United States, in spite of the duty of 8c. per lb. These offers did not, however, lead to any significant business; indeed, they did not become energetic until nearly the end of 1908.

PRODUCTION, IMPORTS AND CONSUMPTION OF ALUMINUM IN THE UNITED STATES.

Year.	Production.			Imports.			Exports.	Consumption
				Crude.		Mfrs.		
	Pounds.	Value.	Per lb.	Pounds.	Value.	Value.	Value	Value
1897	4,000,000	$1,400,000	$0.35	1,822	$1,082	$3,647	(a)	$1,404,729
1898	5,200,000	1,690,000	0.33	60	30	13,840	$238,997	1,474,268
1899	6,500,000	2,112,500	0.33	53,622	9,425	7,828	291,515	1,838,238
1900	7,150,000	2,288,000	0.32	256,559	44,455	5,989	281,821	2,056,623
1901	7,150,000	2,238,000	0.31	564,803	104,168	5,580	183,579	2,164,169
1902	7,300,000	2,284,590	0.31	745,217	215,032	3,819	116,052	2,387,389
1903	7,500,000	2,325,000	0.31	498,655	139,298	4,273	157,187	2,311,384
1904	7,700,000	2,233,000	0.29	515,416	128,350	478	166,876	2,494,952
1905	11,350,000	3,632,000	0.32	530,429	106,108	33	290,777	3,015,364
1906	14,350,000	5,166,000	0.36	770,713	154,292	1,866	364,251	4,957,907
1907	26,000,000	10,920,000	0.42	872,474	181,351	1,124	304,938	(b)
1908	13,000,000	4,095,000	0.315	465,317	80,268	2,334	330,092	(b)

(a) Not reported. (b) Impossible to compute accurately in the absence of information as to unsold stocks.

[1] Under this caption should be treated, logically, the ores of aluminum—bauxite, corundum, emery, and the artificial product, alundum, and the sulphates—alum and aluminum sulphate. But in this volume we continue to conform to previous custom, and reports on those substances will be found under the respective captions.

PRICE OF ALUMINUM AT NEW YORK.
(In cents per pound.)

Grade.	Dec., 1905.	July, 1906.	Dec., 1906.	July, 1907.	Dec., 1907.	July, 1908.	Dec., 1908.
99% pure	35	36	38	42	33	28	24
90% pure	33	34	37	41	32	27	23
No. 12 casting alloy	35	36	37	41	32	27	22
No. 21 casting alloy	33	34	35½	39½	30½	24½	22½
No. 31 casting alloy	30	31	33½	37½	28½	22½	21½

The above prices were for ton lots or over; the prices for small lots were 2 to 3c. per lb. higher. The average price for No. 1 ingot, 99 per cent. pure, in 1908 was 31.5 c. per lb.

WORLD'S PRODUCTION OF ALUMINUM.
(In metric tons.)

Year.	Great Britain.	France.	Switzerland Germany. Austria.	North America.	Totals.
1897	(a)310	470	800	1,815	3,195
1898	310	565	810	2,359	4,034
1899	559	763	1,300	2,949	5,571
1900	569	1,026	2,500	3,244	7,339
1901	560	1,200	2,500	3,244	7,504
1902	600	1,355	2,500	3,312	7,767
1903	(b)650	1,570	(b)2,500	3,403	8,123
1904	(b)650	1,650	(b)3,000	3,494	8,794
1905	2,250	4,425	3,675	6,560	16,810
1906	2,500	4,500	4,000	7,325	18,325
1907	3,700	4,700	8,000	16,329	32,529
1908	.....	.....	.....	8,150	.....

(a) C. L. Neve Foster, British Mineral Statistics for 1897.
(b) Statistics of Metallgesellschaft, Frankfurt am Main.

Market Conditions.—It has been the policy of the Aluminum Company of America to make the price for its product all that the trade would bear. This has excited more or less adverse criticism and also has aroused dissatisfaction and dislike on the part of the consumers, so that the latter have been looking anxiously forward to the time when competition might develop after the expiry of the patent monopoly. From the standpoint of the producer the policy was probably sufficiently wise. Consumers were willing to pay a price yielding 150 to 200 per cent. profit for all the metal that the company could produce; indeed the company was strained to the utmost to increase its capacity; consequently why not make hay while the sun was shining?

If the Aluminum Company is open to any criticism, it is for failing to bow more quickly to adverse conditions. Toward the end of 1907, demand having dwindled to a comparatively small rate, the company made a rather grudging concession in price and shut down two-thirds of its smelting capacity, rather than attempt to revive consumption by a sweeping reduction in price. At this time the management was imbued with the well-known policy of the U. S. Steel Corporation, which Judge Gary expounded on several occasions, that the cutting of prices does not stimulate consumption, wherefore prices are best maintained. Thus the

Aluminum Company of America maintained its price steadily at 33c. until the threatening of foreign competition forced it to reduce.

On Sept. 30, 1908, the convention among the European producers was dissolved, and immediately there was a scramble for business in a market which had suffered severely by the enormous falling off in orders from automobile manufacturers, and from competition of ferrosilicon in the iron trade. The price in Europe fell quickly to as low as 13½c. per lb., and at the end of 1908 offers to sell were made on terms that could net hardly more than 12½c. f.o.b. works. The American producer is protected by the duty of 8c. per lb. upon ingot aluminum, but in spite of that preposterous duty, the slump in Europe was so great as to compel successive reductions in price here. The last quotation was 24c. per lb. nominally, but probably considerable discounts from that price are made to well-informed customers, inasmuch as high-grade European aluminum was freely offered here at the close of 1908 as low as 21½c., and some business was done at that figure.

Cost of Production.—The present price puts aluminum on a basis comparable with those of tin and copper. Weight for weight it is cheaper than tin, but bulk for bulk it is cheaper than either tin or copper. This level of price ought to stimulate greatly the consumption of aluminum, especially when the revival in general trade goes a little further. We look upon aluminum as having now fallen to somewhere near its natural level, around which the price will hover, rather than as having suffered a slump to a very low price from which there will be a recovery. The producing capacity of Europe is increasing, while the expiration of the Bradley patent in 1909 will permit free competition in the United States just as there is now in Europe. Even at present prices there is a good profit to the Aluminum Company of America, the manufacturing cost of which, according to excellent authority, is in the neighborhood of 15c. per lb. (exclusive of amortization charges). According to a recent writer in the Frankfurter *Zeitung* the average cost of production in Europe is 17¼c. per lb. (higher than the present selling price). According to *Electrochem. and Met. Ind.* (Dec. 1908) the estimated cost in a proposed French mountain plant (which was not erected but was figured out with the greatest care) was slightly less than 16c. per lb., the cost of electric power in this case being about $8 per horse-power year. Against these estimates stands the fact that aluminum has been sold by European makers at close to 12½c. per lb., implying either an enormous loss or a direct-manufacturing cost of less than 16c. The estimate of that figure apparently includes a proper allowance for amortization, which of course manufacturers are in the habit of postponing in times of stress. Undoubtedly, also, there are important differences in the positions of the

various European smelters, and the market price is established by the most favorably situated under the present condition of over-supply.

Prospects for Competition.—A reason why the importation of foreign aluminum failed to amount to much when it was offered here at less than the price of the American producer was the failure of the brokers making the offers to bring hither a stock of the metal from which prompt delivery could be made. By the end of 1908 this was done. However, there was a disinclination to make any large commitment in that way inasmuch as it was realized that the American producer, protected by the enormous duty of 8c. per lb., could drop the price below that at which the foreign metal could be offered and still could realize a profit for itself. This condition stands in the way of any extensive importation from abroad.

More likely is the development of competition on this side, the last of the protecting patents having expired in February, 1909. Entry into the business is already being considered seriously by at least one powerful metallurgical interest. However, such competition will doubtless be delayed by the existing scarcity of American metallurgical engineers acquainted with the art of aluminum smelting, although French and German engineers will be available for the purpose.

New Works in 1908.—Among the new concerns that came into existence under the ægis of the convention, and are now competing in the open market, are the Société Electro-Métallurgique Française (Froges), whose works at St. Michael (Savoy) dispose of 12,000 h.p. and turn out 8000 to 10,000 tons per annum: the Compagnie de Produits Chimiques d'Alais et de la Cammargue, with works (10,000 h.p.) at St. Jean (Savoy); the Société Electrochimique de Paris, with works (10,000 h.p.) at Bourg; and the Società Italiana per la Fabbricazzione dell' Aluminio, whose works (also of 10,000 h.p.) are at Bussi.

Views of the Aluminum Company of America.—The following statement (from which we have excised certain obviously disingenuous arguments) by the Aluminum Company of America to the Ways and Means Committee, in December, 1908, in support of its contention that the American duty of 8c. per lb. on ingot should be retained is important:

"The largest use for aluminum abroad is for military equipment for infantry. The German, French and other foreign governments buy large quantities of aluminum, and thus give to the aluminum works situated within their borders a large and permanent trade which cannot be taken away, while necessarily in the United States the field has only been in general consumption—in domestic utensils, wire for electrical conductors, etc.—and to create this market is a very considerable source of expense to the United States producer.

"During 1905, 1906 and the first part of 1907 the foreign producing capacity was no more than equal to the foreign consumption, and, in fact, for a short period was so much less that an acute shortage existed, accompanied by a rise in price. This condition caused the five then existing foreign companies to increase their plants and induced seven new companies to erect works, so that now the foreign aluminum producing capacity is at least six times the foreign consumption of even 1906-07. All the foreign companies thus not only have immense surplus producing capacity, but have large surplus stocks of aluminum. This condition is not a temporary result of the present commercial depression, but is a condition which will continue for many years to come, and is compelling all the foreign companies to turn to the United States market, as is shown by the constantly increasing importations and by the fact that the domestic producer is being met on every sale by a flood of foreign quotations at constantly lowering prices.

"The aluminum business lends itself with unusual ease to the disposal of excess product. From behind their prohibitive tariff wall of more than double the United States tariff, foreign aluminum companies can and do quote their surplus stock, delivered in New York, at prices substantially less than the cost of production in this country, and which, even with the United States duty added, compete with the domestic prices that give only a fair profit to the domestic producer.

"Foreign aluminum producers sell for export at 12@13c. per lb. f.o.b. New York. The domestic price of aluminum is 20@22c. per lb., which is as low as aluminum can be manufactured and sold under American conditions of manufacture.

"Trimmings from aluminum sheet and old sheet, wire, castings, etc., are easily remelted and run into ingots. A large number of metal merchants abroad make a practice of getting together this junk and scrap aluminum, remelting it, and offering it on this market as new aluminum. As is the case with other metals, such remelted ingots are, for certain purposes, nearly as good as new.

"The aluminum business has been only ordinarily profitable. Dividends have been paid by the Aluminum Company of America in but 11 years out of 20, and the average dividend for the 11 years has been 8 per cent. on a very conservative cash capital, making an average return to the stockholder of less than 4½ per cent. During 1908 no dividend has been paid. The position of the Aluminum Company of America is due not to combination, control of raw material, etc., but to meritorious patents, the chief of which have been taken out by its officials and have been adjudicated by the United States courts. These patents have in the main expired and will all expire within a few months."

According to Arthur V. Davis, of the Aluminum Company of America, in testimony before the Ways and Means Committee, Nov. 27, 1908, during the last four or five years the producing capacity of aluminum abroad was confined to five companies, and these companies were hardly able to supply the demand outside of the United States; but in the fall of 1907 and in the spring of 1908, seven new companies started almost simultaneously, and at the same time the five old companies finished large additions to their plants. The consumption of aluminum abroad in 1907 was about 7000 tons.[1] In 1908 it was substantially less on account of the depression. "The manufacturing capacity by means of these additions and these new plants is about six times the normal consumption of the Continent, and the foreign companies are looking to this market to get rid of their surplus product, and so far as we can see there will be no let-up to their invasion; and every sale that we make, every quotation that we make, is now made in direct competition with the foreign producers."

Mr. Davis continued as follows: "Labor costs in the manufacture of aluminum are relatively greater than in any other metal. Our ore (bauxite) exists in Georgia, Alabama, Tennessee, and Arkansas. We have to ship this bauxite to our water-power plant, an average distance of 1300 miles. In France, where they have better bauxite than we have, i.e., containing more alumina, they have only 150 miles to ship. We ship the crude ore from our mines to East St. Louis, Ill. At East St. Louis we manufacture this material into what is known as alumina. It takes a ton of bauxite, a ton of coal, and a ton of limestone to make half a ton of alumina, and we had therefore to select some central point where we could find coal, limestone, and bauxite at the cheapest cost. The limestone is required to be of very high quality. It must be 99 per cent. $CaCO_3$. In France they pay for furnace men, who constitute the largest part of the labor, from 4 francs (80c.) a day up to $1 (5 francs) a day, and the men work 12 hours. We have to pay an average of $2 per day for our furnace men, and our men work only eight hours a day. In the aluminum business the administrative charges, the overhead charges, are very high. Our overhead charges run 3 or 4c. per lb., without counting depreciation or counting interest on our investment, but simply our bonded indebtedness and our administrative charges. The French manufacturers are in the position of being able to throw aluminum into this country without regard to overhead charges, on the general principle that whatever they can sell over and above their home consumption they can sell at shop cost; and aluminum is being quoted here much below our cost price f.o.b. New York. The freight from France to this country,

[1] If this figure be correct there must have been a large accumulation of stock at the beginning of 1908.—*Editor.*

or from any foreign country to this country, is 24c. per 100 lb., which is practically negligible as compared with the price of aluminum."

Uses of Aluminum.—The various uses of aluminum have been along lines considerably different from those generally credited to it. Steel manufacturers are extensive consumers, using it as a deoxidizer in their converting departments. A large manufacturer of structural shapes and plates is said to take not less than 50 tons per month for this purpose. Where weight is a factor in the construction of machinery, such as automobiles and motor boats, aluminum, both as sheet and as castings, and in the form of an alloy, is largely used. The difficulty of soldering aluminum sheets has heretofore prevented its general adoption for a wide range of usage, but with more general experimenting it is reasonable to suppose that some better method of fastening pieces together than riveting will be found. Although the electrical conductivity of aluminum is high it has not yet been extensively used as a substitute for copper in that field. One reason, perhaps, is that there has not been enough inducement in cost. In California, however, it has been used more than in any other section, largely because of favorable climatic conditions and of the long distance lines, one being 180 miles long. This use as well as the use in alloys may reasonably be expected to increase. Indeed, at the end of 1908, the drawers of copper wire found that they had to face serious competition from aluminum, and some important contracts were awarded in favor of the latter metal. Aluminum electrical conductors have now been in successful use for 10 years under favorable conditions, but there have been failures under severe conditions, as in Utah.

THE METALLURGY AND USES OF ALUMINIUM IN 1908.[1]

BY JOSEPH W. RICHARDS.

In its brief life of 50 years, since the first industrial plant was established, the aluminium business has had many alternations of quick expansion and level running, probably each more remarkable than the preceding. Henri St. Claire Deville started his commercial plant in 1859, with the problem confronting him of creating the market for his product. For 25 years the price remained steady at about $10 per lb., and the yearly output about two to three tons. Then came successively the Grätzel electrolytic process, the Castner sodium improved method, and the Hall process, which together in five years reduced the price to $1.50 per lb. and increased the output by leaps and bounds. For just 20 years the Hall process in America and the similar Héroult process abroad have held the field against all comers, the supply alternately exceeding and then

[1]The spelling "aluminium" has been retained in this article in deference to the well known wish of Professor Richards. Elsewhere in this volume the conventional American form of "aluminum" is employed.—*Editor.*

falling behind the demand, and the price going down to the present value of 15c. per lb. abroad (22c. in the United States). From a chemical curiosity in Wöhler's time to a commercial novelty in Deville's, from a semi-rare metal by the Castner process to a cheap, every-day metal by the Hall process, have been the successive changes, and all this has happened within 50 years. Some years ago, I said that the commercializing of the new metal aluminium, with its unique properties, was one of the half dozen great metallurgical achievements of the 19th century. This statement appears even more true today.

A shortage in the supply of aluminium creates much more disturbance in the metallurgical world than a falling off or a lack of demand. A thousand people, or perhaps ten thousand, use the metal, while only one firm produces it in America, and not a half-dozen abroad. Consumers who cannot get all the metal they want to order, immediately complain long and loud of the uncertain supply, and increase each other's dissatisfaction by public protestations. On the other hand, a shortage in demand, with stocks piling up in the works and workmen turned off, inconveniences but a very few firms, and, for reasons of policy, they prefer to keep quiet rather than to become calamity-howlers. The latter condition prevailed in 1908, about as acutely as the former condition had prevailed in 1907. In consequence of the general increase of consumption in 1907, the manufacturers spent immense sums in increasing their capacity in 1908. This largely increased capacity, amounting in America to an increase of between four and five fold over 1907, coincided with a sharp business depression, which caused orders to fall off greatly and prevented the purchase of the usual stocks in advance. The combination was most unfortunate for the manufacturers; with 350 per cent. increased capacity they sold less metal than in the previous year, and therefore had to stand heavy financial losses.

With the present productive capacity, which is estimated to be 50,000 tons per year in America and an equal amount in Europe, and with an estimated total consumption in 1908 of 12,000 tons, it will probably be some years before the demand catches up to the possible supply. Yet, I am not very positive as to this. The reduction of price at the end of 1908 has been so great, especially in Europe, that great demands may arise from new applications of aluminium which this low price makes possible. At this price, copper is out of the race as a material for electric transmission lines, brass is dearer than aluminium for most useful purposes, and even zinc commences to feel the neighborhood of its youthful competitor, while nickel and tin are hopelessly distanced.

Present conditions are wonderfully favorable for a tremendous development of the old and also of new uses of aluminium—the development

which many of us have looked forward to for a decade or more. One hundred thousand tons a year may not be too great a supply at the close of another year, with aluminium cooking utensils as cheap as enamel ware, aluminium electric conductors cheaper than copper, and aluminium articles of all descriptions as cheap as those of brass.

As far as developments inside the producing industry, in 1908, there are none to record for the United States other than those already almost finished at the beginning of 1908, and recorded in THE MINERAL INDUSTRY, Vol. XVI. The Aluminium Company of America is, furthermore, planning no further developments for 1909.

In Europe, the British Aluminium Company expected its 1908 output to double that of 1907, but this expectation was disappointed, as previously stated. The plans of this company are for ultimately operating 60,000 h.p., and the development of its great Loch Leven scheme in Scotland, has not been interrupted. In March, 1908, the Aluminium Corporation, Ltd., of Newcastle on Tyne, started production of commercial aluminium; its plant also includes three carbon baking furnaces for making electrodes. The plant of the same company at Dolgorrog, near Conway, North Wales, was started in September. It has a yearly capacity of 1500 tons, containing 25 reduction pots in series run by a current of 7500 amperes at 120 to 130 volts. The same company also owns bauxite mines in southern France. The new works in Italy, at Bussi, near Rome, were operated, but at a financial loss, which was made up by concessions as to the cost of power from the other electrochemical enterprises of this company.

Methods of Reduction.—As is generally known, the Hall patents for the composition of his electrolyte expired three years ago, and the Bradley patents for the working with electrically generated internal heat expired in February, 1909. This leaves the broad principles of the process, as it has been practised for 20 years, open to the public. This fact, however, has not discouraged inventors from attempting other methods of reduction which might prove cheaper than the present one. These attempts have been mainly along the lines of direct electric-furnace reduction of alumina by carbon. Mr. Tone's success in thus reducing silica to silicon has inspired several attempts to solve the question of producing aluminium in this manner. (Incidentally, it is worth remarking that silicon produced by such direct reduction is selling at present—90 per cent. pure—at less than 5c. per lb.) On the investigation side, the experiments of Dr. R. S. Hutton and J. E. Petavel, at the Electrochemical Laboratory of the University of Manchester, have been made in an enclosed electric furnace working under high pressure, up to 200 atmospheres. (Paper in *Philosophical Transactions*, Jan. 17, 1908; abstract in

Electrochem. and Met. Ind., March, 1908.) In treating a mixture of alumina and carbon by resistance heating, no aluminium was produced, nor was aluminium carbide, even, formed. Treating it by arc heating, reduction took place, aluminium carbide being the chief product, but some malleable metal was also obtained. The experimenters conclude also that under atmospheric pressure the temperature of reduction of alumina is above the boiling point of aluminium. The minimum temperature of reduction under pressure is the melting point of alumina. They conclude by stating that "the necessary conditions for the successful direct reduction of alumina by carbon seem to be fairly well defined, the outstanding problem being chiefly a matter of the arrangement and construction of the furnace."

E. Viel, in U. S. patent, 883,594, of March 31, 1908, follows the same lines, proposing the reduction of alumina mixed with carbon by an operation similar in principle to Tone's electric-furnace reduction of silica. The details of his specification show lack of understanding of the temperatures involved, and his directions for handling the reduced metal or alloy are somewhat visionary.

In the matter of preparing a pure alumina for reduction, F. J. Tone, superintendent of the Carborundum Company's works, and inventor of the silicon furnace, has taken out a group of patents (U. S. patents 906,172; 906,173; 906,338; 906,339; of Dec. 8, 1908) having for their object the production of silicon from kaolin, with the production of pure alumina as a by-product. By mixing the clay with sufficient carbon to form carbon monoxide with the oxygen of the silica, the latter is reduced to silicon, also any iron oxide to iron, and titanium oxide to titanium, while the alumina remains behind unreduced and purified. If iron is mixed with the charge, ferro-silicon is obtained. These reactions are:

$$Al_2O_3,2SiO_2+4C=Al_2O_3+2Si+4CO.$$

$$Al_2O_3,2SiO_2+4C+Fe=Al_2O_3+FeSi_2+4CO.$$

If an insufficient amount of reduction carbon is used, some silica remains unreduced. If one-fourth the silica in the clay remains unreduced, a residue containing approximately one-fourth silica and three-fourths alumina remains, which is a new and homogeneous material of great hardness and valuable abrasive and refractory qualities; it is as hard as some corundum, and very tough. Varieties of this material may be produced containing 5 to 35 per cent. of silica. A typical reaction for this purpose is: $2(Al_2O_3,SiO_2)+6C=2Al_2O_3+SiO_2+3Si+6CO$.

Welding Aluminium.—M. U. Schoop has organized the Aktiengesellschaft für Autogene Aluminium-Schweissung in Zurich, and has patented

his method in many countries. The oxygen-illuminating gas blowpipe is used, and the essentials of success are practice and the use of the special flux, which dissolves aluminium oxide fluently and makes the operation easy and effective. I visited the plant in Zurich during 1908, and was surprised at the skill attained in applying the process. The joints to be united are sprinkled with the flux and heated just to melting by the blowpipe flame, working at a speed of about one meter per minute on thin metal to one-fourth that speed on heavy sheet. Boilers, vats, etc., of the most complicated shape are thus made practically into one piece. Microscopic, chemical and mechanical tests of the seams show an absolute homogeneity, strength equal to the adjacent metal, and unimpaired malleability.

Aluminium Coins.—It is officially announced that the French Government will coin 5 and 10 centime (one and two cent) pieces of aluminium, to replace the unsightly and obnoxious coppers. Even the number of pieces to be coined is stated: 562,500,000 of the smaller and 348,750,000 of the larger pieces. They will weigh respectively, 1½ and 3 grams instead of 5 and 10 grams. I advocated this substitution in a series of articles in the *Aluminium World* in January-March, 1895, which were written at a time when the U. S. Government was experimenting in this line. Doubt was then, and has again been, raised as to the durability of aluminium coins, and to test the matter practically I have carried a small aluminium medal in my purse with other loose coins ever since 1895. The wear has been about the average wear observable on nickel coins which have been in circulation that length of time.

Aluminium Paper.—A substitute for tin foil is thus described in the *World's Paper Trade Review,* Sept., 1907, p. 457: The Wickel process is successfully operated in France. The paper is coated with varnish, dusted evenly with finely powdered aluminium, excess removed, and the paper then calendered, embossed, stamped, etc. It gives a metallic paper absolutely free from arsenic or poisonous metals.

Explosions of Powdered Aluminium.—M. M. Richter (*Chem. Zeit.,* 1908, p. 136) states that powdered aluminium, becoming quickly coated with an insulating layer of oxide, does not conduct electricity appreciably, and that for this reason the stream of aluminium powder leaving the machine in a polishing mill does not conduct away the frictional electricity produced. Potential differences up to 3000 volts are in this manner rendered possible in the polishing mills, and explosions of the dust are known to take place from this cause; in one case an explosion occurred one minute and a half after a new plant was started in operation.

Aluminothermy.—The use of aluminium for reducing other metals, the Goldschmidt process, received much attention in 1908, particularly in

the line of using alloys of aluminium with other metals as the reducing agent. The manufacture of such alloys for this purpose has also received special attention. M. Meslans (U. S. Patent, 875,668, Dec. 31, 1907) electrolyzes fused calcium chloride, using melted aluminium beneath as cathode, and produces alloys with any desired proportion of calcium. Alloys with barium, strontium and lithium are similarly produced. These alloys give high thermic reduction effects, since the calcium, etc., has even greater affinity for oxygen than aluminium has, and the mixed calcium-aluminium oxides formed unite to a fusible aluminate slag which is thinly liquid at the temperatures obtained. In the iron welding process, an amount of iron turnings equal to one-third the weight of the mixture may be added and melted by the excess heat at command. The best mixture is obtained by using an alloy represented by the formula Ca_3Al_4. The heat effect of this alloy is higher than pure aluminium, and it works particularly satisfactorily in the reduction of metallic chromium from chromic oxide. The alloy $Ca_3Al_4Si_3$ is also practicable, the silicon cheapening the mixture, but not lowering its thermic effect below the necessary point. This alloy forms an alumino-silicate slag. The very low price of metallic silicon (7c. lb., December, 1908, in Germany) makes this modification of commercial advantage and importance.

K. A. Kuehne (U. S. Patent, 878,210, Feb. 4, 1908) proposes to use chlorates, perchlorates or peroxides in the alumino thermic mixture, to increase the intensity of the temperature obtained. The idea appears questionable; I have not heard of its industrial use.

Von Kügelgen and Seward (U. S. Patent, 881,934; March 17, 1908) produce magnesium-aluminium alloy for metallo-thermic purposes by a process similar to Meslans', using melted aluminium as a cathode, but an electrolytic bath above consisting of magnesia dissolved in magnesium-alkaline double fluoride, such as magnesium-lithium fluoride.

Weston and Ellis (*Trans.* Faraday Society, 1908, p. 60) find that finely powdered aluminium mixed with pure carbon can be ignited by a fuse of barium peroxide and magnesium ribbon, reacting with vivid incandescence. The product consisted principally of aluminium carbide, Al_4C_3, mixed with nitride and oxide. The air in the crucible really starts the reaction, the oxidation raising the temperature so high that the carbide and nitride can form.

The same investigators found that powdered aluminium reduces boron tri-oxide and silica, forming aluminium borides and amorphous silicon.

A reaction which may possibly throw some light on matte composition is that studied by Hugo Ditz (*Metallurgie*, 1907, p. 786). He ignited powdered aluminium with iron pyrites, in the metallo-thermic way. 3 FeS_2 with 4 Al gave less than half the iron in the metallic state, the

rest remaining probably as Al_2S_3. FeS. The same product results from action on FeS, the reaction being

$$4FeS+2Al=Al_2S_3. FeS+3Fe.$$

Miscellaneous Uses.—The early engineering difficulties met in using aluminium for low and high tension electrical conductors have been overcome, and its utility in this field is now beyond question, as is also its future use on a very large scale. When properly constructed, aluminium cable lines have proved eminently satisfactory in all parts of the United States. For several years, automobile castings of aluminium for engine bed, gear case, rear axle housing and numerous other parts have been in successful use. The present low price has stimulated its use for other castings, and also, as sheet, for the bodies. Sheet aluminium is well adapted for automobile and carriage bodies, as it holds paint quite as effectively as wood and has the metallic characteristics of non-warping, etc. The main parts of many electrical and other physical instruments are now made of cast or sheet aluminium; such as meter covers, frames, dials, indicating needles. In color printing and lithography, the use of aluminium plates in place of zinc plates and lithographic stones, is successful and increasing. In chemical works, manufacturers are finding increasing use for aluminium. Pipes for conveying sulphurous acid gas or acid liquor to or from the digestors are practicable and durable, not being attacked as much as brass or iron, and not sagging with changing temperature as does lead pipe. It is also useful in nitric acid condensers and acetic acid stills.

Aluminium tubing has been found to resist very well in condensers for concentrating ammonia vapors from by-product coke ovens, being practically unattacked by either sulphur compounds or ammonia. Aluminium-lined copper pipe is also suitable for surface condensers. In the rubber industry, it is used for mandrels, rubber-shoe lasts and in manufacturing tubing. Fruit jar tops are made in very large quantities of aluminium, since it is practically unattacked by fruit acids and juices. Even if attacked to a minute extent, the compounds formed are not poisonous, as would be the case with zinc or tin. Aluminium steam-jacketed kettles are used in preserving fruit, and in manufacturing of food products. Sterilizing and pasteurizing apparatus for preparing sanitary food products are best made of aluminium. Steam-jacketed kettles and aluminium tubing find extensive use in the stearic acid and candle industry. Their particular advantage here is almost entire freedom from corrosion and no discoloration of the products. Tubing for conveying glycerine, obtained as a by-product from soap manufacture, has similar advantages.

Mr. E. Blough has a review of the industrial uses of aluminium in *Trans.* American Electrochemical Society, XIII (1908).

Alloys.—Alfred G. C. Gwyer (*Zeit. f. Anorg. Chem.*, 1908, p. 113) writes a 40-page article on the scientific study of alloys of aluminium with copper, iron, nickel and cobalt. He found lead and cadmium not to alloy with it. By Tammann's method of thermal analysis he has determined the existence of $CuAl_2$, CuAl, Cu_3Al; $FeAl_3$; $NiAl_3$, $NiAl_2$, NiAl; Co_3Al_{13}, Co_2Al_5, CoAl. Of the iron alloys, only those with over 70 per cent. iron are magnetic; of the nickel alloys, those with over 85 per cent. nickel; with cobalt alloys those with more than 68.5 per cent. cobalt.

L. Donski (ibid., 1908, p. 185) describes the alloys of calcium and aluminium. With up to 8 per cent. calcium, the color of aluminium is unchanged and hardness is increased; with more calcium the alloys are brittle and porous; with high calcium they are again tough, but are unstable in the air. One compound occurs, $CaAl_3$, and two eutectics, at 610 deg. C. with 8 per cent. calcium, and at 550 deg. with 75 per cent. calcium.

G. Hindrichs (ibid., 1908, p. 414) finds manganese and aluminium to form two definite compounds, Mn_3Al and $MnAl_3$.

Walter Fraenkel (ibid., 1908, p. 154) examined the aluminium-silicon alloys. At 10 per cent. silicon is found a eutectic, freezing at 576 deg. C. The elements are miscible in all proportions and do not chemically combine.

A. V. Sapeschinkoff (*Jour.* Russian Phys. and Chem. Soc., 1908, XCV, 665) investigates the hardness of aluminium-zinc and aluminium-tin alloys. The hardest alloys contain 30 per cent. of zinc and 40 per cent. of tin.

W. Grossman (U. S. Patent, 886,579, May 5, 1908) makes a casting alloy of 7 to 8.5 per cent. copper, 4.5 to 5.5 per cent. tin, and the rest aluminium. The castings are said to be homogeneous and very rigid.

D. P. Smith has studied the alloys with potassium. These two metals do not combine chemically, nor do they mutually dissolve each other in the liquid state or as solid solution.

Alloys with Copper.—F. G. A. Wilm (*Brass World*, November, 1907) claims that aluminium hardened by copper is improved by chilling it from temperatures near its melting point. For alloys with different contents of copper, the minimum temperatures from which they should be chilled are as follows: 1 per cent. copper, 435 deg. C.; 2 per cent., 471 deg. C.; 3 per cent., 482 deg. C.; 4 per cent., 494 deg. C.; 6 per cent., 500 deg. C.; 10 per cent., 505 deg. C.; 15 per cent., 508 deg. C. For example, the 4 per cent. copper alloy is chilled from a temperature above

494 deg. C. It is said to have its tensile strength thus increased nearly one-half, without reducing its ductility.

Aluminium Bronzes.—These bronzes have been studied assiduously by the British National Physical Laboratory. Tests of the effect of temperature on tensile strength of 10 per cent. bronze show strength fairly well maintained up to 350 deg. C. and decreasing rapidly above 400 deg. C. 6.7 per cent. bronze, however, decreased regularly in strength with increasing temperature, and fell off more rapidly above 400 deg. C. A great deal of investigation was put upon the effect of small quantities of manganese upon these bronzes, without reaching conclusive results because of impurities in the manganese used. The indications are that the physical character of the bronzes can be decidedly improved by such additions.

Manganese Alloys.—The British National Physical Laboratory has investigated the aluminium-manganese alloys, finding some unexpected properties. The alloys rich in manganese have a peculiar tendency to attack porcelain and fire-clay. Alloys with 30 per cent. and also with 65 per cent. of manganese disintegrate spontaneously and rapidly to fine crystalline powder.

ALUMINUM SULPHATE AND ALUM.

Aluminum sulphate occurs in nature as the mineral alunite. This occurs in connection with the gold ores at Goldfield, Nev., and it has been proposed to leach the salt from the waste-dumps at that place, but this has not yet been attempted. Immense deposits of alum rock also exist along the Gila river in Grant county, N. M., that it has also been proposed to utilize commercially.[1] However, these western occurrences are probably too remote to be of value at the present time. The market for aluminum sulphate is chiefly in the East and a price of more than $20 per 2000 lb. can not safely be estimated, while the freight rate to market would hardly be less than $15 per ton, leaving too small a margin for mining, manufacture, and packing.

PRODUCTION AND IMPORTS OF ALUM AND ALUMINUM SULPHATE INTO THE UNITED STATES. (b)
(In tons of 2000 lb.)

Year.	Production.						Imports. (a)	
	Alum.			Aluminum Sulphate.			Quantity.	Value.
	Quantity.	Value.	Per ton.	Quantity.	Value.	Per ton.		
1903.....	7,574	$210,910	$27.85	80,726	$1,614,520	$20.00	2,162	$107,948
1904.....	11,563	319,189	27.60	74,481	1,417,867	19.04	896	19,991
1905.....	10,114	289,716	28.65	93,917	1,660,515	17.67	1,282	26,242
1906.....	15,613	450,125	28.83	89,246	1,613,050	18.07	1,183	23,193
1907.....	10,404	361,900	34.78	106,821	2,008,046	18.80	1,562	35,191
1908.....	7,700	236,710	30.74	97,255	1,835,213	18.87	1,407	24,929

(a) Includes alumina, aluminum hydrate or refined bauxite, alum, alum cake, aluminum sulphate, aluminous cake, and alum in crystals or ground. (b) As reported by the U. S. Geological Survey.

The production of alum and aluminum sulphate in the United States, as well as elsewhere, is made from bauxite. The chemistry of the manufacture was fully described in THE MINERAL INDUSTRY, Vols. III and VII. The American producers are the following:

General Chemical Co., 25 Broad Street, New York.
Pennsylvania Salt Manufacturing Co., Philadelphia, Penn.
Harrison Brothers, Philadelphia, Penn.
Charles Lennig & Co., Philadelphia, Penn.
Erie Chemical Co., Erie, Penn.

[1] C. W. Hayes, Contributions to Economic Geology, 1906, pp. 215-223.

Franklin H. Kalbfleisch Chemical Co., Broadway and 16th Street, New York.

Cochrane Chemical Co., 55 Kilby Street, Boston, Mass.

Merrimac Chemical Co., Boston, Mass.

Detroit Chemical Co., Detroit, Mich.

The greatest use for alum is probably in the paper trade, where it is utilized for sizing or giving a hard finish to paper. Aluminum sulphate and also crystals of potash alum are used in water purification. Crystals of potash alum are used in the dyeing trade. A small quantity of alum is used in medicines.

ALUNDUM.

There was a large decrease in the production of alundum in the United States during 1908, due to the depressed condition of the industries in which this abrasive is used. The prices for alundum grain and grinding wheels were the same as during 1907. The Norton Company, of Worcester, Mass., is still the sole manufacturer of alundum. The quantity and value of alundum produced since 1904, when its manufacture was begun, are shown in the accompanying table.

PRODUCTION OF ALUNDUM IN THE UNITED STATES.

Year.	Pounds.	Value.
1904	4,020,000	$281,400
1905	3,612,000	252,840
1906	4,331,233	303,186
1907	6,751,444	405,086
1908	3,160,000	189,600

The quantity of alundum manufactured in 1908 is not a real indication of the business done in alundum grain, which is sold in the loose form for grinding and polishing. On account of the large stock of alundum on hand at the beginning of 1908, the production was curtailed and this stock was drawn on to some extent for the needs of that year. As a matter of fact, the sales of alundum grain showed a large increase.

AMMONIA AND AMMONIUM SULPHATE.

By C. G. Atwater.

The production of ammonia in the United States in 1908 was affected by the unfavorable conditions existing in all lines of business. This was particularly true with regard to the larger of the sources of production, the by-product coke oven, which naturally shared in the depressed state of the iron industry. Beyond an addition to the by-product coke ovens in existence in the United States by the completion and starting up of two new plants, aggregating 330 ovens, the sources of ammonia remained practically unchanged. The additional output due to these two new plants is unfortunately not apparent in the figures for production, the increase having been swallowed up by the losses due to plants shut down. The production of the United States of ammonium sulphate and sulphate equivalent for the last 11 years is given in Table I. It is understood that this table includes only ammonia actually recovered as such from the carbonization of coal or bone.

TABLE I.—UNITED STATES AMMONIA PRODUCTION, EXPRESSED IN SULPHATE EQUIVALENT.

(Tons of 2000 lb.)

Year.	Tons.	Year.	Tons.	Year.	Tons.
1898....	17,000	1902....	36,124	1906....	(a) 75,000
1899....	(a)19,500	1903....	41,873	1907....	99,309
1900....	(a)27,600	1904....	54,664	1908....	(a) 87,600
1901....	(a)29,279	1905....	65,296		

(a) Estimated.

It will be noted that the production for 1908 shows a falling off of nearly 12,000 tons or about 12 per cent. from that for 1907, a loss that is due to the lessened coke-oven output, as above mentioned. The production from this source was only 49,600 tons as against 62,700 tons in 1907. These figures are compiled from reports made to me by the works in operation during the year. The supply from the coal gas works of the country, the other main source, does not seem to have suffered so much from the dull times.

The amount of ammonium sulphate and sulphate equivalent consumed in the United States together with the importation of sulphate as such and the average market price for each year is given in Table II. As

the exports of sulphate of ammonia from the country are negligible the consumption is arrived at by taking the sum of the production given in the previous table and the imports. The prices are averaged from the

TABLE II.—UNITED STATES AMMONIA CONSUMPTION, EXPRESSED IN SULPHATE EQUIVALENT. (a)
(In tons of 2000 lb.)

Year.	1900	1901	1902	1903	1904	1905	1906	1907	1908
Imports.....	8,411	14,486	18,146	16,777	16,667	15,288	9,182	32,669	34,274
Total consumption.....	36,011	43,765	54,270	58,650	71,331	80,584	84,182	132,000	121,874
Average price.....	$57.40	$55.16	$59.90	$62.10	$61.71	$62.92	$62.33	$61.93	$59.90

(a) The figures for consumption and price are for the calendar year, while those for imports are for the fiscal year ending June 30.

regular market quotations. The figures for 1908 indicate that the consumption practically followed the home production down the scale, though there was a slight increase in imports, the price meanwhile falling $2 per ton. There is no reason to assume that this falling off in the consumption of ammonia is anything more than a passing phase of the straitened business conditions which will disappear with the renewal of general activity. The low price prevailing probably accounts for the imports being but little, if any, larger than in 1907, in spite of the diminished home supply, as foreign producers wishing to sell here have to face charges for freight, costs and tariff aggregating about $13 per ton. When the English price is quoted as $57 there is little inducement to ship here for $60 unless there is a surplus that must be taken out of the market if prices are to be preserved elsewhere.

However, in spite of the drop in consumption last year, the United States remains the second in rank of the ammonia consuming countries, Germany being the first by a large margin, and England the third.

United Kingdom.—The production of ammonium sulphate and sulphate equivalent in the United Kingdom is given in Table III. The production for 1907 given as 316,000 tons by Messrs. Bradbury & Hirsch early in the year was later shown by the Alkali Inspector's reports to be some 3000 tons above the mark. The excess appeared in the returns from the coke ovens and more than balanced an underestimate in the output of the coal gas plants. The correct figures for 1907 are given, together with the estimates for 1908. It will be noted that the increase for 1908 is very small, in fact the smallest of any year since 1898, indicating that the business conditions are having the same influence in England as here. A small loss in the gas works and iron works production is offset by a gain in the coke and carbonizing product, a condition the opposite of what it is in the United States.

TABLE III.—AMMONIUM SULPHATE AND SULPHATE EQUIVALENT PRODUCED IN THE UNITED KINGDOM. (a)

(Tons of 2240 lb.)

Year.	1901	1902	1903	1904	1905	1906	1907	1908
Gas works	142,703	150,055	149,489	150,208	155,957	157,160	165,474	164,000
Iron works	16,353	18,801	19,119	19,568	20,376	21,284	21,024	20,000
Shale works	40,011	36,931	37,353	42,486	46,344	48,534	51,338	51,000
Coke ovens	12,255	15,352	17,438	20,848	30,732	43,677	53,572	} 79,000
Producer gas and carbonizing works	5,891	8,177	10,265	12,880	15,705	18,736	21,873	
Total	217,213	229,316	233,664	245,990	269,114	289,391	313,281	314,000

(a) These figures are from the Alkali Inspector's reports, except those for 1908, which are from Messrs. Bradbury & Hirsch's review.

Allowing for the change in the 1907 figures already mentioned the home consumption was about 84,500 gross tons in 1907 and was 83,000 gross tons in 1908. This includes the sulphate equivalent of the ammonia used for chemical manufacture, as for ammonia soda, as well as the sulphate used in agriculture.

The total exports for 1908 amounted to 234,921 gross tons, of which Spain (including Portugal) took the largest single item, 51,667 tons. Japan being second with 38,745 tons and the United States third with 28,923 tons. Germany and France each practically doubled their imports of 1907 taking 24,426 and 23,634 tons respectively.

The average price for 1908 is given as £11 12s., being a drop of 3s. 8d. from that of 1907, this being for goods delivered f.o.b., Hull. The prices for Liverpool and Leith ruled from 2s. 6d. to 3s. 9d. per ton above those for Hull until near the close of 1908.

TABLE IV.—PRODUCTION OF AMMONIUM SULPHATE AND SULPHATE EQUIVALENT FROM BY-PRODUCT COKE OVENS AND GAS WORKS IN GERMANY. (a)

(In metric tons of 2204.6 lb.)

Year.	1900	1901	1902	1903	1904	1905	1906	1907	1908
Gas works	16,500	17,000	18,000	20,000	21,000	22,000	.....	(b)30,000	.....
Coke ovens	88,500	113,000	117,000	120,000	152,000	168,000	.....	257,000	.....
Total	105,000	130,000	135,000	140,000	173,000	190,000	(b)235,000	(b)287,000	(b)313,000

(a) Dr. N. Caro, *Zeit. f. angew. Chem.*, Sept. 14, 1906.
(b) Deutsche Ammoniak-Verkaufs-Vereinigung, 1906-7-8.

Germany.—The figures given in Table IV for the production of Germany show that that country still maintains its astonishing increase in sulphate of ammonia production. The figure for 1908, 313,000 tons, is within 1000 tons of the English output for the year, so that we may now consider the two countries as on a par as producers. This fact, however, has but little effect on the market in other countries, as Germany's

capacity for production is fully matched by the home demand. This absorbed 284,000 metric tons in 1908, or as much as England, the United States and France put together.

The import into Germany for 1908 is given as 47,265 metric tons, which besides that coming from England, include 17,928 metric tons from Austro-Hungary. The export amounted to 72,001 metric tons, the principal items being 18,898 tons to the Dutch East Indies, 15,095 to Belgium, 11,514 to Holland and 9889 to Japan.

France.—The production of sulphate of ammonia and its equivalent in France is given by *L'Engrais* as 57,600 metric tons for 1908 as against 52,700 tons for 1907. The imports from England and Germany total 27,671 tons for 1908, there being probably imports from other countries as well for which no figures are at hand. The consumption of the country is estimated by the Deutsche Ammoniak-Verkaufs-Verein at 87,000 tons.

Japan.—The demand for sulphate of ammonia in Japan has continued, though not at the rate it was taken in 1907, which indeed was hardly to be expected. There is reason to think that the shipments in that year were beyond the immediate requirements of the country, though this would possibly not have been the case but for the contraction of business which was felt in Japan as elsewhere.

As already noted, Germany exported sulphate to Japan, as well as England, the total for these two countries for 1908 being 48,634 tons.

TABLE V.—EXPORTS OF AMMONIUM SULPHATE FROM THE UNITED KINGDOM TO JAPAN.

(Tons of 2240 lb.)

1901	1902	1903	1904	1905	1906	1907	1908
1,290	2,429	3,612	14,981	33,861	33,237	64,270	38,745

TABLE VI.—ESTIMATED WORLD'S PRODUCTION OF AMMONIUM SULPHATE AND SULPHATE EQUIVALENT.

(Metric tons of 2204.6 lb.)

Country	1902	1903	1904	1905	1906	1907	1908
England	233,100	237,520	250,050	273,550	294,170	318,400	319,130
Germany	135,000	140,000	173,000	190,000	235,000	287,000	313,000
United States	32,800	38,000	49,600	59,250	68,000	90,120	79,500
France (*b*)	40,000	52,000	43,000	47,300	49,100	57,200	57,600
Belgium and Holland (*b*)	38,000	35,000	(*a*)39,000	24,200	30,000	(*a*)55,000	(*d*)35,000
Spain (*b*)	45,000	45,000	48,000	10,000	10,000	(*c*)12,000	(*d*)80,000
Italy (*b*)				4,500	5,000	11,000	
Other Countries (*b*)				40,500	40,000	65,000	
Total	523,900	547,520	602,650	649,300	731,270	991,200	884,200

(*a*) Including Norway, Sweden and Denmark. (*b*) Estimates from *L'Engrais*. (*c*) Including Portugal. (*d*) Estimates from Deutsche Ammoniak-Verkaufs-Vereinigung, 1908.

ANTIMONY.

The production of antimony in the United States in 1908 was small, the mining of this ore being generally unprofitable at the prices of the year. Mathison & Co., on Staten Island, N. Y., and the Chapman works at Oakland, Cal., continue to be the only smelters producing antimony. Only the former was in operation in 1908. The chief domestic source of the metal continues to be the antimonial lead produced by the silver-lead smelters.

Market.—The New York market for antimony in 1908 was quiet and without the sensational fluctuations in price that characterized the market in 1907. The chief factors which influenced low prices were heavy ship-

AVERAGE MONTHLY PRICES OF ANTIMONY IN NEW YORK.
(Cents per pound.)

	Jan.	Feb.	Mar.	Apr.	May	June	July	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
1901.													
Cookson's	10.00	10.00	10.00	10.31	10.25	10.25	10.25	10.25	10.12	10.09	10.00	10.00	10.12
Hallett's	9.12	9.22	8.90	8.94	8.75	8.75	8.75	8.43	8.50	8.47	8.37	8.31	8.74
Others	9.25	8.85	8.77	8.73	8.63	8.63	8.63	8.50	8.37	8.34	8.25	8.00	8.55
1902.													
Cookson's	10.00	10.00	9.87	9.87	9.87	9.87	9.75	9.75	9.69	9.44	9.25	9.20	9.71
Hallett's	8.17	8.04	8.06	8.06	8.17	8.25	8.25	8.15	7.92	7.72	7.44	7.25	7.96
Others	7.86	7.75	7.75	7.75	7.90	8.00	8.00	7.90	7.65	7.37	7.22	6.92	7.67
1903.													
Cookson's	8.25	8.25	8.25	8.25	8.00	7.50	7.44	7.15	7.00	7.00	6.56	6.75	7.53
Hallett's	7.00	7.00	6.87	6.87	6.75	6.69	6.50	6.40	6.34	6.25	6.25	6.35	6.69
Others	6.75	6.62	6.50	6.50	6.50	6.44	6.25	6.19	6.00	6.00	6.00	5.95	6.31
1904.													
Cookson's	6.938	7.594	7.875	7.875	7.531	7.200	7.188	7.188	6.913	6.984	7.592	8.388	7.439
Hallett's	6.250	6.781	6.825	6.750	6.578	6.438	6.485	6.688	6.537	6.578	7.328	8.160	6.783
Others	5.688	6.203	6.475	6.406	6.203	5.961	5.969	6.062	6.015	6.172	7.204	8.088	6.371
1905.													
Cookson's	8.375	8.375	8.375	8.219	8.406	11.025	12.625	14.500	13.700	13.000	12.500	14.000	11.100
Others	8.063	8.063	7.638	8.125	8.406	10.175	11.875	13.500	12.900	12.000	11.250	12.750	10.400
1906.													
Cookson's	15.0	16.0	17.5	21.31	25.25	26.0	25.25	25.0	24.5	25.2	26.14	26.25	22.78
Hallett's	14.0	15.0	16.5	20.81	24.38	25.0	24.25	24.0	24.0	24.81	25.25	25.24	21.94
Others	13.5	14.25	16.15	20.25	23.31	24.0	23.19	22.75	22.25	23.63	24.50	24.70	21.73
1907.													
Cookson's	25.906	25.062	24.90	24.125	21.937	15.75	11.875	10.906	10.75	11.75	11.00	9.662	16.969
Hallett's	25.219	24.062	23.75	21.344	18.562	13.812	10.50	9.687	10.00	10.406	9.937	9.05	15.527
Others	24.156	23.437	23.025	20.875	17.75	12.65	10.125	9.375	9.65	10.047	8.906	8.088	14.840
1908.													
Cookson's	9.344	9.266	9.000	8.969	8.875	8.734	8.594	8.313	8.234	8.284	8.640	8.200	8.704
Hallett's	9.031	9.016	8.650	8.672	8.625	8.531	8.375	8.150	7.922	7.922	8.075	8.063	8.419
Others	8.344	8.406	7.988	8.297	8.250	8.094	8.125	7.850	7.609	7.625	7.775	7.688	8.004

ments of Chinese ore which readily cared for the demand and the restricted use of the metal by the railroads due to the commercial depression. Throughout 1908 prices for Cookson's fluctuated between narrow limits; in January the average price was 9.344c. per lb. and in December

it was 8.20c. per lb. Hallett's brand was very scarce at the close of the year and Hungarian antimony was not prominent. Low prices affected these brands more than Cookson's.

There was a steady decline in the price for powdered needle during the year. The market opened at 18c., but by March 1 a 16c. basis was current. May brought a 13c. level and by the end of July there were sellers at 9c. The decline continued, 7¾c. being recorded early in September. Toward the end of that month there was a check to the declining course of ore abroad and the market for powdered needle on spot rallied to 9c. early in October. The advance could not be maintained and further declines were recorded during the closing weeks, 7½c. prevailing at the year's end.

Antimony Mining in the United States.

The production of antimony in the United States in 1908 continued small, the low range of prices for the metal checking the industry. In fact, but little of the production reported in 1908 was shipped, most of

ANTIMONY STATISTICS OF THE UNITED STATES.
(In tons of 2000 lb.)

Year.	Imports.		Exports.		Production.			Consumption.
	Metal or Regulus.	Ore.	Metal or Regulus.	Ore.	In Hard Lead.	From Domestic Ore.	From Imported Ore. (a)	
	Short tons	Short tons	Short tons	Short tons	Short tons	Short tons	Short tons	Short tons
1897.....	573	2,751	.....	.....	2,217	245	1,100	4,135
1898.....	1,013	1,863	13	17	2,118	250	738	4,106
1899.....	1,580	1,991	Nil.	Nil.	1,586	234	796	4,196
1900.....	1,816	3,018	21	Nil.	2,476	151	1,207	5,638
1901.....	1,837	866	Nil.	25	2,235	50	336	4,458
1902.....	2,871	840	3	104	2,904	Nil.	294	6,032
1903.....	2,563	1,337	40	Nil.	2,552	Nil.	535	5,610
1904.....	2,028	1,245	16	214	2,515	Nil.	412	4,939
1905.....	2,869	988	Nil.	Nil.	2,561	Nil.	395	5,825
1906.....	3,950	1,124	12	Nil.	3,358	150	450	6,866
1907.....	4,331	1,380	24	6	2,240	105	552	7,204
1908.....	4,057	1,640	Nil.	Nil.	2,621	180	656	7,514

(a) Estimated at 40 per cent. extraction from net imports of ore.

the ore being held at the mines, awaiting a more favorable turn in the market.

Alaska.—Veins of antimony ore have been prospected on Manila creek, about 15 miles north of Nome; also on Last Chance creek, and near the mouth of Goldbottom creek on Snake river.

Idaho.—The Stanley Consolidated Mining and Milling Company, of Spokane and Burke, produced some antimony ore, assaying 50 per cent. antimony, which was held at the mine. Shipments were made of ore light in antimony, but containing about $20 gold per ton, for which pay-

ment was received for the gold, but nothing for the antimony content. The other antimony mines in Idaho were idle in 1908.

Washington.—The Lucky Knock Mining Company, of Loomis, made a small production of ore assaying 50 per cent. antimony. During the most of 1908 attention was directed toward development work, which it is intended to prosecute vigorously during 1909.

Utah.—The Utah Antimony Company produced a considerable tonnage of ore, averaging about 50 per cent. antimony, but made no shipments. The other mines in the same district were idle. The Utah Antimony Company, at Antimony, Garfield county, has a smeltery (sublimation process), but it has not yet been operated.

Nevada.—A small quantity of ore was mined, but none was shipped. Smith & Green operate a mine in Bullion cañon, 15 miles south of Mill City, and have a small reverberatory furnace, using sage brush as fuel, by which regulus, containing 80 to 90 per cent. of antimony is produced. E. W. Brackett, of Columbus, Ohio, located a 4-ft. vein of antimony ore in southern Nevada in 1908.

THE PRINCIPAL SUPPLIES OF ANTIMONY ORE. (a)
(In metric tons.)

	1897	1898	1899	1900	1901	1902	1903	1904	1905	1906	1907	1908
Austria	864	679	410	201	126	18	41	103	1,673	1,071	910	.....
Bolivia	.....	.....	1,213	1,174	190	126	59	7	17	571	.....	.....
Canada (f)	Nil.	1,118	(c)	6	219	13	128	87	340	1,425	2,048	.....
France & Algeria	5,466	4,571	7,592	7,963	9,867	9,715	12,380	9,065	12,543	13,567	25,200	.....
Hungary	1,800	2,201	1,965	2,373	323	748	205	1,080	949	580	2,598	.....
Italy	2,150	1,931	3,791	7,609	8,818	6,116	6,927	5,712	5,063	5,704	7,900	.....
Japan (b)	348	1,006	712	81	119	88	153	104	96	97	.....	.....
Mexico (c)	5,873	5,932	10,382	2,313	5,103	1,279	1,856	1,775	2,035	2,418	4,615	4,046
N. S. Wales (d)	172	84	332	252	90	57	13	111	394	2,490	1,780	106
New Zealand	10	.....	.....	5	30	.....	.....	.....	.....	.....	.....	5
Portugal	417	245	59	38	126	68	83	31	84	481	.....	.....
Queensland	.....	.....	41	.....	.....	.....	.....	Nil.	24	Nil.	Nil.	42
Spain	354	130	50	30	10	67	42	245	77	180	205	.....
Turkey	400	(e)	1,173	267	224	(g)481	(g)1,903	(g)298	(g)188	(g)1,036	.....	.....
United States	454	(e)	544	300	100	Nil.	Nil.	Nil.	Nil.	267	190	326

(a) From official reports of the respective countries. It will be observed that this table omits the production of China, for which no statistics are available. (b) Mostly crude antimony. (c) Export figures, except for 1903, which represents production. (d) Metal and ore. (e) Not reported. (f) Previous to 1906 the figures represent exports for the fiscal year ending June 30; the figures for 1906 and 1907 were collected by *The Mineral Industry* and represent production for the calendar year. (g) Exported.

PRODUCTION OF ANTIMONY METAL IN FOREIGN COUNTRIES. (b)
(In metric tons.)

	1896	1897	1898	1899	1900	1901	1902	1903	1904	1905	1906	1907
Austria	422	425	343	271	153	114	24	14	36	90	Nil.	207
France	969	1033	1 226	1,499	1,573	1,786	1,725	2,748	2,116	2,396	3,433	3,950
Hungary (a)	500	523	855	940	846	706	683	732	1,007	756	1,322	841
Italy	538	404	390	581	1,174	1,721	1,574	905	836	327	537	610
Japan	517	823	235	229	349	429	528	434	321	190	627	.....

(a) Regulus. (b) China, which is a large producer of antimony is not included in this table, statistics not being available.

Antimony Mining in Foreign Countries.

All over the world the mining of antimony ore appears to have been affected acutely by the great drop in the value of the metal and the low range of prices in 1908.

Australia.—In 1907 the Victorian Syndicate, Ltd., at Costerfield, Victoria, produced 3900 long tons of antimony ore from which was obtained 830 tons of concentrates valued at £12,450, and assaying 45 to 50 per cent. antimony, and 35 to 38 dwt. gold per ton. The Heathcote Syndicate, Ltd., at the same place, produced 600 tons of ore yielding 70 tons of concentrates valued at £840. The concentrates from both companies were smelted in England. In Queensland, antimony ore is mined at Thornborough, Kingsborough, Woodville and Northcote, in the Hodgkinson district, but the low prices of 1908 checked operations generally.

Canada.—According to a report (in August, 1908) of the correspondent of the British Board of Trade at St. John, New Brunswick, a new company has acquired the antimony deposits at Lake George, York county, New Brunswick, and has begun active operations. The property contains two well-defined veins. Two shafts have been sunk to a depth of 220 and 260 ft. respectively, and eight other shafts to depths of 25 to 100 ft., all showing pay ore, and proving the veins for more than a mile in length. The veins vary from 6 in. to 4 ft. in thickness, 20 in. being the average. Assays show 50 to 78 per cent. antimony. The property is about 24 miles from Fredericton, and access both to that city and to St. John, the chief shipping port of the province, is obtained either by rail, or by water navigation on the St. John river during the summer. The company plans to mine about 50 tons a day and estimates that there is about 50,000 tons of ore developed.

China.—This country is a large producer of antimony ore which is shipped as ore and metal, and as liquated sulphide. A good deal of the last, known as needle antimony, is imported from China into the United States. The exports of crude antimony from Changsha in 1907 were 1750 long tons, against 2375 in 1906 and 1885 in 1905; of ore, 3092 in 1907, against 2271 in 1906 and 1987 in 1905. The officials and gentry of Hunan, China, convinced that the foreign merchants manipulate the antimony market against them, propose starting a refinery of their own, in which modern methods will be introduced. A loan from the Provincial Government to help pay the cost of this experiment was obtained. A Herrenschmidt plant has been sold for installation in China. The Imperial works at Wuchow does an extensive business. The antimony mine at Kug-Kong is reported to be producing an average of 70-75 tons of ore per month.

(By T. T. Read.)—Antimony is said to occur in the provinces of Kiangsi, Kuei-chou and Ssu-chuan, but nearly all of the production comes from Hunan. The deposits are to the north-west of T'ung-t'ing lake; I-yang is the chief center, but the ore is derived from several localities. From the mines it is brought in shallow-draft boats to Chang-sha, where the regulus is liquated. The ore that I saw at the works was a clean stibnite, which had obviously been hand-picked to free it from gangue. The liquation is carried on in small pot furnaces, the pots being about 15 in. in diameter. The regulus is sent to Hankow, where it is refined, and so much as is desired is converted into metal. The residues are at present exported to Europe to be treated for their content of silver and other metals, but it is not improbable that the principal native company will eventually treat these themselves, as they have in their employ C. Y. Wang, who is known to the profession as the author of a recent paper on the metallurgy of antimony, and also S. T. Kong, who has studied in the United States. The amount which passed through the customs in 1907 was 3957 tons of regulus, and 14,810 tons of ore. The incomplete figures for 1908 indicate an enormous growth in the local smelting industry, as the figures from two of the three principal points show a production of 8302 tons of regulus and 1745 tons of ore.

France.—The industry in this country in 1908 suffered the same depression as elsewhere. Most of the French mines decreased their productions. Some, like the Franco-Italienne Company, discharged a good number of their staff while awaiting a rise in quotations to £40 or £50. In the meantime, existing stocks had to be sold by some means or other. Consequently very low offers were made. There was much apprehension over the realization of a large stock in Auvergne. The existence of these unsold stocks was a weight upon the industry.

Great Britain.—Two veins, carrying ore of over 50 per cent. metal, free from galena, were slightly worked at Bassenthwaite about 60 years ago. A smelting company is stated to be contemplating reopening, and, according to expert reports, there is a reasonable prospect of finding workable ores and also arsenical pyrites.[1]

Portugal.—A zone of mineralized quartz veins carrying stibnite and native gold, 10 km. in width and 60 km. in length, extends to the south-east from the mountain of Vallongo (district of Oporto) to the banks of the river Arda.

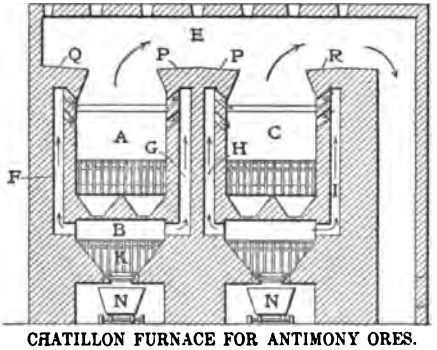
Progress in the Metallurgy of Antimony.

During the last year or two metallurgical literature has been rich in contributions upon this subject. Several important papers appeared in

[1] *Min. Journ.*, Jan. 9, 1909.

1908. This branch of metallurgy is no longer a secret art. The older method of liquating stibnite ore and precipitating the antimony by iron is fast disappearing. The more modern method is to charge the ore with fuel in a small shaft furnace and urge the fire with a large excess of air. The antimony sulphide liquates out, is oxidized and the oxide is condensed in suitable chambers and flues in combination with the furnace. The purity of the condensed oxide varies greatly in the different chambers of the condensers, though it is not uncommon to obtain a product carrying 98 to 99 per cent. oxide. Arsenic oxide is the greatest impurity. For reduction to metal this oxide is briquetted and charged into a reverberatory furnace with coke and carbonate of soda in the proportions of 700 oxide, 70 soda and 80 carbon.

Chatillon Process.—Emmanuel Chatillon, of Brionde, France, in *Min. Journ.*, July 25, 1908, described a new process of his invention (French patent, 382,504, Oct. 3, 1907). According to this, volatilization-roasting of the ore is performed in a furnace composed of two double cupolas, *A* and *C*, united by a horizontal chamber, *E*, common to both. Each of these cupolas, *A* and *C*, is mounted on another little cupola furnace, *D* and *B*, which receives the cinder from the furnace above and completes



CHATILLON FURNACE FOR ANTIMONY ORES.

the extraction of volatile substances, and at the same time serves to cool the cinder and pre-heat the air supplied to the upper cupolas through flues in the masonry. Each of the lower cupolas is furnished on two sides with inclined grate bars, *K*, and on the other two sides with either masonry or similar inclined bars, thus forming a kind of hopper. The bottom of this hopper consists of a perforated iron grate. By drawing out this grate the spent material is allowed to drop into the truck *N* for removal.

The two upper furnaces, *A* and *C*, have a common top, *E*, the wall between the two furnaces terminating at least 1 m. below the arch. The

upper part of this dividing wall is enlarged on each side by a brick projection, *P*, and constitutes a hearth with surface of about 4 sq.m. The charge of ore is introduced upon this hearth, where the antimony sulphide is melted and liquated out, running half into one cupola and half into the other, passing in its fall through the hot air from the cooling of the slag in the little cupolas below. It is thus almost immediately oxidized and volatilized. The benches, *Q* and *R*, serve as hearths in the same way as *P*. After the sulphide has liquated out, the residue is pushed into the cupolas to complete the extraction.

From the cupolas the gases are drawn by fans into cooling and condensation chambers. On their way to the latter, they are utilized for various purposes, such as heating furnaces in which oxides are reduced to metallic antimony, heating liquating furnaces for the production of antimony sulphide, distillation of ore with or without oxidation, and with or without steam, for various coloring materials with base of antimony; reduction of barium sulphate; heating of air for the dryers for fine ore and other material. The final condensing apparatus consists of sheet iron tanks, of rectangular form, through which water is circulated. The tanks are completely inclosed in chambers, which are so arranged as to compel the gas to circulate over the entire surface of the tanks. The gas cooled in this way passes to a bag-house, which is equipped with bags of coarse cotton cloth, or wool, usually 50 to 60 cm. in diameter and 8 to 11 m. in length. They are hung and arranged precisely as in lead and zinc smelteries.

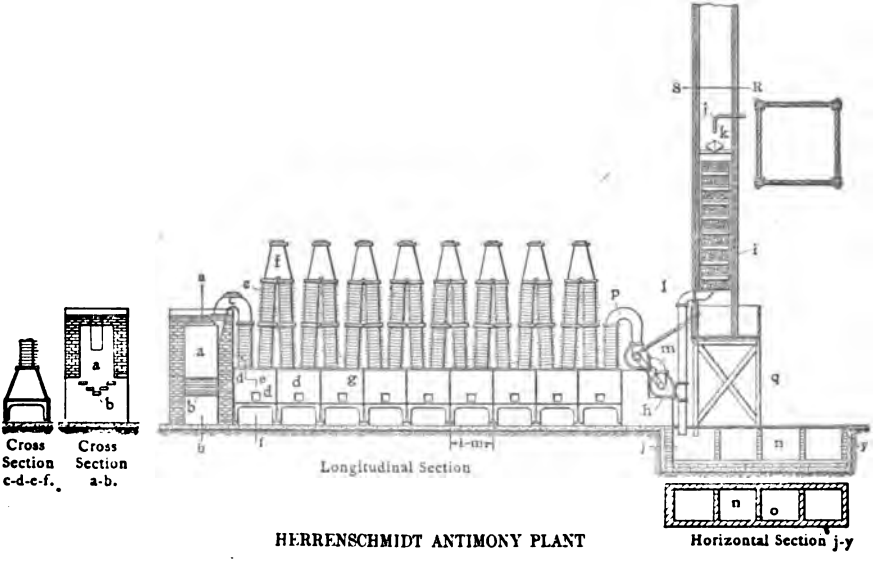
Herrenschmidt Process.—This was described by C. Y. Wang, in *Min. Journ.*, July 25, 1908. His article was based upon a personal study of the new process as carried out in the works of Mr. Herrenschmidt, at Le Genest (Mayenne) France.

The guaranteed results of such an apparatus are that the yield in oxide will not be less than 90 per cent. of the metal contained in the ore. With ores coming directly from the mine, and containing 10 to 15 per cent. of antimony, six tons per 24 hours can be treated, with a consumption of 4 to 5 per cent. of gas coke or 6 to 7 per cent. of charcoal. With ores containing 18 to 20 per cent. of antimony the capacity is 4½ tons per 24 hours, with a consumption of 5 to 6 per cent. of gas coke, or 7 to 8 per cent. of charcoal. The gold, if present in the quartz of the ore, is recovered from the scoria. For each apparatus four workmen are employed per 24 hours, with 3 h.p. for the fans and pump.

The best size of the ore to be treated is from 1 to 4 cm. The dust must be sifted out; but it may be agglomerated with 7 to 8 per cent. of clay and then charged with the ore.

The roasting furnace is at first charged with old scoria up to a hight of about 10 cm. above the last one of the iron bars forming the bottom;

then wood and charcoal are put in to a hight of another 10 cm.; as soon as they are lighted and burn freely, another charge of 300 kg. of old scoria, mixed with 6 per cent. of charcoal, is put in; and, finally, when the furnace is hot, a charge of 50 kg. of ore (of about 15 to 20 per cent. Sb), mixed with 2 kg. of charcoal, is put in, and this is continued until the furnace is full to within 30 cm. from the mouth. The same charge is continued every 12 to 15 min., according to the speed of the fans, which is from 1200 to 1400 r.p.m., and the scoria is discharged accordingly. Gas coke gives better results if it is obtainable; and if it is used, only 2 to 6 per cent., or 1 to 3 kg. for every 50 kg. of the same class of ore, are necessary, according to the richness of the ore roasted. The temperature of roasting should not be more than 400 deg. C.

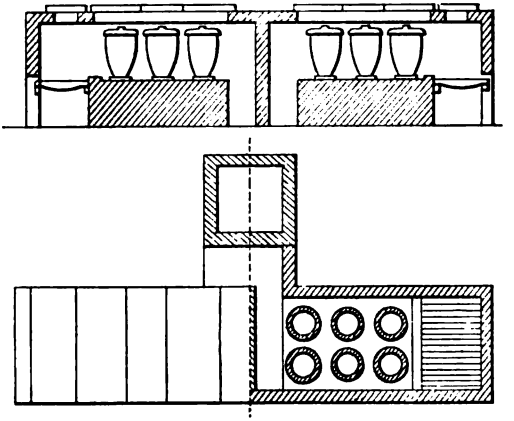


HERRENSCHMIDT ANTIMONY PLANT

Fig. 1 is a longitudinal section of the apparatus. Fig. 2 is a cross-section along *a-b*. Fig. 3 is a cross-section along *c-d-e-f*. Fig. 4 is a horizontal section along *j-y*.

The broken mineral is charged, together with a certain percentage of fuel, into the roasting furnace, *a*, the bottom of which is inclosed by horizontal iron bars, *b*, arranged symmetrically. The volatile antimony trioxide is delivered by two fans, *h*, into the settling chambers, *d*, through pipes, *e*, consecutively. Finally the last of the oxide is caught by the water coming from the coke tower, *i*, at the top of which is a tilting water distributor, *k*. The roasting furnace is built partly of bricks and partly of castings; the condensation chambers are of cast iron; the coke tower is of wood. The length of a single condensation chamber is 1 m.

In the first experiment, of which this article is a report, the furnace was lighted at 8:30 a.m. on April 24, and it was not until 10:25 a.m. when the furnace was hot enough to admit the first charge. The working was continued regularly until 11 p.m. the same day, the total working time being 12 hours and 55 min. The charges were run thus: Each charge consisted of 50 kg. of ore, minus the dust, which had previously been sieved off. The first three charges were mixed with 10 per cent. of charcoal, the fourth to the 10th charges with 4 per cent., the 11th to the 19th with 2 per cent., and from the 19th to the last with 4 per cent. The total charges amounted to 2850 kg. of ore. The products of roasting (Sb_2O_3) were: 1st chamber of condensation, 76 kg.; 2nd, 65; 3rd, 65; 4th, 48; 5th, 49; 6th, 44; 7th, 34; 8th, 24; 9th, 19; total, 424. It should be noted that the cleaning of the chambers and pipes to get all the oxide out was done in the next experiment.



REVERBERATORY FURNACE FOR REGULUS SMELTING IN CRUCIBLES.
Scale 1:40.

In the second experiment the first charge commenced at 11:05 a.m. on Apr. 25, and the last charge stopped at 10:30 p.m. the same day, a total of 11 hours and 25 min. Each charge consisted of 50 kg. of ore, mixed with 4 per cent. of charcoal. The total charges amounted to 2725 kg. of ore. The product was: 1st chamber of condensation, 176 kg.; 2nd, 103; 3rd, 96; 4th, 61; 5th, 45; 6th, 38; 7th, 32; 8th, 28; 9th, 23; total, 602. These numbers represent the actual amount of oxide extracted from the ore, plus the cleanings of oxide, due in part to the first experiment. Further cleaning of the pipes and chambers gave 32 kg. The basins of condensation, *n*, in Fig. 4, gave 178 kg. of dried oxide. If all had thoroughly been cleaned up there would have resulted at least 5 kg. of oxide more.

The ore treated contained 19.5 per cent. of antimony. The oxide found in the first chamber contained 23 per cent. of impurities. An analysis of an average sample of the oxide from the rest of the chambers gave 0.5 per cent of impurity.

Total quantity of oxide obtained: 1st experiment, 424 kg.; 2nd experiment, 602; dried oxide (from basins), 178; cleanings, 32; estimated cleanings, 5; total, 1241 kg. Ore treated: 1st experiment, 2850 kg.; 2nd experiment, 2725; total, 5575. The actual quantity of pure oxide obtainable was 1178 kg., which is equivalent to about 981 kg. of pure antimony. In 5575 kg. of ore there was 1087 kg. of Sb. Hence the extraction was a little more than 90 per cent. The scoria was found to contain only 1.3 per cent. Sb. Practically in 24 hours 5½ tons of such ore can be treated with a consumption of 4 per cent. charcoal. It is probable that at least seven tons can thus be treated per day, if the speed of the fans be increased from 1200 to 1400 r.p.m. Generally speaking, it costs about 70 francs to produce one ton of oxide by this process in France.

In discussing the above article, J. Henry Rickard, of Newquay, Cornwall, said (*Min. Journ.*, Aug. 1, 1908): "I had experience from 1890 to 1901 in the treatment of antimony by volatilization; during this time I erected several large plants, modified others, and have seen some 10 different works treating ores by the same system as is described as the 'New Process.' Emmanuel Chatillon, the inventor of the process for treating low grade ores by sublimation, also the patentee of a process for obtaining Sb_2O_3 direct from the ores, used the coke tower 20 years ago, besides all sorts of appliances with water jets, filtering beds made of faggots of wood, shavings, etc. Giraud & Co., of the same town, erected a tower with filtering beds composed of coke, pottery, etc., with water spraying. At Brionde (Haute Loire), Massiac (Cautal), Couches (Cautal), Collel-de-Deze (Lozere), Marseilles, Rosia (Tuscany), and in other antimony works, the organ-pipe system, combined with fans and water-jets, formed the basis of the condensing plants. In one case, besides a long length of fire-brick chambers, I put in 200 pipes, fans, and a chamber with divisions and water sprays to lay the last trace of uncondensed oxide, or rather all that could be practically condensed. The Herrenschmidt process, according to drawings and description, offers no new features as far as the apparatus or handling of same are concerned. The furnace may be slightly altered in size, shape, or arrangement of fire bars; the series of pipes are connected at top and bottom by castings instead of wrought iron or brickwork; the pipes are corrugated, or made of ailette iron, giving a larger cooling surface, and perhaps more friction to the gases passing through them. The cooling

question, according to several experiments on a large scale, has no effect on the condensation after a certain point; the loss of oxide is purely mechanical, and, in spite of water-jets, filtering cloths, mechanical screens, and a length of over a mile of zig-zag flues following the condensing plant, a certain amount of very fine particles of oxide will escape. The percentage of fuel used in the two trials augurs well for cheap working; one must take into consideration that these were run in a clear furnace, heated to a fairly high temperature, before any charges were put in. In practical work, say, on a three months' run, the quantity consumed would probably be higher; but this depends not only on the richness of the ores treated, but also on their structure, size of stuff, and nature of the gangue. Some ores can be treated at a low temperature, whilst others require a much higher one. I doubt if four men per 24 hours can attend to both charging the ore and fuel, also discharging the scoria, in regular work, especially with small charges every 12 min. The results obtained are good, but I must confess that I do not see anything to call the means employed 'a new process,' nor the object for a patent, unless the substituting of iron for brick, or the condensed space occupied by the furnace, condensing pipes, coke tower, and basins could form a new departure from the more elaborate plants that have been erected elsewhere."

To the above Mr. Herrenschmidt replied (*Min. Journ.*, Dec. 12, 1908): "In fact, it is only a question of a new apparatus. It is just because the old process of condensation or sublimation had only given insufficient results, that I searched for, and, I believe, found, that which many others had searched for before me—viz., an apparatus capable of giving a perfect condensation." Mr. Herrenschmidt then presents data to show that he, not M. Chatillon, is the inventor of the process of treating low-grade antimony ores by sublimation. The Herrenschmidt process is controlled by the Société des Brevets à Procédés, H. Herrenschmidt, 21 Rue de Moscow, Paris.

Pautrat's Proposal.—G. Pautrat, in *Revue de Chimie Industrielle,* November, 1908, criticizes the present volatilization processes, which he says have numerous defects. The temperature is irregular and much too great. Thus there are often difficulties in removing slag which is sometimes vitrified. Another inconvenience is that the temperature required to oxidize antimony sulphide in the form Sb_2O_3 is 350 to 400 deg. C. This furnace gives a much greater temperature, and easily exceeds 700 deg. C. required to decompose arsenio-sulphide of iron usually in the ore. Thus the antimonious oxide collected contains antimonic acid, Sb_2O_5, not merchantable in some cases, arsenic and arsenious anhydrides, and even iron oxide. If it be considered that the antimony oxide obtained can

partly be used for paints, it can be seen that the process is often unsuitable. Transformation of the oxide into metal is effected in reverberatory furnaces with coal and sodium carbonate. Here, again, there are irregularities in the work of manufacturers. Thus some fire the furnace with coke, though a long flame is necessary to reach the charge; briquettes are clearly the fuel to be used. The following method, according to Mr. Pautrat, can be substituted, with profit: Closed muffle furnaces for oxidation, in which the antimony sulphide is volatilized, being oxidized in a special chamber (Rasse-Courbet process), or where oxidation is produced at a specific temperature avoiding entrainment of arsenic. Regulus furnaces can be constructed like those described in the *Revue de Chimie Industrielle,* March, 1907, which can treat the ore with iron or reduce the oxide with carbon. The latter reduction is usually effected with charcoal, but would be much more economical with ordinary coal dust.

Reduction of Antimony Oxide.—According to L. Le Meunier (French patent, 392,169, July 2, 1908) antimony oxide is intimately mixed with 10 to 15 per cent. of charcoal or anthracite, 2 to 5 per cent. of a flux such as sodium carbonate, and 12 to 15 per cent. of water, the mixture being afterward agglomerated in a suitable machine. The product, in the form of lumps, is melted in a cast iron crucible or reverberatory furnace while still wet. This last condition, while modifying the violence of the reaction, more than doubles its velocity. After the layer of slag has been removed, a mixture of oxide, 90, and carbonate, 10 per cent., is added so as to cover the charge, which is finally run into an ingot mold.

Germot Process.—Antonin Germot proposes to treat sulphide ore by injection of air to obtain metallic antimony and a sublimated sulphide, or an antimony oxide or oxysulphide suitable for use in the arts, or for subsequent treatment to produce antimony. Antimony sulphide is first melted in a converter, and air is then injected into the molten mass. The oxygen burns part of the sulphur in the ore, producing sulphurous acid and antimony, which remains in the crucible. The operation becomes continuous by addition of further quantities of ore. The sulphurous acid escapes through a pipe at the top of the converter, and entrains antimony sulphide fumes, which are condensed in special compartments. If, instead of working in closed vessels, a current of air is brought to act in these fumes, antimony oxide or oxysulphides are produced, according to the percentage of air.[1]

Treatment of Lead-Antimony Ore.—It has been shown to be quite feasible to smelt antimony ore along with silver-free galena in the blast furnace, producing antimonial lead, which can be used directly for the manufacture of babbitt metal and other alloys. This has been done by

[1] *Revue Chimique,* October, 1908.

the Hoyt Metal Company, at Granite City, Ill., and also at Anhalt and elsewhere in Germany, and also in France. Lead and antimony occur together in the mineral jamesonite, which is not of uncommon occurrence. G. P. Ives and I. D. Ossa have made experiments on the smelting of this ore as it occurs in the Black Hills, S. D. (*Eng. and Min. Journ.*, May 1, 1909). Ore tested contained from 4 to 17½ per cent. antimony and 8 to 32 per cent. lead. A base bullion containing 79 per cent. lead and 16 per cent. antimony was produced. It was found that the smelting of this ore can be done easily in the blast furnace without previous roasting, but fine concentrates should be roasted, and a mixture that will form about 8 per cent. of matte should be made. It is desirable to have some copper in the matte. The volatilization of antimony is upward of 10 per cent., but this can be recovered by passing the fumes from the blast furnace through a baghouse. Antimony can be obtained from the base bullion by the Betts process of electrolytic refining.

Antimonial Mixtures.—C. W. Leavitt & Co., St. Paul building, New York, give the following formulas in which antimony is used: Britannia metal—Antimony, 10 per cent.; tin, 90 per cent. Pewter—Tin, 89.30 per cent.; bismuth, 1.80 per cent.; antimony, 7.10 per cent., and copper, 1.80 per cent. A hard mixture of babbitt suitable for ordinary requirements—Tin, 100 lb.; antimony, 10 lb.; copper, 10 lb. Another good bearing metal—Lead, 80 lb.; antimony, 15 lb.; tin, 5 lb. A good linotype mixture—Lead, 83 lb.; antimony, 12 lb.; tin, 5 lb. An electrotype metal—Lead, 100 lb.; antimony, 10 lb.; tin, 5 lb.

ARSENIC.

During 1908 the only concerns in the United States which recovered arsenic from ores were the Puget Sound Reduction Company, of Everett, Wash., and the Anaconda Copper Mining Company, of Anaconda, Mont. The most important development during the year was the beginning at Salt Lake City of an arsenical plant by the United States Smelting Company. This plant will treat the flue dust from the smeltery. The production, imports and consumption of arsenic in the United States for a period of years are shown in the accompanying table.

STATISTICS OF WHITE ARSENIC OF THE UNITED STATES.

Year.	Production.			Imports.			Consumption.	
	Pounds.	Value.	Per lb.	Pounds.	Value.	Per lb.	Pounds.	Value.
1897	....	....	....	7,242,004	$352,284	$0.05	7,242,004	$352,284
1898	....	....	....	8,686,681	370,347	0.04¼	8,686,681	370,347
1899	....	....	....	9,040,871	386,791	0.04¼	9,040,871	386,791
1900	....	....	....	5,765,559	265,500	0.04½	5,765,559	265,500
1901	600,000	$18,000	$0.03	6,989,668	316,525	0.04½	7,589,668	334,525
1902	2,706,000	81,180	0.03	6,110,898	280,055	0.04½	8,816,898	361,235
1903	1,222,000	36,691	0.03	7,146,362	256,097	0.03¾	8,368,362	292,788
1904	996,456	29,504	0.03	6,391,566	226,481	0.03½	7,388,022	255,985
1905	1,545,400	50,225	0.03¼	6,444,083	219,198	0.03½	7,989,483	269,423
1906	1,663,000	83,150	0.05	7,639,507	336,609	0.04½	9,302,507	419,759
1907	2,020,000	101,000	0.05	9,922,870	553,440	0.05½	11,942,870	654,440
1908	2,603,505	99,193	0.03¾	9,592,881	417,137	0.04¾	12,196,387	516,330

Some auriferous mispickel was shipped to the Everett smelter from various parts of the State of Washington for the recovery of the contained arsenic. Part of the year the East Helena smelter of the American Smelting and Refining Company shipped its baghouse flue dust to this smelter for the recovery of the arsenic. The Daly Reduction Company, in British Columbia, a producer of arsenical gold ore, also ships to the Everett smelter. On Mineral creek, Pierce county, 60 miles east of Tacoma, Wash., an arsenical ore was discovered. It is said to run high in arsenic and plans are being made to develop the deposit.

The principal producers of arsenic in North America are: Puget Sound Reduction Company, Everett, Wash.; Anaconda Copper Mining Company, Anaconda, Mont.; Deloro Mining and Reduction Company, Deloro, Ont., Canada; United States Smelting Company, Salt Lake City, Utah;

Canadian Copper Company, Copper Cliff, Ont., Canada; Compañia Minera Peñoles, Mapimi, Mex.

Market and Prices.—The local arsenic market was exceedingly irregular during 1908 and the tendency in most cases was toward a lower level. Spot arsenic at the beginning of the year was offered at 5½@6c. This was fairly well maintained through February, but with the continued lack of demand, the price was cut to 4¾c., and later to 4¼c. There was a marked absence of speculative buying during the year, and to this fact may be attributed the unsettled state of affairs. Importers had previously contracted freely for deliveries which were still incomplete, and as demand came to a standstill, accumulation resulted. The low level touched during May, when sales were reported at 3¾c., was the lowest price since December, 1905. Some of the larger holders were ready to unload at option prices to rid themselves of the surplus. In a measure they succeeded and considerable quantities of arsenic actually changed hands at 3¼c. These transactions did not affect the statistical position of the market, for the arsenic did not enter into consumption and was still for sale. By scattering the stock, the market was strengthened, in that holders were in a better position to await a more favorable opportunity for the disposal of stock.

The European shippers became somewhat discouraged by the long series of declines, and some of the English mines curtailed operations. It was expected that Canadian and domestic reduction plants would make good any failure of production on the other side, but these hopes were not realized, and the English mines, not producing enough to supply their home consumption, purchased on the Continent. During the fall months sales were made as low as 3c. Heavy shipments on standing contracts, principally to keep agreements good, relieved the market of part of its burden in December, and the prices were advanced to 3¼c., where they remained until the close of the year.

Arsenic in Foreign Countries.

The *Revue Minière* of Sept. 24, 1908, gives the following tonnage for the world's consumption of arsenical products in 1907: Soda, 10,000; copper acetoarsenite, 7000; arsenious acid, 6500; lead arsenate, 5250; arsenic bisulphide, 2500; copper arsenite, 1500; arsenic trisulphide, 1240; potash arsenite, 495; metallic arsenic, 215; arsenic acid, 10; other products, 5; total, 40,215. Note is made of the important rank of copper arseniate, used for marine paints and insecticides, with lead and soda arseniates. Arsenic green for stained papers is prohibited in Prussia and Sweden. The European countries producing arsenic are, in the order

of their importance, Germany, France, Belgium, Spain and Portugal. The production of mispickel, one of the chief ores, was as follows: France, 6500 tons; Germany, 6300; Spain, 4800; England, 2300; Portugal, 1400; total, 21,300 tons.

Canada.—During the first eight months of 1908 there was imported from Great Britain 113,812 lb. arsenious oxide; from the United States 10,327 lb.; total value, $3825. The importations of arsenic oxide for the same period were: Great Britain, 83,268 lb.; United States, 23,610 lb.; other countries, 72,819 lb.; total value $7512. During 1908, 4000 tons tons of arsenical pyrite, containing 40 per cent. arsenic, were exported. This originated in Eastern Canada, and is exclusive of the amount produced in the Sudbury district. The Canadian Copper Company, Copper Cliff, Ont., and the Deloro Mining and Reduction Company, Deloro, Ont., are prominent producers of arsenic. The production of white arsenic in Canada during 1908 was 634 tons, valued at $38,054.

China. (By T. T. Read.)—Arsenic is produced in southwestern China, but I have been unable to ascertain the exact point of origin. Probably it is immediately west of the T'ung-t'ing lake, near the antimony deposits. A little over 5000 tons passed through the customs stations on the Hsiang river in 1908.

England.—The output of arsenical pyrites in 1908 was 3167 tons in addition to 1919 tons of arsenic.

Mexico.—The Compañía Minera de Peñoles, Mapimi, Durango, treats its flue dust, which is composed largely of arsenical oxides, in a reverberatory furnace. The arsenic is refined and disposed of as a by-product.

Newfoundland.—Arsenopyrite deposits of promising magnitude have been uncovered at Moreton's Harbor, Notre Dame bay. One has been slightly developed, and from it a few tons of picked ore were shipped several years ago, to Wine Harbor, Nova Scotia. This ore contained a little gold. Other deposits have been explored slightly on land licensed to George Hodder, of Moreton's Harbor.

Silesia.—Since 1895 mining has been carried on actively for both gold and arsenic. The most conspicuous rocks in the Reichenstein district are mica schists with inclusions of dolomitic limestone, with which lenses of arsenical ores are associated. The Reicher Trost is the only mine working at present. Its operations are on the largest orebody in the district. The deposit is 460 ft. long at the fourth level. Below this there is a gradual pinching out until on the ninth level the deposit is only 140 ft. long. The thickness of the deposit is variable. Near the sixth level its maximum thickness is 100 ft. The arsenical ore (leucopyrite) occurs in fine acicular crystals intergrown with associated contact minerals. Where the ore occurs with serpentine it is amorphous. The best grades

of ore are pyritic, carrying 50 per cent. leucopyrite. All of the ore carries a small amount of gold.

WORLD'S PRODUCTION OF ARSENIC.
(In metric tons.)

Year.	Canada. (a)	Germany. (b)	Italy.	Japan.	Portugal.	Spain. (b)	United Kingdom. (a)	United States. (a)	France. (d)
1896	Nil	2,632	320	6	.....	271	3,674	.....	.....
1897	Nil	2,987	200	13	524	244	4,232	.....	.....
1898	Nil	2,677	215	7	751	111	4,241	.....	.....
1899	53	2,423	304	5	1,083	101	3,890	.....	2,000
1900	275	2,414	120	5	1,031	150	4,146	.....	4,705
1901	630	2,549	.....	10	527	120	3,416	272	7,491
1902	726	2,828	.....	12	736	71	2,165	1,226	5,372
1903	233	2,768	50	6	698	1,088	916	554	6,658
1904	66	2,829	80	4	1,370	400	992	452	3,117
1905	Nil.	2,535	.....	8	1,562	1,140	1,552	701	3,627
1906	Nil.	3,052	.....	(c)	1,222	1,114	1,625	754	6,534
1907	317	5,000	73	(c)	(c)	2,400	1,497	916	7,900
1908	634	(c)	(c)	(c)	(c)	(c)	1,919	1,301	(c)

(a) White arsenic. (b) Oxide, sulphide, etc. (c) Not yet available. (d) Ore.

Technology and Uses.

Hydrometallurgical Extraction.—A process devised by T. Barton and T. B. McGhie[1] aims to extract arsenic from ores and particularly from speiss without the necessity of working and recovering arsenical fumes, which are not easy to control. The process is most applicable to ores or speiss containing not less than 30 per cent. arsenic, in the form of arsenides. Ores containing much gangue are preferably smelted rough to produce a speiss, which can then be treated in the following way: The material is finely ground and then mixed with enough soda ash so that after all the arsenic shall have been transformed into arsenate there shall still be a small excess of alkali. The mixture is then carefully rabbled in any convenient reverberatory furnace until all the arsenic has combined, while the metals become oxidized. The mass is then leached with water, which removes nearly all the sodium arsenate. The solution is then evaporated until the sodium salt crystallizes on cooling. If desired, this may then be dehydrated by further heating. Using a material containing 40 per cent. arsenic, the resultant anhydrous sodium arsenate will contain 44 per cent. arsenious oxide, while the leached residue will retain not more than 2 per cent., depending on the nature of the crude material used. If the rabbling in the furnace has not been carefully done, the charge may have sintered, preventing the complete transformation of the arsenic. In that event, the whole charge is passed over a 10-mesh sieve; the oversize is then reground and mixed with enough niter to react with the remaining arsenic, usually equal parts of niter and of undecomposed raw material, roasted, and treated as above. If

[1] *La Revue des Produits Chimiques*, Sept. 15, 1908.

the sintered portion of the original charge can be ground fine, the necessity for the niter may be obviated. It is sometimes more convenient to grind less finely and use the niter.

Uses.—Arsenious oxide, or white arsenic, is used as an insecticide; in the preservation of furs and leather; also as a preservative of wood, where it prevents the fungus growth. The most important demand for arsenic is in the manufacture of Paris green, this product being employed as an insecticide. Arsenic is also extensively used as a fixing medium, or mordant, in connection with aniline dyes, and in printing of calico and wall paper. Among the important colors in which arsenic is used are Scheele green, London purple and in certain yellows, reds and grays. Arsenic is also used to a certain extent in the manufacture of glass and enamel; it also finds a small use in medicine. Arsenic soap is another familiar use. Realgar, which is an arsenic sulphide, burns with a bright white light and is used in the manufacture of pyrotechnics. Arsenic hardens lead and finds employment in this connection in the manufacture of shot.

ASBESTOS.

Although the production of asbestos in the United States showed a slight increase in 1908, industrial conditions were far from satisfactory. Over-production at the Canadian mines, whence consumers in this country obtain their chief supply, retarded development and caused a curtailment of output. The Sall Mountain Asbestos Company, of New York (operations in Georgia), and the Lowell Lumber and Asbestos Company, Lowell, Vt., were the only producers in the United States. The latter closed its plant after operating a few months in 1908. The only plants in the United States which are prepared to produce finished asbestos are those of the two companies mentioned. The plant of the American Asbestos Company, at Bedford City, Vt., has not been operated for two years.

ASBESTOS STATISTICS OF THE UNITED STATES.

Year.	Production.			Imports.		
	Short Tons.	Value.	Value per Ton.	Manufactured.	Unmanufactured.	Total.
1897	840	$ 12,950	$15.42	$10,570	$264,220	$ 274,290
1898	885	13,425	15.17	12,899	287,636	300,535
1899	912	13,860	15.20	8,946	303,119	312,068
1900	1,100	16,500	15.00	24,155	331,796	355,951
1901	747	13,498	18.08	24,741	667,087	691,828
1902	1,010	12,400	12.27	33,313	729,421	762,734
1903	(a) 887	(a) 16,760	(a) 18.90	32,058	657,269	689,327
1904	(a) 1,480	(a) 25,740	(a) 17.40	51,290	700,572	751,862
1905	3,100	126,300	40.74	70,117	776,362	846,479
1906	(a) 1,695	(a) 28,565	(a) 16.85	96,162	1,010,453	1,106,615
1907	950	11,700	12.32	200,371	1,104,110	1,304,481
1908	1,350	24,000	17.78	147,548	1,068,342	1,215,890

(a) Statistics of the United States Geological Survey.

The asbestos deposits on Caspar mountain, Natrona county, Wyoming, were developed in 1908 by the North American Asbestos Company. It is proposed to erect a mill in the spring of 1909. The asbestos is of the chrysotile variety and occurs in serpentine, in granite and schist. The deposit as proved is about 200 ft. wide. The North American company is also developing prospects at Smith creek and Deer creek, near Caspar mountain.

Vermont. (By G. H. Perkins.)—Asbestos is widely distributed over the State of Vermont, especially the northern portion. It has been mined only at Belvidere mountain where several openings have been

made. At present those on the south side of the mountain are not worked, but on the north side there are several openings worked by the Lowell Lumber and Asbestos Company and a finely equipped mill has been built. The machinery here is much of it of peculiar design and separates the fiber from the crushed rock by air blast. The present capacity is 50 tons of rock per day from which 10 tons of separated fiber are obtained. The material is chrysotile and much of it is of excellent quality, but the fiber is very short. Four grades are separated.

Prices.—The prices for crude No. 1, crude No. 2, crude and paper stock asbestos from 1899 up to the end of 1908 are shown in the accompanying chart.

Asbestos in Foreign Countries.

Africa.—The most important deposits of asbestos are found in the Carolina district, Cape Colony. According to the report of the Carolina Asbestos Company, Ltd., for the year ending Sept. 30, 1908, of the asbestos recovered, 62 per cent. was over 1 in. in length; for the last six months of the year, however, the material over 1 in. averaged only 40 per cent., which is approximately what may be expected under present conditions of working. The fluff obtained by sieving the free fiber out of the fines constituted 14 per cent. of the year's output; the proportion during the last three months of the year, however, was 38 per cent. The proportion of fluff is likely to increase in the future, for in the stopes furthest into the hills, the fiber comes away from the serpentine more readily than in the outside workings; also a much larger proportion is set free when the rock is blasted. Blasting increases the quantity of fiber in the fines and decreases the quantity of lump to be cobbed. It is to be noted that the easier the fiber comes away from the rock, the better is the quality of the asbestos. On the other hand, when, as at present, all the recovering of asbestos and freeing it from intermingled particles of rock is done by hand, much more labor is required to obtain a ton of clean asbestos from the free fiber in the fines by sieving, than is required to obtain the same weight by cobbing from the lumps. The approximate ratio of labor required is as 4 : 3. Undoubtedly the fines could be treated more cheaply, and with a greater recovery of fiber by the use of mechanical sorting appliances.

Although one or two barren patches have been met with in the mines, this was only to be expected, as they occur in all serpentine asbestos-bearing formations. The railroad was opened to Carolina at the beginning of February, 1908. Material is transported to Carolina by wagon and railed from there. This reduced the transport expenses during the

last half of the financial year from £2 to £1 per 2240 lb. Transport contracts for the coming year were made at the rate of 18s. 4d. per ton.

Canada.—The utilization of short-fiber asbestos in the manufacture of boards has stimulated the opening of new mines. The East Broughton mines, which produce no long fiber, have consequently acquired some importance and new companies have been organized. Consul Paul Lang reports from Sherbrooke that asbestos has been discovered near Eastman,

TABLE OF PRICES OF CANADIAN ASBESTOS, 1899-1908.

Quebec, and that several companies have organized to develop the new mines. The Orford Mountain railroad is near these properties, and a spur will be built from the main line to the mills, which are already in course of construction. The mines near Danville, about 65 miles from this new find, and at Thetford Mines, Black Lake, and Broughton, and several smaller places on the line of the Quebec Central Railway, about 100 miles from Eastman, are now the only producers of this mineral in Canada.

The principal asbestos-producing districts in Quebec are the East Broughton and Black Lake. In the former, the Boston Asbestos Company

operated its new 250-ton mill, and the Frontenac Asbestos Company erected a new 300-ton mill. The Eastern Townships Asbestos Company produced from 12 to 15 tons of asbestos per day during most of 1908. The Broughton Asbestos Fiber Company was closed for four months, but is now prepared to produce on a large scale. About 60 men are employed. The Johnson Asbestos Company, of Thetford, completed its new mill.

The Canadian production of asbestos in 1908, based on reports from 12 operating companies, which employed 4284 men and paid in wages $1,066,774, was as follows: Crude, 3671 tons ($699,521); mill stock, 61,485 tons ($1,852,075); total asbestos, 65,156 tons ($2,551,596); asbestic and asbestic sand, 24,011 tons ($34,660); total products, 89,168 tons, valued at $2,577,302. For 1907 the total product amounted to 90,426 tons, valued at $2,505,043.

STATISTICS OF ASBESTOS IN CANADA. (a)
(In tons of 2000 lb.)

Year (b)	Production. (b)				Exports (c)		Imports (d)
	Asbestos.		Asbestic.				
	Short Tons.	Value.	Short Tons.	Value.	Short Tons.	Value.	Value.
1897	13,202	$ 399,528	17,240	$45,840	10,969	$ 510,916	$ 19,032
1898	16,124	475,131	7,661	16,066	18,424	510,368	26,389
1899	17,790	468,635	7,746	17,214	14,520	453,176	32,607
1900	21,621	729,886	7,520	18,545	18,164	490,900	43,455
1901	32,892	1,248,645	7,325	11,114	26,715	864,573	50,829
1902	30,219	1,126,688	10,197	21,631	33,072	1,131,202	52,464
1903	31,129	915,888	10,548	13,869	30,661	955,405	75,465
1904	35,635	1,167,238	13,011	13,006	34,636	984,836	83,827
1905	50,670	1,486,359	17,594	16,900	41,127	1,311,524	116,836
1906	59,283	1,970,878	20,127	17,230	59,864	1,689,257	138,000
1907	62,018	2,482,984	28,519	22,059	56,753	1,669,299	200,371
1908 (e)	65,157	2,551,596	24,011	34,660	59,051	1,730,755	191,204

(a) From *Annual Reports* of the Geological Survey of Canada, and the *Statistical Year Book* of Canada. (b) Production is given for calendar year; exports and imports are for fiscal years ending June 30 up to and including 1907. In 1908 the fiscal year ended March 31. (c) Mainly crude asbestos. (d) Manufactured articles entirely. (e) From report of Superintendent of Mines for Quebec.

Russia.—The principal deposits of asbestos are found in the Ural mountains. The more important lie along the Sysert river, 30 versts from the Sysertski works on Asbestos mountain. The chief deposit is near the station of Bashenovo. The companies operating here, with their approximate production in poods per annum, are as follows: Poklevsky-Koziell, 200,000; Baron Girardeau-de-Sukanton, 150,000; Korevo, 130,000; Kreutzer & Devallet, 15,000; Baron Kusov, 1000. According to a Russian geologist, the deposits cover a length of about nine versts and are about 600 sajens wide. The thickness has not been determined, as operations are confined almost entirely to the surface. The asbestos fiber separates easily from the gangue. The mineral is distributed throughout the ser-

pentine, the rock carrying approximately from 2 to 3 per cent. of asbestos.

The following is the production of asbestos in the Urals since 1899: 1899, 164,430 poods; 1900, 234,756; 1901, 268,537; 1902, 275,183; 1906, 489,680; 1907, 571,194.

The market for Russian asbestos is in Europe, where the output is sold in advance. The best quality of Russian asbestos for some time brought about 3.20 rubles; second quality, 1.50 rubles; third quality, from 1.20 to 1.50 rubles per pood (1 ruble=51.7c.; 1 pood=36.097 lb.). The present prices are much higher than this. These figures refer only to small lots; it is impossible to obtain large quantities, as all the large companies contract their output ahead.

It has lately been reported that a German syndicate has purchased the principal deposits of asbestos in the Urals. The Korieff mine, the Poklevsk mines and the mines of Baron de Sukanton have passed into German hands. The Poklevsk property includes the only mill operated in the whole region.

Other Countries.—In Vol. XVI of THE MINERAL INDUSTRY, we made mention of asbestos operations in Australia, the Island of Cyprus, India, Italy and the Philippine Islands. We have no reports of production from these sources. In Belgium the asbestos industry is beginning to assume prominent proportions. The principal factories are at Antwerp and Auvelais. In France small deposits of asbestos are worked in the Pyrenees, in Dauphiny and on the Island of Corsica.

ASPHALTUM.

The asphaltum industry is an extremely difficult one to report statistically. Besides the natural products, there is a very large output of asphaltum residue obtained in the treatment of petroleum in California; also in the treatment of other petroleums by the Standard Oil Company. This residual asphaltum is of much more importance than the natural products, at least in so far as the United States is concerned. Statistics of production as reported by the U. S. Geological Survey are given in the accompanying table.

PRODUCTION OF ASPHALTUM AND BITUMINOUS ROCK IN THE UNITED STATES.
(Tons of 2000 lb.)

States.	1905			1906 (a)			1907 (a)		
	Tons.	Value.	Per Ton	Tons.	Value.	Per Ton	Tons.	Value.	Per Ton
Bituminous Sandstone.									
California	22,500	$63,000	$2.80	20,418	$47,427	$2.32	34,531	$84,016	$2.43
Kentucky	7,530	34,885	4.63	1,629	7,330	4.50	6,993	33,397	4.77
Oklahoma (c)	(e)5,000	11,500	2.30	738	2,029	2.75	4,002	11,627	2.90
Arkansas	(e)1,500	7,500	5.00	900	5,400	6.00	.....	.....	.....
Georgia (f)	.....	.....	.....	400	8,500	2.13	.....	.....	.....
Total	36,530	$116,885	(g)$3.20	24,085	$70,686	(g)$2.93	45,526	$129,040	(g)$2.83
Asphaltic Limestone	6,029	42,000	6.96	*Nil*	.....	.....	*Nil*	.....	.....
Asphaltum (b)									
California	57,687	545,503	9.46	71,539	711,150	9.94	96,537	1,129,941	11.70
Oklahoma (c)	*Nil*	.....	.....	*Nil*	.....	.....	70	1,400	20.00
Missouri	(e)1,000	17,500	17.50	*Nil*	.....	.....	(i)4,650	42,500	9.15
Texas	113,500	851,250	7.50	24,993	307,952	12.32	53,649	929,857	17.33
Total Asphaltum	172,187	$1,414,253	(g)$8.20	96,532	$1,019,102	(g)$10.55	154,906	$2,103,698	(g)$13.58
Gilsonite (d)									
Utah	10,516	110,144	10.47	12,947	159,600	12.33	20,719	569,440	(h)27.48
Oklahoma (c)	(e)1,000	25,000	25.00	1,952	16,432	8.42	966	7,743	8.03
Mastic									
California	*Nil*	.....	.....	*Nil.*	.....	.....	*Nil.*	.....	.....
Kentucky	*Nil.*	.....	.....	2,543	24,158	9.50	1,744	16,568	9.50
Pennsylvania	2,000	18,000	9.00	*Nil.*	.....	.....	.....	.....	.....

(a) From the Mineral Resources of the United States. (b) Includes hard and refined, or gum, liquid or maltha, and oil residues. (c) Indian Territory included before amalgamation into one State. (d) Includes gilsonite, elaterite, grahamite, osokerite and "tabbyite." (e) Estimated. (f) First reported separately in 1906. (g) Average value per ton. (h) The average value of the gilsonite alone was only $26.22 per ton. (i) Mostly from Kentucky.

As stated in THE MINERAL INDUSTRY last year, the merits of oil asphalt as a paving material have been acknowledged, the residual asphalts derived from California and Texas petroleums being now employed extensively in the preparation of pavements. The United States, so far as now known, possesses no large deposits of natural asphalt comparable to those of Trinidad and Venezuela.

The Asphaltum Industry.

By T. Hugh Boorman.

The most prominent feature in the asphalt industry in 1908 was the increase in its use for rendering roads dustless and more durable under automobile traffic. The necessity for new methods of surfacing roads was so apparent that an international congress was held in Paris in October, 1908, at which the attendance of 800 delegates was anticipated; the opening session discovered the presence of nearly 2000 representatives from nations interested in the problem. The following countries sent delegates: Austria, Belgium, Brazil, Bulgaria, Chile, China, Cuba, Denmark, France, Germany, Great Britain, Greece, Holland, Hungary, Italy, Japan, Luxemburg, Mexico, Monaco, Norway, Persia, Portugal, Roumania, Russia, Siam, Spain, Sweden, Switzerland, U. S. of America, and Uruguay. The conclusions were not definite, but the establishment of a permanent organization was perfected, under the name of the International Highway Commission, with headquarters in Paris, and future investigations and results of experimental road building will be widely diffused among the nations. It was also decided to hold a second congress in Brussels in 1910.

That asphalt, liquid asphalt, and asphalt rock will be important factors in road building seems to be an accepted fact, and American asphalt will be in demand to an extent that may increase the demand for its production equaling that for American portland cement. It has been impossible to collect the statistics of the actual production of American asphalt in 1908, as for some unexplainable reason, refineries do not seem willing to give the information. It is probable that one corporation, hitherto not recognized as a factor in the asphalt industry, has produced not less than 50,000 tons of asphalt refined from heavy oil with an asphalt base.

Kentucky rock asphalt has been rolled on the surface of macadam roads in several States where the freight rates have made the cost of such construction allowable. Asphalt concrete has been laid extensively and has caused a large consumption of California, Texas, and Kansas petroleum asphalts.

During 1908 a refinery for the production of asphalt from Texas material was in full operation at Curtiss Bay, Baltimore, operated by the Ellis Company. No new refineries are reported as having been erected in California. The Uvalde rock asphalt mines at Cline, Texas, were not operated during 1908.

The question of tariff revision during the latter part of 1908 brought out the statement that there were 27 concerns producing, refining, and marketing California asphalt in 1907, the production being 98,000 tons;

in 1908 it was probably somewhat less. The present cost of production of California asphalt is stated by James R. Weeks, of the California Asphaltum-Sales Agency of San Francisco, as follows: Crude material, $3.50 per ton; refining and fuel, $1.50; barrels, $2.50; transportation to New York, $10; marketing, $1.50; total, $19 f.o.b., New York. In a letter to the sub-committee on tariff revision by Col. J. W. Howard was the following interesting statement: "Domestic asphalt or its equivalent, asphalt cement, has displaced during the last 10 years one-half of the importations, and under a modified tariff, will continue, under a healthy competition between domestic asphalts, to replace practically all imported asphalts."

An illustration of the present healthy competition is found in the several powerful groups and many small producers which compete with each other at home, while several export from the United States, as follows:

(A) A California group of about 27 competing producers who have a special $10 per ton freight rate to Atlantic seaboard cities, with proportional reductions to interior points.

(B) A Trinidad group which leases a deposit in Trinidad, West Indies, to the New Trinidad Lake Asphalt Company, which together with the Barber Asphalt Paving Company is controlled by the General Asphalt Company. The latter company, directly or indirectly, produces asphalt in California and other States. It is possibly the strongest concern and does not depend entirely upon either foreign or domestic asphalts. It uses its products in cities between the Atlantic and Pacific coasts.

(C) A Venezuela group which may be regarded as composed of the A. L. Barber Asphalt Company, importing Bermudez asphalt from Venezuela and selling it to members of the Independent Asphalt Association and others. This group includes 25 or more contractors and purchasers. Cost of production and especially freight rates from New York to the West, where most asphalt is used, prevents Bermudez, or Venezuela asphalt from more than a feeble competition with the domestic product.

(D) A Texas group consisting of several producers such as the Texas Company, the Sun Company, the Ellis Company and others, producing asphalts which compete with each other and with all others from elsewhere. The Texas asphalts are extensively used for many purposes and are sold at lower rates to consumers than asphalts from any other source and are delivered at cities not only on the Atlantic coast, but also in the middle west.

(E) A Kansas-Indian Territory group of producers has a large refinery in Kansas and competes, with its products of various asphalts or bitumens, over a large area as far east as New York. This is the newest source of supply.

(F) A Utah group of producers competes within itself and with others. It has long supplied special high-grade asphalt for various uses and because of its peculiar and excellent quality it is sold for purposes other than paving and for the highest prices and it is even exported.

(G) Miscellaneous producers of asphalt and its natural and prepared compounds are situated in Kentucky and elsewhere in the United States. Although, individually, many are small, their aggregate production is large.

California produces refined asphalt, f.o.b. works, more cheaply than in any other part of the world. This is due to the enormous amount of asphalt-petroleum found over large areas of that State, which with little labor, and the use of some of the oil itself as fuel, is made into refined asphalt. The problem of California asphalts is entirely one of freight. In fact this applies to almost all the domestic production of crude and refined asphalt.

Asphalt Mastic.—The duty of $3 per ton on rock asphalt mastic has stimulated the manufacture of this class of material in the United States. Crude rock asphalt from Germany, Sicily, and other European countries, paying a duty of only 50c. per ton, has been imported in large quantities and manufactured into mastic blocks at works in New York, Brooklyn, and Maurer, N. J. At Bowling Green, Ky., mastic is made from asphaltic sandstone, and a plant will be shortly in operation near Philadelphia for the purpose of making an artificial asphalt mastic with similar properties, it is claimed, to those of the Neuchatel asphalt mastic.

Asphalt for Roofing.—The increasing demand for prepared roofings has called for heavy supplies of asphalt. Most of the manufacturers of the higher grades of such material have special methods of blending different asphalts with a small quantity of "secret" ingredients. The feature of such processes is the mixing of some hard asphalt, of the nature of gilsonite, with heavy Texas asphaltic oil. Cuban asphalt is frequently used to give a heavy body to the surfacing, but as the deposits of this asphalt are of irregular form, and one of the best-known mines, the Bejucal, has already played out, other more abundant varieties must be looked to for supplying the demand for such material.

Asphalt Production in the United States.

Arkansas. (By A. H. Purdue.)—The asphalt deposits of Arkansas occur in the Trinity sands near the Cretaceous-Paleozoic contact. The only place where they have been quarried is in Pike county, about two miles southeast of Pike City. At this place the asphalt occurs irregularly distributed through incoherent sand overlain by sand and clay, and is 10 to 12 ft. beneath the land surface. None of the material was quarried in 1908.

California.—While statistics are not obtainable the production of liquid asphalt in 1908 must have exceeded that of 1907 which was estimated at 110,000 tons. The refineries at Oleum and West Berkeley, near San Francisco, were both running throughout the year. The Bakersfield and Sunset refineries were reported as having their full quota of business, the Obispo claims an output of 40,000 tons and the Los Angeles refineries accumulated no stock. Many of the refineries, however, were utilized to some extent in the preparation of road oil for which the local calls were heavy; in fact the demand for both asphaltum, and asphalt oil for home use in California has been steadily growing and the California producers have also to supply the needs of the fast-growing cities of Tacoma, Spokane, and Seattle.

The following is an abstract of an article by A. S. and H. N. Cooper of the University of California, giving an account of their research work in which a new constituent of asphaltum has been discovered:

In asphaltum there are three principles sufficiently distinct to be easily separated by several known solvents, but they do not exist in definite proportions. They are: (1) A yellow resin, sometimes brown, solvent in absolute alcohol. (2) A brownish-black resin, soluble in ether and insoluble in alcohol. These two taken together are known as petrolene. (3) A black substance having a conchoidal fracture, dissolving neither in alcohol nor in ether, but perfectly soluble in chloroform, carbon bisulphide and spirits of turpentine. This principle is known as asphaltene. The fourth principle discovered, which is not soluble in any known solvents of petrolene or asphalten has been named crockerite.

Crockerite exists in vast quantities throughout the rocks in the oil fields of California, and is a good indication of the presence of petroleum. It may exist without petrolene or asphaltene being present. It may also be absent from asphaltum. Crockerite is absolutely insoluble in glymol (which is a clear odorless oil useful for detecting minute traces of difficultly soluble asphaltum), and it is not attacked by cold strong acids, but it is slightly soluble in cold fuming nitric, and completely soluble in boiling fuming nitric. When the excess of nitric acid is evaporated the product has an aromatic odor, not unpleasant. When two pieces are briskly struck together they emit an odor like burnt rubber. The coloring matter of numerous black or dark-colored rocks is crockerite. The laminae of these shales are rendered very distinct by the bituminization of some and the less bituminization of others.

Kansas.—The plant at Independence was busy, as the Sarco brand of blended asphalts was extensively used during 1908.

Kentucky.—The exploitation of Kentucky rock asphalt seems to have been confined to the operations of the Wadsworth Stone and Paving

Company, of Pittsburg, which shipped both rock for the surfacing of macadam roads and asphalt mastic. The good roads movement has also greatly increased the business of the Indian Refining Company at Georgetown, which uses Kentucky liquid asphalt in its road construction and sells it to municipalities throughout the East.

Oklahoma. (By Charles N. Gould.)—Asphalt is found extensively over all the southern part of Oklahoma, but chiefly in the region south of the Arbuckle mountains. The greater part is found as rock asphalt, or rock impregnated with asphalt, although in some places, as near Atoka and Cornish, it is found in an almost pure, semiviscous state. Pure bitumen is derived from distillation of petroleum.

The asphalt in Oklahoma occurs along fault planes. The faults are often of many thousands of feet throw and have cut beds which carried the heavy petroleum, which has impregnated the rock along the fissure. The asphalt is called lime-asphalt or sand-asphalt according as the stone along the fault is limestone or sandstone. Often, in Oklahoma, beds of sand asphalt and lime asphalt occur side by side.

The asphalt varies considerably in composition as shown by the analyses made by Professor De Barr of the State University of Oklahoma. The lime asphalt contains all the way from 2 to 10 per cent. of hydrocarbon, the remaining being calcium carbonate, and the sand asphalt contains from 10 to 20 per cent. hydrocarbon. Shale asphalt is rarely utilized.

The so-called asphalt veins vary in width on either side of the fault line from 2 to more than 50 ft., according to porosity of the rock. New deposits are continually being found in Oklahoma, and much of the material is now being utilized. According to the report of the Chief Mine Inspector, the production of asphalt in the 11½ months ending Oct. 31, 1908, was 5946 tons, which sold for an average of $12 per ton.

Texas.—It is difficult to define the quantity of asphalt refined by the Gulf Refining Company, the Texas Company, and the Sun Company, as oil and asphalt are so closely connected. These companies sell large quantities of material ranging from crude petroleum to the high-melting-point asphalt for varnish, so that the production is apt to be in two lists—petroleums and asphaltum—and their sales of oils go into the manufacture of so many "blended" asphalts that statistics may be misleading.

Utah.—The output of gilsonite probably exceeded 10,000 tons; in addition to the operations of the General Asphalt Company, the American Asphaltum and Rubber Company, the Standard Asphalt and Rubber Company, and the Raven Mining Company, all of Chicago, mined large quantities during 1908.

Asphalt Mining in Foreign Countries.

Descriptions of asphalts have been given in former volumes so fully that it is not necessary here to reiterate them. The following table prepared by Col. J. W. Howard shows most of the countries producing asphalt, but it does not include the deposit now being exploited in Nigeria (West Africa) recently reported in the *Engineering and Mining Journal.*

CLASSIFICATION OF IMPORTANT BITUMENS AND COMPOUNDS

Glance-Pitch, Asphaltum, Maltha.		Asphalt and Bituminous Rock.	
Pure or nearly pure (with small per cent. of mineral and organic matter).	Compounded with earthy matter (large per cent. of silicates, alumina, etc.).	Compounded with limestone (containing silicates, etc.).	Compounded with sandstone (containing carbonate of lime, etc.).
Argentina.	Argentina.	Austria.	France.
Barbadoes	Australia.	Cuba.	Germany.
China.	Barbadoes	France.	Italy.
Cuba.	China.	Germany.	Russia.
Egypt.	Colombia.	Hungary.	Sicily.
Equador.	Cuba.	Italy.	Spain.
Honduras.	Egypt.	Russia.	Turkey.
Japan.	Equador.	Sicily.	United States.
Mexico.	France.	Spain.	
Russia.	Germany.	Switzerland.	
Syria.	Honduras.	Turkey.	
Turkey.	Japan.	United States.	
United States.	Mexico.		
Venezuela.	Peru.		
	Russia.		
	Syria.		
	Trinidad.		
	Turkey.		
	United States.		
	Venezuela.		

Sicily.—The importations of crude sicilian rock asphalt in 1908 amounted to about 5000 tons from the United Limmer Asphalt Company's Mines at Ragusa. Formerly shipments to American ports were made by the Val de Travers Asphalt Paving Company, of London, and the Sicilian Natural Asphalt Company, of Palermo. Its use is now confined, in the United States, to the manufacture of mastic. Shipments of this product manufactured by H. & A. B. Aveline & Co. of Catania have been debarred recently by the heavy duty.

Trinidad.—Mining for manjak was carried on almost continuously during the year at Vistabella in Naparima. The importations of asphalt into the United States from Jan. 1 to Sept. 15, consisted of 60,794 tons of pitch lake asphalt, 2879 tons of "epurè"[1] (dried) and 2921 tons of land asphalt.

Venezuela.—The importations of Bermudez lake asphalt in 1908 were 35,189 tons. The troubles in regard to concessions there have been ventilated in the daily press. The matter of consignee of the pitch lake will

[1] This is refined by crude process.

probably little affect the amount of output. The change of government may however lead to renewed importation from Maracaibo; probably also larger exports may be made by the English and other concessionaires of Guanipa and smaller properties.

The Manjâk (Glance Pitch) Deposits of Barbados, W. I.

By R. H. Emtage.

Manjâk occurs in veins similar to other solid hydro-carbons, e.g., the albertite deposits of New Brunswick, grahamite of West Virginia, and gilsonite of Utah and Colorado. The formation in which the veins occur ranges from the Cretaceous to the late Tertiary in what is locally known as the "Scotland Series" (so-called by an early historian because the hills in the district recalled his native landscape), through which they are widely dispersed. This series is exposed over an area of about 25 square miles, and small exposures also occur outside that area within a mile or two of the coast, where the coral limestone capping is thin. This capping covers the remaining 140 square miles, which therefore presents a more level aspect, and attains a thickness of 250 ft. in parts toward the center of the island. Although the coral covered area attains an altitude of 1000 ft. toward the northeast end of the island, yet, as this altitude is reached by a series of gently sloping terraces from west and northwest, the general impression conveyed to the casual observer, approaching the island from that side, is one of comparative flatness.

The principal rocks comprising the "Scotland Series" are argillaceous sandstones, interspersed with clays, and in places highly calcareous, showing thin bands of interbedded limestone. Although exposed only over an area comprising about one-seventh of the island, this series is certainly co-extensive with the island, and has been frequently exposed at the base of the coral limestone at the bottom of wells sunk for water. An intermediate series of Oceanic ooze also occurs in many parts of the island, and is often visible overlying the sandstones throughout the "Scotland" district where the coral rock is absent, or only cropping up occasionally as boulders. This Oceanic ooze attains an extreme thickness of 300 ft. at Mount Hilloby, the highest point (1104 ft.) of the island, and probably, at one time, practically covered the "Scotland" area, but is now largely displaced by denudation aided by surface tillage.

The district shows considerable faulting, which often extends for long distances across the country. The longest of these faults may be traced from the coast in a southwest direction inland for a distance of 3½ miles where it abuts against the ridge of coral rock. Much local folding is also in evidence, and easy of identification along the hill-sides and water-courses, whose banks are peculiarly free of bush, owing to the high state

of cultivation throughout the island. This faulting, and perhaps, more particularly this "folding" is largely responsible for the manjâk, which is merely the petroleum from the richer oil sands forced up into the cracks produced by the folds, and solidified, after the evaporation of its more volatile constituents. The iridescent appearance of many of the outcrops, especially where they lie in contact with a surface oil sand, clearly indicates the origin of manjâk. And if further evidence on this point were needed it may be had at one of the mines in what is, so far proved, the richest oil area of the island. Here a vein disappeared altogether at a depth of 175 ft. by gradual stages, giving place to an oil sand, which consistently improved as the manjâk vein diminished, and which yields a couple of barrels of heavy asphaltic oil per week at that depth. At this mine the gradual transition of the hydrocarbon from the liquid to the solid state can be clearly traced—the dark, heavy oil of the consistency of molasses becoming by slow degrees a dull black, plastic wax, which in turn changes, after long exposure, to a lustrous, crystalline, jet-like, substance. This is the only mine at which this transition can be so clearly traced, and it is the one at which the oil sands are most in evidence—a well drilled eight years ago about half a mile northeast of this mine struck oil at a depth of 800 ft., which actually bubbled to the surface, but since then, for some reason not yet explained no further drilling has been done in that district. That the district offers reasonable prospects to bona-fide oil prospecting there can be little doubt. More or less oil saturation at varying depths has been detected in the mines of other localities, although there is no definite evidence of the effect of such saturation on vein values. I am of the opinion that this will depend on the position of the veins in relation to the larger faults of the district.

Mode of Occurrence.—The outcrops of the manjâk veins are marked by very variable characteristics, sometimes following the line of stratification for some distance, sometimes forming a net-work of stringers in a hard oil sand, and again cutting obliquely across the stratification in a well-defined, steeply-pitching vein enclosed in a bold selvage of plastic clay. These last generally indicate the more permanent veins, and show in less degree the lenticular structure, which however is everywhere evident. Those veins which conform to the stratification at the outcrop either soon pinch out, or else pitch steeply regardless of stratification. The outcrops are chiefly detected in the many local anticlinal arches which traverse the district at various points or along the edges of steeply tilted sandstones. The veins are pockety, and of variable width according to the nature of the ground traversed; a vein which runs 3 or 4 ft. in a soft, arenaceous schist will soon pinch to as many inches when meeting with a compact oil sand, or may split up into a series of small string-

ers in the presence of a slaty, blue-gray, oil shale which is a marked feature of the hanging wall in one locality. This wall is strongly and frequently slicken-sided, as may be expected from the many local faults and the frequent folding experienced. Nowhere has any well-defined foot wall been observed within the shallow depths yet reached. In two places, widely apart, in which clay and chalk (the Oceanic ooze referred to) and much water predominate, it was found that below the 200-ft. level the vein structure apparently ceased, giving place to a series of pockets, entirely disconnected, so far as could be traced, although the slicken-sided appearance of the contiguous ground suggests the possibility of the inter-passage of oil. These pockets, which varied in size from a couple of tons to a few hundred tons resembled in their mode of occurrence those of the solid dull-looking bitumen of Albania, described by M. Gounot in *Annales des Mines,* Vol. IV, Part 7, of 1903. Unlike the Albanian deposits, it is the highest-grade (in luster, intensity of color, and purity) bitumen of Barbados which occurs in this disconnected pockety manner. The solid, brighter bitumen of Albania occurs in parallel veins, similarly to the typical Barbados manjâk.

No great depth has yet been attained at any of the mines of Barbados, and it is therefore premature to suggest what may be the possibilities with development; but there seems to be a tendency for the more permanent veins to branch out into more or less parallel veins below the 200-ft. level, thus producing a series of blind veins often showing no connection with the surface beyond a meager outcrop. The general trend of these veins, so far as prospecting has yet proved, is approximately north-northeast and the dip is anything between 60 and 85 deg., and ranges, so far as my experience goes, between north and northwest. The prevailing dip of the country rock in the vicinity of these mines is north, and the opinion holds ground with geologists that the thrust which caused the up-throw of these rocks was exerted from the west. It will no doubt be found, as development proceeds, that the dip of the several veins is influenced by the strike of the series of local anticlines in the immediate vicinity.

It is not unusual for these branch veins to develop much larger than the main vein for some distance, and at intervals a group of them will develop pockets of several hundred tons in conjunction with the main vein. The shattered and fissured condition of the veins resembles the albertite occurrence of New Brunswick described by Dr. R. W. Ells of the Canadian Geological Survey rather than the more uniform structure of the Utah-Colorado gilsonite, of which one vein is reported as attaining a thickness in places of 8 to 16 ft. with "no admixture of rock." The occurrence of "horses" is very frequent in the Barbados deposits, and it is seldom that a greater thickness than 4 ft. occurs anywhere above

the 250-ft. level. But, unlike the albertite deposits, which were unworkable above 450 ft., these deposits are sometimes sufficiently productive to be workable within 10 ft. of the surface, and generally within 40 ft., where weathering has not affected the quality. This, of course, is due to the higher prices which the high grades of Barbados bitumen command. And the experience of at least one locality is that the vein-value above the 200-ft. level affords no basis for estimating the quantity to be found below that level, for at that depth a series of branch veins are found which considerably extend the area of workable ground. But enough development has not yet been done throughout the manjâk district for me to suggest whether this rule is likely to hold good for other localities, nor to what extent this process develops below that level.

Early History and Methods of Working.—The existence of manjâk in the island had long been known; indeed, the following official reference dated 1676 by the then Governor is on record: "There is a kind of metal, much resembling cannel coal in Lancashire, called *Moniack* (the "o" no doubt a printer's error in the early record; the "c" is retained by some to the present time.—R.H.E.) with which sugar is boiled." But although the surface outcrops had frequently been dug and used as fuel under the several plantation furnaces, yet no systematic attempt had ever been made to mine it, until it was brought to the notice of certain Boston men in 1895. They were evidently attracted by its high calorific properties, and forming themselves into the Boston Fuel and Power Company, sent out a representative to report. He succeeded in interesting the late Walter Merivale, then manager of the local railway company, who obtained a concession and started mining at the east end of the island at a spot marked "Bituminous coal" on an old map of 1848. Never allowing grass to grow under his feet, Mr. Merivale, before he had reached a depth of 60 ft., or knew anything as to probable yield, cost of production, or much else about his mine, promptly signed a contract with this company to deliver 5000 tons of manjâk within five years at $7 per ton delivered at Bridgetown. Luckily for him the company was not prepared to receive his early shipments as promptly as he made them, and was no less anxious than he to cancel the contract a few weeks after signing it. The bitumen from this mine, which more nearly resembled the Syrian variety that in 1905 commanded £37 ($185) in Europe, fetched from $120 to $140 per ton in New York a year ago. The pockety nature of this deposit always rendered this mine an uncertain quantity, and it ceased work in March, 1907.

Since the exploitation of the deposits was first started 12 years ago no great change has taken place in the method of working. Crosscuts are driven at frequent intervals to the veins from small vertical shafts.

In the pockety deposits, where a permanent vein is absent, the cost of prospecting usually exceeds that of productive mining, which is always a very variable quantity, being high or low according as the pockets are small or large, and the ground wet or dry. All work is by pick and shovel, and hand windlasses suffice for hoisting where the shafts do not exceed 120 ft. Below that depth it is usually economical to install a small steam hoist. In a few places where the ground proved wet the steam hoist came into operation earlier, but where the pockety formation makes the question of future development uncertain, its installation may be considerably delayed; and a series of payable pockets were worked out to the 180-ft. level over a period of two years entirely by manual hoisting at no very excessive cost in that item.

The main difficulty encountered in the mines is due to excessive side pressure from a weak foot wall. My experience has been that the ground of the pockets, when not wet, is stronger than that of the fissure veins, and that where one or more branch veins meet the main vein and form a local pocket the ground is usually weakest at that point. This is no doubt what might be expected from the nature of the cause of the fissure; and careful timbering must be practised in order to avoid waste of the brittle mineral. The smaller the percentage of dust the better.

In the process of shaft sinking much gas is met with in contact with the oil shales or sands, and below the 200-ft. level the safety light of the pick-man has more than once been put out by such "blowers." At one shaft on reaching 260-ft. level mud was periodically thrown up for several weeks with sufficient force to spatter timber 6 ft. from the bottom of the shaft. A fine white "gas sand" is also encountered in certain localities, usually in lenticular masses similar to the manjâk deposits of those places, and sometimes within 25 ft. of the surface. This increase of gas with depth speaks well for the continuance of the oil sands, which everywhere out-crop in the district, but which, owing to the absence of systematic drilling, have been nowhere properly tested. That the entire district warrants careful oil prospecting is the opinion of the few American geologists who have recently visited it. The main superiority of manjâk over gilsonite for the varnish and japanning trades is its intense black color, in which respect it more nearly resembles nigrite, which is no longer mined. The following is a comparative analysis of Barbados manjâk, and gilsonite of Middle Park, Colo., the latter taken from *Mines and Minerals* of some few years ago.

	H_2O	Ash	C	Vol. mat.	H	S	O+N	Total	Sp.Gr.
Manjak	2.49	2.70	83.62	.....	8.29	0.85	2.05	100	1.098
Gilsonite	1.00	4.50	59.00	35.50	.....	.....	.....	100	1.145

BARYTES.

By Albert H. Fay.

The commercial depression of 1908 had a demoralizing effect on the barytes industry. Foreign competition also was a factor in the decreased production of the United States.

STATISTICS OF BARYTES IN THE UNITED STATES.

(In tons of 2000 lb.)

Year.	Production.			Imports.				Consumption.	
		Value.		Crude.		Manufactured.			
	Short Tons	Per Ton.	Total.	Sh. Tons.	Value.	Sh. Tons.	Value.	Sh. Tons.	Value.
1897.....	26,430	$4.00	$105,720	502	$ 579	1,300	$13,822	28,232	$120,121
1898.....	28,247	4.00	112,988	1,022	2,678	687	8,678	29,956	124,344
1899.....	32,636	4.20	137,071	1,739	5,488	2,111	22,919	36,486	165,478
1900.....	41,466	3.90	161,717	2,568	8,301	2,454	24,160	46,488	194,178
1901.....	49,070	3.22	157,844	3,150	12,380	2,454	27,062	54,674	197,286
1902.....	58,149	3.21	186,713	3,929	14,322	3,908	37,389	65,986	238,424
1903.....	(a)50,397	3.02	152,150	6,344	22,777	5,716	48,726	62,457	223,653
1904.....	(a)65,727	2.66	174,958	6,689	27,463	5,920	48,658	78,336	251,070
1905.....	53,252	3.68	196,041	7,879	36,796	4,827	39,803	65,418	272,649
1906.....	63,486	3.98	252,719	9,189	27,584	4,808	37,296	77,483	317,599
1907.....	65,579	3.83	251,308	18,344	77,683	10,006	96,542	93,929	425,533
1908.....	38,546	3.38	130,409	12,197	58,822	3,037	29,168	53,780	218,399

(a) Statistics of the U. S. Geological Survey.

Industrial Conditions and Prices.—The unusual activity in the paint industry in 1907 resulted in several new mills being built; these had scarcely started operations when the panic caused many of them to close down. During 1908 the paint business was dull and the barytes industry suffered proportionally. Quotations for the first 10 months were: American ground, $14@21; floated, $22; foreign floated, $19.50@22.50 per short ton, f.o.b. New York. In October the price for domestic fell $2@3 per ton. The average price for domestic ground was $17.06 per long ton.

The accompanying table gives the comparative costs of domestic and foreign crude barytes, f.o.b. New York, compiled from the tariff hearings before the Committee on Ways and Means, of the House of Representatives, Nov., 1908.

The barytes industry in the United States occupies an unfortunate position, inasmuch as the deposits of the mineral occur at so great distances from the manufacturing centers on the Atlantic seaboard. Under

these conditions, the Eastern paint manufacturers are able to import the crude material from Germany and England at a lower price than the domestic product brings in New York. However, the cost prices given in the accompanying table, in some cases at least, seem extraordinarily high,

COMPARATIVE COST OF DOMESTIC AND FOREIGN BARYTES.

Authority.	Domestic.			Foreign.		
	Cost at railroad	Freight.	Total cost f.o.b. N.Y.	Cost at seaport	Ocean freight.	Total cost f.o.b. N.Y.
W. D. Gilman Co., Tenn.	.....	$5.25	.....	.....	.....	$6.50
R. B. Doherty, Tenn.	$3.00	5.25	$8.25	.....	.....	6.62
Kentucky Barytes Co., Ky.	5.50	4.25	9.75	$3.74	$1.00	4.74
M. E. Rhodes, St. Louis.	5.00	4.60	9.60	.....	.....	6.00
Carolina Barytes Co., N. C.	3.50	4.50	8.00	.....	.....	6.40
C. J. Staples, Buffalo, N. Y.	6.91	(a)4.65	11.56	3.74	1.00	4.74
Average, per ton.	$4.78	.....	$9.43	$3.74	.....	$5.83

(a) St. Louis.

especially in view of the fact that they are based on labor at 75c. to $1.50 per day. The producers of crude barytes are asking for a tariff of from $5 to $15 per ton, while the present duty is only 75c. per ton. Some of the producers of ground or floated barytes are demanding a duty of $12 per ton on the floated product. This they claim is necessary in order that they may compete with foreign producers.

Barytes Mining in The United States.

Georgia.—The only deposits that have been worked to any extent are at Cartersville. The mineral occurs in residual clay on the top of a quartzite formation, and is usually free from the limonite scale so prominent in the Tennessee deposits. The two companies operating in this district are Nulsen, Klein & Krausse Manufacturing Company, St. Louis, Mo., and John T. Williams & Son, Bristol, Tenn. The barytes shipped from this district in 1907 was approximately 2000 long tons. The output for 1908 did not exceed 500 tons. No new work was done during 1908. The Georgia Barytes Company of Asheville, N. C., was organized in 1908 to operate a mine at Elton, Murray county, Ga., and equipped itself to produce 3500 tons per year, but later went into the hands of a receiver.

Kentucky.—The Dix River Barytes Company at Danville worked throughout 1908 and shipped practically all of its crude barytes to New York. The grade of mineral produced by this company is high, and the plant is equipped for an output of 10,000 tons per year.

(By C. J. Norwood.)—The Nicholasville plant is operated by the Kentucky Barytes Company. It has a capacity of 50 tons per day, is situated

in the heart of the barytes section of Kentucky, and has excellent facilities for shipping the crude and finished product. This company, owning its own mines, is independent of outside sources for its supply of mineral.

The barytes deposits of this State have been but recently developed, and the indications are that Kentucky will soon be classed among the leading barytes-producing States of the Union. The chief deposits now operated are situated in Anderson, Bourbon, Clark, Fayette, Franklin, Gerrard, Harrison, Henry, Jessamine, Mercer, Owen, Scott and Woodford counties. F. J. Fohs, of the Kentucky Geological Survey, has now in hand the study and mapping of the deposits. With less than half of Fayette county alone investigated, 25 veins have been noted, some of which have been traced for more than three miles, nearly all of which have been prospected to some extent. Many veins are also known in the other counties, and some of them are being worked. The veins are known to extend down to a depth of 250 to 275 ft., in other words, from the top to the base of the Lexington formation (Trenton), and the width will average 3 ft. The barytes is exceptionally free from silica and iron.

Missouri.—More than 50 per cent. of the barytes produced in the United States during the last 25 years has come from Missouri, whereof Washington county produces at least three-fourths. The barytes from this county is of good quality and makes a splendid pigment. There was some mining done during 1908, but on account of the depression in the industry, there is reported to be at least 10,000 tons mined and at the railroad stations awaiting orders for shipment. Wages are 75c. to $1.25 per day. Royalty in most cases is $1 per ton. The mining is all open pit work; no expensive machinery is used.

New York.—The factory at Long Island City was operated during 1908 by Vivian, Bond & Co., of New York. The supply of crude mineral was obtained entirely from Kentucky. Late in 1908 the plant changed hands, and is idle at present. The Barium Production Company has a plant on Barren Island for grinding and bleaching barytes. The crude mineral is mined in Nova Scotia, and shipped by water to the factory.

North Carolina.—The majority of the barytes deposits in this State, and especially those along the French Broad river, occur as veins in fault zones from 2 to 8 ft. in thickness. Thus the mineral is found in place and systematic mining can be carried on. Many of these deposits are well up on the mountain slopes and can be worked to advantage from adit levels and should be capable of producing at low cost. At present the cost of delivery to the railroad station is $2.50@4 per long ton, which includes a royalty of from 35 to 50c. per ton. The cost of mining is from $1 to $1.75; haulage 75c. to $1.25 per ton, according to distance. Wages are $1.25 per day for mine laborers and $1.50 to $2 for foremen.

During 1908, the Carolina Barytes Company put in a tramway from the mine to the mill. From 75 to 100 men were employed, one-fourth of whom were used at the mill. Early in the year the mill was enlarged and an extra turbine installed to furnish power from the French Broad river.

Tennessee.—The majority of the mines in this State were closed down in the latter part of 1907, and only a few of the best properties were operated in 1908. Although labor is cheap, the cost of mining is high, for the reason that the deposits are pockety, uncertain and irregular in size, which prohibits the installation of steam shovels and a good tramway system. The majority of the workable deposits are so situated as to necessitate an average haul to railway of five miles by wagon over very poor roads. This freighting is usually done by the farmers living in the immediate vicinity of the mine at the rate of $1.25 per 2240 lb. for a five-mile haul. The mining carried on in the Sweetwater district is all open-pit work, with pick, shovel and wheelbarrow. The barite occurs as nodules in a tough residual Knox dolomite clay, and it is necessary to use some powder to break the ground. The mining cost varies from 75c. to $2 per long ton, depending largely on the amount of mineral in the clay, and the depth of the overburden. The royalty is usually 25c. per ton.

Most of the mining in 1908 was in the Sweetwater district, the principal operators being John T. Williams & Son, W. D. Gilman Company and R. B. Doherty. The output was small. A portion of the raw material was shipped to St. Louis. The Gilman Company operated its plant in a small way most of the year. In addition to the plant for bleaching barytes, it has a small plant for manufacturing barium carbonate. A little work was done in Cocke county, near Wolf creek, in what is known as the French Broad district. There are at least two good properties in this county, one owned by the Commercial Mining and Milling Company, which produces a very white, crystalline mineral, occurring as a vein in quartzite. The other is owned by John T. Williams & Son and has been opened enough to insure a large tonnage of cheap material from a 7-ft. vein between sandstone strata. At Bristol, John T. Williams & Son remodeled the bleaching plant, took out the bur mills and added tube mills for fine grinding. The capacity of the plant is 30 tons of finished product per day. It was in operation only about one-quarter of the year.

Barytes Mining in Foreign Countries.

Australia.—A deposit of barytes was opened recently near Buchan, Victoria. The vein has been well developed, and in some places is 5 ft. wide. Four tons of the material were shipped to Melbourne for a mill

test, which proved satisfactory. A company has been formed to exploit the deposit.

Canada.—Exports for the year 1908 amounted to 3509 tons, valued at $13,690.

England.—Important discoveries of barytes have been made in Ireby parish, Lancashire, on an estate recently purchased by James Gurney, of Iver, Bucks. The veins are said to run 2½ miles, and show between walls up to 18 ft. of solid barytes of exceptionally good color, and up to 98.50 to 99 per cent. of barium sulphate; oxide of iron, 0.05; magnesia, nil; lime, trace to 0.7; silica, 0.76. Fluorspar is also associated with the barytes, and is a valuable by-product. Carbonate of barium, or witherite, has been found in the beds of streams; a search is to be made to discover its origin. The Scordale Barytes Mines, Ltd., has erected an extensive plant to grind barytes.

The production of barytes and other barium compounds for 1908 was 32,074 tons, as compared with 30,343 tons for 1907.

Germany.—Robert P. Skinner, U. S. Consul General at Hamburg, reports that the best grades of barytes from the Harz and Thuringian mountains are naturally white and can be used without first being bleached. To some extent these grades are utilized in the porcelain and white-glass industries. The cheaper grades of barytes come from the Rhine, are shipped in crude form, and these, both hard and soft, are exported to the United States, where they are bleached. Hamburg exporters invariably buy on terms delivered at seaboard. The price Dec. 1, 1908, was $5.47@5.71 per ton. The ocean freight from seaboard to New York runs from $1.70 to $2.19 per ton. The German rate from mine to seaboard, shipments being made in large barges, is from $1.19 to $1.42 per ton. This would make the price at the mine from $4.05@4.52. Since the perfection of a pooling arrangement prices have been advanced from 50 to 75c. per ton. Labor cost, it is stated, has also advanced.

Notes on the Uses and Manufacture of Barium Sulphate, Carbonate, Chloride and Lithophone.

Barytes as a Pigment.—Barytes undergoes no chemical action when used as a pigment. When mixed with linseed oil alone or in conjunction with some colored pigment, the color remains unchanged; it is thus inert as to color. It is, therefore, considered a good base upon which to precipitate any desired color. Barytes is used also to dilute other pigments which have a tendency to destroy the oil. Barytes, moreover, has the property of giving "tooth" to a coat of paint, yielding a surface that is not so hard and smooth, but to which a second or third coat of paint will adhere persistently.

Barium Chloride.—At Bristol, Tenn., a plant for the manufacture of barium chloride has been built which has a capacity of 10 tons of crystals per day. The crude material consists largely of jig tailings, slimes and barytes containing much iron. The barite is mixed with coal and is given a reducing roast in an 80-ft. rotary cement kiln, yielding a mixture of barium sulphide and carbonate. This is leached with water to take out the soluble sulphide and the residue is digested in hydrochloric acid, which dissolves the carbonate. The sulphide solution is acidified with hydrochloric acid. This acid is manufactured at the plant. The chloride liquor is then filtered, purified, evaporated to saturation point and allowed to crystallize. Barium chloride is used extensively in the color industry in the preparation of lakes; it is used also in the manufacture of wall-paper. The fused chloride is used in a small way in tempering tool steel.

Manufacturers of Barium Products.—In the following table are given the names and addresses of all the consumers of crude barytes in the United States that are known to us:

Name.	Address.	Operation.
Nulsen, Klein & Krausse Mfg. Co.	St. Louis, Mo.	Bleach, grind, float.
Finck Mining and Milling Co.	St. Louis, Mo.	Bleach, grind.
Point Mining and Milling Co.	Mineral Point, Mo.	Bleach, grind, float.
Nulsen, Klein & Krausse Mfg. Co.	Lynchburg, Va.	Bleach, grind.
Pittsburg Baryta and Milling Corp.	Richlands, Va.	Bleach, grind.
Commercial Mining and Milling Co.	Knoxville, Tenn.	Bleach, grind.
William D. Gilman Co.	Sweetwater, Tenn.	Crush, jig, roast, salts.
John T. Williams & Son.	Bristol, Tenn.	Bleach, grind, roast, salts.
Carolina Barytes Co.	Stackhouse, N. C.	Bleach, grind.
Hot Springs Mfg. Co.	Hot Springs, N. C.	Bleach, grind.
Delaware Barytes and Chemical Co.	Dover, Del.	Bleach, grind, salts.
Barium Production Co.	New York, N. Y.	Bleach, grind.
Dix River Barytes Co.	Danville, Ky.	Bleach, grind.
Hammill & Gillespie	Stamford, Conn.	Bleach, grind.
Cawley, Clark & Co.	Newark, N. J.	Lithophone.
N. Z. Graves & Co.	Philadelphia	Lithophone.
Harrison Bros. & Co.	Philadelphia	Lithophone.
Excelsior Mfg. Co.	Newport, Del.	Lithophone.
Grasselli Chemical Co.	Grasselli, N. J.	Lithophone.
New Jersey Zinc Co.	Palmerton, Pa.	Lithophone.
Cheeseman Chemical Co.	Scranton, Pa.	Lithophone.
Becton Chemical Co	Becton, N. J.	Lithophone.
American Paint and Pigment Co.	East Alton, Ill.	Roast, salts.
E. E. Dwight & Co.	Webb City, Mo.	Roast, salts.
Krebs Pigment & Chemical Co.	Newport, Del.	Lithophone.
Georgia Barytes Co.	Asheville, N. C.	Bleach, grind.
Vivian, Bond & Co.	New York	Grind and bleach.
Kentucky Barytes Co.	Nicholasville, Ky.	Grind and bleach.

Barium Carbonate.—Imported carbonate of barium (witherite) is employed to neutralize the soluble sulphates in clay used for the manufacture of terra cotta. The quantity of witherite mined in the United States is small, but a small amount of precipitated carbonate is produced. This product finds a small use in the manufacture of glass, whereto it adds

luster and brilliancy. It is also used as a chemical reagent in analytical work.

Lithophone.—An important business in lithophone has already been established in this country. This was at first confined to the imported lithophone, but within the last few years several companies have started to manufacture this pigment. The industry is growing rapidly, and the American lithophone has now been brought to such a stage of perfection that it is replacing the German and English product in the American market. (The manufacture of lithophone was discussed in Vol. XVI of THE MINERAL INDUSTRY.)

Bibliography.—Various articles on barytes have been published in THE MINERAL INDUSTRY, giving statistics each year. In addition to the statistical feature, Vol. II gives a brief account of the history of the barytes mining industry. In Vol. VIII is a description of a process for making caustic baryta, and also a method for the determination of barium in barytes. In Vol. X is an account of the technology of barytes, including a process for the manufacture of barium oxide. In Vol. XIII is a discussion of the process of bleaching barytes and a history of the barytes industry 1887 to 1894 by W. D. Gilman. Vol. XIV gives a description of the Gilman plant at Sweetwater, Tenn. In Vol. XV, E. K. Judd reviews the industry in the United States, with notes on mining, milling, bleaching and roasting barytes. In Vol. XVI is an account of the barytes mining in Canada, by E. K. Judd, and notes on the mining and manufacture of barium products by Edwin Higgins.

BAUXITE.

Events of importance in the bauxite industry of the United States were few during 1908. There were several transfers of property to the larger companies, but little new work was undertaken. The year was characterized by a decrease in shipments in every quarter. As may be noted from the accompanying table, the production was far below that of 1907.

PRODUCTION OF BAUXITE IN THE UNITED STATES.

(In tons of 2240 lb.)

State	1898	1899	1900	1901 (a)	1902	1903 (a)	1904	1905	1906	1907 (a)	1908 (a)
Alabama.....	13,848	14,144	650	18,038	5,577	22,374	7,087	17,094	27,131	97,776 (b)	14,464
Georgia.....	12,943	19,619	20,715		19,000		16,909				
Arkansas.....	.....	3,050	2,080	867	4,645	25,713	24,016	30,897	51,200		(b)37,703
Total.....	26,791	36,813	23,445	18,905	29,222	48,087	48,012	47,991	78,331	97,776	52,167

(a) Statistics of the United States Geological Survey. (b) Production of Tennessee included.

The chief producers of bauxite in the United States are the Republic Mining and Manufacturing Company, 1111 Harrison Building, Philadelphia, Penn.; Aluminum Company of America, Pittsburg, Penn.; National Bauxite Company, Rome, Ga., and John H. Hawkins, Rome, Ga. The Norton Company, of Worcester, Mass., is a calciner of bauxite for use in the manufacture of the artificial abrasive alundum. The mineral mined by the Bauxite Mining and Manufacturing Company goes chiefly into alum and abrasive materials. This company is developing the use of bauxite for refractory purposes. The major portion of the bauxite produced by the Aluminum Company of America is used in the manufacture of aluminum, but this company also sells a large quantity of the mineral to manufacturers of aluminum sulphate.

Industrial Conditions and Prices.—France was the chief competitor in the American bauxite industry in 1908. French bauxite brought $3.50 @4 per ton for the white, and $2.30@3 for the red variety. Shipment by water to this country costs approximately $2 per ton, and the duty is $1 per ton. American bauxite brought $5@7 per ton, the freight rate to Eastern points being $4@5 per ton. Poorer grades were sold at less than $5 per ton. The cost of mining varies between $2 and $4 per ton.

Bauxite in the United States.

The chief bauxite fields of the United States are the Georgia-Alabama and the Arkansas districts. The first-named district includes a series of deposits extending from Adairsville, Bartow county, Ga., to Jacksonville, Ala. The mineral is persistent to great depths. The clay walls surrounding the deposits will not permit deep mining, although work is frequently done as far as 150 ft. beneath the surface. In addition to these two fields bauxite is also mined in Tennessee. The deposits are situated on the southeast slope of Missionary Ridge, near Chattanooga, and are worked by the National Bauxite Company. Here the geologic and structural conditions are essentially identical with those of the Georgia-Alabama district.

Alabama. (By Eugene A. Smith.)—While bauxite is known to occur in several counties in the State, no mining has been done except in the vicinity of Rock Run in Cherokee county. The two companies which have shipped practically all the bauxite produced in Alabama are the Republic Mining and Manufacturing Company and the General Bauxite Company. The amount shipped in 1908 was about 6000 long tons. This is a large falling off from the production in 1907, which was about 11,000 tons.

Arkansas.—The principal deposits are in Saline and Pulaski counties. At the deposits of the Aluminum Company of America, south of Bryant, Saline county, operations were almost at a standstill during the summer of 1908, but later in the year shipments were resumed.

(By A. H. Purdue.)—The bauxite industry in Arkansas suffered from the depression of 1908, and the increasing competition of French bauxite was severely felt. For these reasons no improvements were made in the plants during the year. The Arkansas bauxite is used principally for the manufacture of alumina, which in turn is used to make aluminum. A small amount is used for the manufacture of alum and refractories, but the bulk of this trade is probably controlled by the French and the Georgia-Alabama producers. It is reported that the French producers of bauxite are importing a large amount of ore for the manufacture of alumina in competition with the Arkansas product, and are endeavoring to displace the Arkansas bauxite used in the manufacture of aluminum. Statistics for the production of bauxite for 1908 are not at hand, but the producers believe that the amount did not exceed one-third that of 1907.

Georgia. (By S. W. McCallie.)—The chief output of bauxite at present is from Floyd county in the vicinity of Rome and Cedartown. The

newly discovered deposit in central Georgia, near Macon, which appear to be of considerable promise, is so far undeveloped. However, a company has recently been organized to develop this new field. Many inquiries have of late reached the office of the State geologist concerning the low-grade bauxite which has heretofore been discarded in the Rome and the Cedartown districts. Should this material find a ready market for fire-brick and other refractory purposes, it would, no doubt, stimulate bauxite mining in Georgia, as the removal of this low-grade material from the mines has always added largely to the expense of mining.

CONSUMPTION OF BAUXITE IN THE UNITED STATES.

Year.	Production.			Imports.		Exports.		Consumption.	
	Long Tons.	Value.	Per Ton.	Long Tons.	Value.	Long Tons.	Value.	Long Tons.	Value.
1896	17,096	$42,740	$2.50	2,119	$10,477	.....	.....	19,215	$53,217
1897	20,590	51,475	2.50	2,645	10,515	2,537	$5,074	20,708	56,916
1898	26,791	66,978	2.50	1,201	4,238	1,000	2,000	26,992	69,216
1899	36,813	101,235	2.75	6,666	23,768	2,030	4,567	41,449	12,436
1900	23,445	85,922	3.66	8,656	32,968	1,000	3,000	31,101	115,889
1901	(a)18,905	97,914	4.23	18,313	66,107	1,000	3,000	36,218	144,021
1902	29,222	128,206	4.39	15,790	54,410	*Nil.*	.....	43,112	175,875
1903	(a)48,087	171,306	3.56	14,889	49,684	*Nil.*	.....	62,976	220,990
1904	48,012	166,121	3.46	15,475	49,577	*Nil.*	.....	63,487	215,698
1905	47,991	203,960	4.25	11,726	46,517	*Nil.*	.....	59,717	250,477
1906	78,331	352,490	(c)4.50	17,809	63,221	*Nil.*	.....	96,140	415,711
1907	(a)97,776	480,330	4.91	25,065	93,208	*Nil.*	.....	122,841	573,538
1908	(a)52,167	263,968	5.05	21,679	87,823	*Nil.*	.....	73,846	351,791

(a) Statistics of the United States Geological Survey. (c) Estimated.

Bauxite in Foreign Countries.

Austria.—Polley[1] states that bauxite deposits of excellent quality are of frequent occurrence in the province of Istria, in the form of pockets in red sandstone. They vary from 20 to 80 ft. in length, 12 to 45 ft. in width, and 25 to 60 ft. in thickness. Shallow borings between the visible pockets reveal the existence of numerous others, the average being one to every 15 holes sunk. As the depth is only about 10 ft. from the surface, the work is inexpensive and profitable. In the area open for free prospecting it is calculated that 1000 pockets can be worked, with an average yield of 300 tons of good bauxite. The deposits are favorably situated, being in the vicinity of five railway stations (Domenica, Visignamo, Visinada, Raccotole, and Pisino), as well as the port of Torre. Railway carriage to Trieste costs 2s. 6d. to 3s. per ton, and by water from Torre to Trieste about 1s. 9d., or to Italy 3s. 3d. per ton. The chemical composition of Istrian bauxite is fairly constant, so that the costly grading necessary in other places, where the silica content varies from 1 to 15 per cent., is avoided. The cost of mining is low, the estimate per ton

[1] *Mont. Zeit.*, Jan. 15, 1909.

delivered at Trieste being 8s. 9d., as compared with about 18s. per ton for imported French bauxite of inferior quality.

France.—The first discovery of bauxite in the world was made in the neighborhood of Les Baux, a few miles west of Marseille, which accounts for the name given the mineral. At present, however, the sources of supply are in the department of Var, a few miles east of Marseille. The exploitation of this field has developed greatly during the last two years. This is due chiefly to the building of a number of new aluminum smelteries. There have been several new discoveries of rich deposits and many newly organized companies have obtained concessions. The deposits are regarded as very extensive. There is a large supply of bauxite that yields from 60 to 65 per cent. alumina, and enormous reserves carrying from 45 to 47 per cent. alumina. In 1907, there was exported from France 110,915 tons of bauxite, valued at $471,113.

The French bauxite may be broadly divided into three classes, as follows: (1) The white mineral, which contains 60 per cent. alumina, a maximum of 4 per cent. iron, and practically no silica; this is utilized in the manufacture of chemicals. (2) Red bauxite, containing 60 per cent. alumina and 3 per cent. silica, which is used in the manufacture of aluminum. (3) A special white bauxite for the manufacture of refractory products, containing 45 per cent. alumina, traces of iron and much silica. These refractory products are much sought for use in industries where high temperatures are maintained. Cupolas, locomotive fire-box linings, and glass furnaces are manufactured from bauxite brick.

TECHNOLOGY.

The Nature of Bauxite.[1]—At a recent meeting of the Société de l'Industrie Minérale, Francis Laur outlined some of the various opinions as to the nature and composition of bauxite.

The well-known diversity in the color and texture of bauxites can probably be referred to differences in the mode of origin. Some deposits have almost certainly been formed by the action of hot springs, while others have equally certainly been formed by the decomposition in place, of richly aluminous rocks, under atmospheric agencies. In either case bauxite is characteristically amorphous. Study as to the nature of bauxite is confused by the double character of alumina, which acts as a base in reaction with silica, forming silicates, and as an acid in the presence of iron, forming ferro-aluminates.

H. Fischer does not admit that bauxite is a mineral species, but considers it a mixture of ferric oxide and red clay. Zirkel considers it a mixture of alumina, silica and ferric oxide. Hoffman looks upon it as a mix-

[1] *Eng. and Min. Journ.*, May 30, 1908.

ture, in widely varying proportions, of diaspore and limonite. Richards and Sainte-Claire-Deville consider it a combination of diaspore and brown hematite. Watson, Philips and Hancock agree in thinking of bauxite as a variable mixture, in which the hydrate of alumina, $Al_2O_3+3H_2O$, is the principal constituent. Liebrich thinks that bauxites have the general composition, $Al_2O_3+nH_2O$, in which variable proportions of the water are replaced by iron and silica. Bauxite would thus occupy a position between diaspore, with 15 per cent. water, and gibsite, with 34.6 per cent. water.

M. Laur concludes, from his own large experience with French bauxites, that the percentage of alumina tends to hold closely between 68 and 69, that the total of the three variable elements, silica, iron and water, holds closely to 27 per cent., the remaining 4 per cent. consisting of accidental constituents such as titanium, vanadium, chromium, etc. Although the three variable elements may replace each other in great diversity, their total is never far from the percentage mentioned. Thus, in the case of the white bauxite at Villevyrac, the iron is almost completely replaced by silica; in the red bauxite of Var, iron replaces almost all of the silica and part of the water; while in the Alabama bauxites, both silica and iron are in subordinate amounts, the ore from that district having almost the theoretical composition of the tri-hydrate. At Baux, the three types are found combined.

Titanium is the most common impurity of the accidental group. It has not yet been determined in what form the titanium occurs, whether as titanate of iron or of alumina, or simply as titanic oxide. Sainte-Claire-Deville is said to have found crystals of rutile in the residue from an analysis of bauxite. The proportion of titanium occasionally reaches 12 per cent., but in ordinary amount it has no ill effect upon the utility of a bauxite, because of its insolubility.

Hartley and Ramage have lately subjected bauxite from Antrim, Ireland, to spectroscopic analysis, by which the presence of impurities in minute amounts could be easily detected. In the so-called pure alumina made from this bauxite they found sodium, potassium, copper, iron, lead, gallium and traces of indium. In the crude bauxite they found, in addition to the above, silver, nickel and chromium. In the aluminum made at Foyers from the same material, they found all the previously mentioned elements, and in addition, calcium and manganese, the last two of which, however, were undoubtedly introduced during the electro-chemical process. In bauxite from New South Wales, other chemists are said to have recognized traces of neodymium and lanthanum.

Relative Value of Bauxite.—Anton Polley states that the assumption that red bauxite with 20 to 27 per cent. ferric oxide is unsuitable for

the production of pure alumina, is based on an error. The gray and light yellow varieties, still frequently used for the production of alumina, contain a large proportion of silica (5 to 20 per cent.) and but little ferric oxide (2 to 8 per cent.), but the method is costly and out of date, sulphuric acid being used for the dissociation, while the ferric sulphate formed takes a long time to crystallize out. Red bauxite containing only from 1 to 3 per cent. silica and 2 to 3 per cent. titanic acid, is preferable for the purpose in view, even though the iron content be higher, and this is the material used at the Niagara Falls works, at Marseille, Laibach and in Swiss works. The alumina is dissociated with calcined soda, the process is under better control, and the ferric oxide is not a source of difficulty since it separates out undecomposed. Good French bauxite contains from 55 to 63 per cent. alumina, 1 to 5 per cent. silica, 2 to 3.2 per cent. titanic acid, and 19 to 28 per cent. ferric oxide. As a rule, the higher the percentage of iron, the lower the silica, which, with the titanic acid, is the only injurious substance present. In fact, none of the existing large works in France, Austria, Germany and Switzerland will use bauxite with more than 3 per cent. silica and 3 per cent. titanic acid, although the ferric oxide may attain 27 per cent. On the other hand, the percentage of iron must be low when the bauxite is used for making fire-resisting material.[1]

Bibliography of 1908.

There was published in *Rassegna Mineraria,* August 1 and 8, 1908, an article by C. Martelli, describing the occurrence and methods of working several bauxite deposits in the province of Abruzzo, Italy.

Of the patents recently issued on the treatment of bauxite, the following are noted:

Sulphate of Alumina.—Improvements in the treatment of the residue or sludge resulting from the production of sulphate of alumina. H. D. Pochin & Co., Ltd., and John H. Richardson, Salford, Manchester, England. (Brit. No. 25,227 of 1907.)

Treatment Process.—Improvements in and connected with the treatment of bauxite or aluminous material. H. D. Pochin & Co., Ltd., and John H. Richardson, Salford, Manchester, England. (Brit. No. 25,228 of 1907.)

[1] *Mont. Zeit,* Dec. 15, 1908.

BISMUTH.

So far as we are aware there was no production of bismuth ore in the United States in 1908, the market conditions offering no inducement. The production of metallic bismuth was 5200 lb., compared to 10,000 in 1907. Of the production in 1908 a part was made by the United States Metals Refining Company, at Grasselli, Ind., and a part by the National Lead Company. The latter recovers the metal as a by-product in the manufacture of acetate of lead, a residue containing 8 per cent. bismuth remaining after dissolving the pig lead that is used. It is probable that the production of bismuth from the Betts-process slime at Grasselli will be increased in 1909.

In 1907 about 10,000 lb. of electrolytic bismuth was produced by the Monsanto Chemical Works, of St. Louis, which obtained bismuth ore from Mexico, together with bullion assaying 78 to 82 per cent. lead and 16 to 12 per cent. bismuth. The electrolytic bismuth was 99.85 per cent. pure. The Monsanto Chemical Works put about $20,000 into its plant and appeared to have a fair prospect for business, but claims to have been checked by threats of the European syndicate, which warned American consumers that if they bought bismuth from other sources than the syndicate, or even encouraged the production of bismuth in the United States, there might be difficulty in securing supplies from abroad at favorable prices if it should be necessary to revert to that source.

The chief sources of bismuth are Bolivia, which exports 200 to 600 metric tons of crude metal per annum, and Saxony. The smelting and refining is done by Johnson, Matthey & Co., of London; the Royal smelters at Freiberg and Oberschlema, in Saxony; and the Deutsche Gold und Silberscheideanstalt, of Frankfurt, Germany. Most of the Bolivian material goes to London. These smelters form a syndicate which exercises a firm control over the market. It is said that by the terms of the understanding the British, French and American market is granted to Johnson, Matthey & Co., while the German concerns have the rest of the world. They must have reaped great profits, judging by the drastic character of the cut made in 1905 when it became necessary to shut out new competition, the price which had previously been 9s. per lb. being then reduced to 5s. On July 4, 1907, it was raised to 6s. 6d.; at which figure it has since remained.

Uses.—Bismuth is used for the manufacture of soft solder, in combination with lead, tin, and antimony for an alloy of low melting point, especially desired in automatic sprinkling heads used with the automatic sprinkling system, now generally installed in most large commercial and

IMPORTS OF BISMUTH INTO THE UNITED STATES.

Year.	Pounds.	Value.	Av. per lb.
1896	124,263	$ 90,950	$0.73
1897	151,374	172,236	1.14
1898	137,205	162,846	1.19
1899	176,668	208,197	1.18
1900	180,433	246,597	1.37
1901	165,182	239,061	1.45
1902	190,837	213,704	1.12
1903	147,295	235,199	1.60
1904	185,905	339,058	1.82
1905	148,589	318,007	2.14
1906	254,733	318,452	1.25
1907	259,881	325,015	1.25
1908	164,793	257,397	1.56

factory buildings. Bismuth salts are in large demand. The oxychloride is largely used in face powders, and the nitrate for cosmetics. The oxide is used to a small extent in making porcelain colors, while the subnitrate and other salts are used for medicinal purposes.

Assay.—F. J. Metzger and H. T. Beans (*Journ.*, Am. Chem. Soc., April, 1908, p. 589) describe a method for the electrolytic determination of bismuth which is said to be free from the difficulties encountered in previous methods. The metal is deposited in compact form and can be washed and dried without loss, and there is no deposit of peroxide on the anode when the electrolysis is complete. The principles of the method are: the use of acetic acid to prevent the hydrolysis of the bismuth nitrate solution, the elimination of free nitric acid by the addition of sodium hydroxide, and the addition of boric acid to improve the character of the deposit and complete the solution of bismuth hydroxide without the addition of large quantities of acetic acid. The bismuth is dissolved as a nitrate in water containing 25 c.c. of concentrated nitric acid per liter. To a known volume of this solution phenolphthalein is added; then sodium hydroxide solution, drop by drop, to alkaline reaction. The precipitate formed is dissolved in acetic acid and then two grams of boric acid are introduced. The solution is diluted, heated to 70 to 80 deg. C. and electrolyzed. A rotating cathode is employed.

BORAX.

The most striking feature of the borax industry during 1908 was the comparatively small number of mines operated in California, the center of the industry. The mines of the Pacific Coast Borax Company, and those of the Sterling Borax Company were the only ones which produced any great amount of the mineral. A limited production was reported by Chinese lessees near Columbus, Esmeraldo County, Nev. The accompanying table shows the quantity and value of the borax produced in California for a period of years.

PRODUCTION OF BORAX IN CALIFORNIA. (a)
(In tons of 2000 lb.)

Year.	Tons.	Value.	Year.	Tons.	Value.	Year.	Tons.	Value.
1897.....	8,000	$1,080,000	1901.....	7,221	$ 982,380	1905.....	46,334	$1,019,158
1898.....	8,300	1,153,000	1902.....	(b)17,202	2,234,994	1906.....	58,173	1,182,410
1899.....	20,357	1,139,882	1903.....	34,430	(c) 661,400	1907.....	53,412	1,200,913
1900.....	25,837	1,013,251	1904.....	45,647	(c) 698,810	1908.....	33,500	117,500

(a) Reported by the California State Mining Bureau, except for 1908. (b) Mostly refined borax, whence the apparent discrepancy in value. Output of the other years is given as crude material. (c) Spot value.

Industrial Conditions and Prices.—The intense competition which proved so disastrous to prices during 1907 was still in evidence in the early part of 1908, resulting in two declines in borax from the already low price, one of ½c. and the other of ¼c. per lb. This brought the price down to 4¼c. per lb. in carload lots. This decline caused a reduction in the price of boracic acid from 7¾c. to 7c. per lb. While no further reductions were noted during the latter part of 1908, it was reported that concessions could be secured for large lots. Toward the end of July the price of impalpable powdered acid was reduced 1c. per pound.

Borax in California.

Prices for borax have fallen so low that most of the mines have ceased operations entirely; indeed nearly all of them have been closed down for a year or more, with little expectation of starting up again for some time. In some instances the plants have been removed altogether, and the properties closed for good; this is particularly the case in the neighborhood of Daggett, San Bernardino county, where the low-grade marsh

muds, or superficial deposits, were being worked. The old colemanite mine at Calico, for many years the most productive mine in the State, has been permanently abandoned, having been virtually worked out by the Pacific Coast Borax Company. Some of the companies operating in that district produced in 1907, but none of them did during 1908. The companies which were working in Ventura county also closed down their properties in 1907, and they will remain idle indefinitely, for the long haul to market makes the cost of production too expensive in that region.

Some of the companies made very little profit even when refined borax sold for 7c. per lb., and when the great reduction occurred in 1907, there was nothing for them to do but quit work. It was about August, 1907, when the Pacific Coast Borax Company reduced the price of the manufactured product; very shortly after that all the borax properties of the State, with the exception of the Lila C. mine belonging to the Pacific Coast Borax Company in Inyo county, ceased operations. In August, 1907, this company ceased operations entirely at Daggett and Calico in San Bernardino county, removed its plant, and commenced its activities in Inyo county. This move had been contemplated for several years, during which period the mine was being developed; so when the new railroad near the mine was completed the company concentrated its efforts at the new property. The deposit is an extensive one, and being well developed, is in a position to make even a much larger annual output than the demand requires. The grade of mineral mined is high and the facilities excellent for working, though labor is high in that region. The company being able to produce largely, at less cost than formerly, was in a position to reduce prices, and thus put its competitors out of business. Those companies which ceased producing in 1907 were the Western Borax Company, of Inyo county, the American Borax Company, the Western Mineral Company and Palm Borate Company of San Bernardino county, and the Columbus Borax Company and Frazier Borate Company, of Ventura county. None of these resumed operations in 1908 at former points of production, though some have exploited new fields.

There are many known borax deposits in the Death valley region; several of these are controlled by the Pacific Coast Borax Company, but others (which with the present means of transportation would not pay) are still unlocated. The fact is that there has been an overproduction; so the larger companies are trying in every way to increase the use of borax products, and have done some experimenting in order to find new uses for boron compounds.

The only new properties of note are those of the Borax Properties, Ltd., a company backed by English capital, which in 1908 bought the property of the Palm Borate Company of Daggett, and also considerable other ground in that vicinity. This company has been engaged for some months in erecting a plant and carrying on extensive operations. The property has not produced any mineral, as only development has been done so far. However, the company expects before long to put its product on the market.

The Sterling Borax Company is the most prominent of the new concerns. It owns the mines in Ventura county of the Stauffer Chemical Company, formerly known as the Frazier Borate Company, the mines of the American Borax Company at Daggett, and the new deposit found a year or so ago at Lang station in Los Angeles county. The refining

SOME OF THE PRINCIPAL SUPPLIES OF BORAX PRODUCTS.

(In metric tons.)

Year.	Chile. (a)	Germany (b)	Italy.			United States. (c)	Total. (d)
			Borax Refined.	Boric Acid.			
				Crude.	Refined.		
1897....	3,154	198	990	2,704	200	7,257	14,563
1898....	7,028	230	702	2,650	166	7,529	18,305
1899....	14,951	183	709	2,674	129	18,466	37,112
1900....	13,177	232	858	2,491	283	23,437	40,478
1901....	11,457	184	544	2,558	347	6,550	21,640
1902....	14,327	196	.....	2,763	.....	15,512	32,798
1903....	16,879	159	.....	2,583	.....	31,232	50,853
1904....	16,733	135	569	2,624	314	41,407	61,782
1905....	19,612	183	(e)1,007	2,700	(e)749	42,036	66,287
1906....	28,996	161	1,062	2,561	562	52,774	86,116
1907....	28,374	114	881	2,305	466	48,444	80,584

(a) Prior to 1903, figures are for borate of lime exports. (b) Boracite. (c) Crude borax. (d) The total falls short of the World's supply, particularly because it fails to include the important production of Turkey. (e) Obtained by treating a part of the crude boric acid reported for 1905.

plants at San Francisco, Chicago and New Brighton are combined in this organization. The deposit is a vein of colemanite of good grade and is at a place where the cost of production is low, as there is plenty of water available and the prices of labor and of transportation are favorable.

The vein varies in width from a few inches to 10 ft., being like most of such veins of a "pockety" nature. The owners of this property state that they are able to compete in output and price with any other mining company of the State. The mine began to produce about October, 1908. The product is shipped from Lang's station to Chicago, New Brighton and San Francisco, the ore being shipped crude as mined, and then refined at the points named. About 700 tons monthly have

been produced thus far but this amount can be readily increased when desired.

The mines of the Pacific Coast Borax Company and the Sterling Borax Company are the only two properties in the State which produced much borax in 1908. As stated, under present conditions, there is little or nothing in the borax-mining industry for any companies except those with extensive high-grade properties, near points of railroad transportation, and with the prevailing low prices there can be little profit even for properties with exceptional advantages.

The production of refined borax in the United States in 1908 was 17,115 tons, valued at $1,537,688. The production of boric acid was 3067 tons, valued at $406,767.

BROMINE.

The production of bromine in the United States increased in 1908 notwithstanding the depressed condition of the market. In Michigan the only producers were the Dow Chemical Company and the Saginaw Salt Company, consequently the statistics for that State have to be grouped with those for other States. It may be noted, however, that Michigan is still the largest producer.

At the end of 1907, bromine was sold as low as 10c. per lb., f.o.b. works, which was stated in Vol. XVI as "believed to be below the cost of production to all the makers except the Dow Chemical Company," but in 1908 bromine sold at 8c. and potassium bromide at 11c. The range of the year was 8 to 10c. for bromine, the opening price being 10c., which price was again the basis of sales at the close of the year. Potassium bromide ranged from 11 to 14c. until late in December, when the price advanced to 18c. These were the prices f.o.b. works.

PRODUCTION OF BROMINE IN THE UNITED STATES.

(In pounds.)

Year.	Michigan. (a)	Ohio and Penna.	West Virginia.	Total. (a)	Metric Tons.	Value.	
						Total.	Per lb.
1897	147,256	241,939	97,954	487,149	221	$136,402	28c.
1898	141,232	226,858	118,888	486,978	220	136,354	28
1899	138,272	193,518	101,213	433,003	196	125,571	29
1900	210,400	196,774	114,270	521,444	237	140,790	27
1901	217,995	227,062	106,986	552,043	250	154,572	28
1902	226,452	194,086	93,375	513,913	233	128,742	25
1903	320,000	180,000	97,000	597,000	271	170,145	28½
1904	646,249	147,807	85,256	879,312	399	215,431	24½
1905	579,434	223,000	97,000	899,434	408	139,492	15½
1906	955,000	203,000	71,000	1,229,000	553	184,350	15
1907	(b)	(b)	(b)	1,062,000	482	138,060	13
1908	(b)	(b)	(b)	1,149,000	521	103,410	9

(a) Includes the bromine equivalent of the bromides produced directly. (b) Not reported separately.

In the New York market, during nearly the whole of 1908 quotations remained on a steady basis at 16c. for potassium bromide, 21c. for sodium bromide and 23c. for ammonium bromide, with little of interest beyond reports, chiefly from London, of the prospects of a settlement of the differences between the powerful German syndicate and the American producers. Developments gave little support to these rumors and the effect

of the keenness of competition, stimulated by the contest over 1908 contracts, assumed tangible form in the announcement of a decline of 1c. in November and one of 2c. in December, establishing the market at 13c. for potash, 18c. for soda and 20c. for ammonia. However, at the very close of the year it was believed that the foreign competition would be eliminated, and the outlook for 1909 was regarded more hopefully.

Valuable data as to the bromine business are to be found in THE MINERAL INDUSTRY, Vol. XVI.

THE TECHNOLOGY OF BROMINE.

BY OSCAR C. DIEHL.

Bromine is produced commercially in Michigan, Ohio, Pennsylvania and West Virginia. In Michigan the industry exists at the following places: Midland, Mount Pleasant and St. Charles. Formerly bromine was also made at Saginaw, St. Louis and East Tawas. Midland is the place where the industry began in Michigan, it being introduced by men from the Ohio river district, who brought with them the process of manufacture then employed in Ohio, which is still being used with slight modifications. The process, starting with the brine from the wells, is described in the following paragraphs:

Ordinary Process.—The brine is pumped into large tanks (called cisterns) in which it is treated with slaked lime, rendering the brine alkaline and precipitating the ferrous iron. (The ferrous iron exists in the brine as carbonate.) After the iron precipitate has settled, the brine is run into other tanks fitted with steam pipes; these are known as settlers. In these, the process of precipitation and clarification is completed and the brine is concentrated to about the salt-making point. It is then ready for the grainers, which are long shallow tanks, usually with four or five lengths of steam pipes. In the grainers the salt is removed from the brine by the usual process of evaporation and crystallization. When practically all of the salt is out, the bittern (mother liquor containing the more soluble salts of calcium and magnesium chlorides, bromides and iodides) is run into other tanks where the evaporation by heat is continued until the liquid has a gravity of 35 to 40 deg. B. It is then ready for the extraction of bromine.

The boiling bittern is run into sandstone stills having a capacity of from 400 to 1000 gal. and treated with dilute sulphuric acid and sodium or potassium chlorate, producing free chlorine, that in turn frees the bromine which undoubtedly exists in the bittern as magnesium bromide.

Formerly manganese dioxide was used in the place of chlorate but it was discarded for reasons of economy and ease of handling. Many of the

manufacturers now use sodium chlorate in place of the potassium salt, it being cheaper, producing more chlorine per pound.

The bittern in the still is kept at the boiling point by introducing steam continuously. The liberated bromine is readily blown out by means of the steam. The vapors of bromine and water coming from the still are condensed in lead pipes running through cold water contained in a rectangular box, 5 to 6 ft. long and 3 ft. deep.

In my own plant, this objectionable lead condenser, which has to be renewed frequently at considerable expense, has been replaced by hollow earthenware discs and coils of the same material. These work perfectly and are of great advantage in preserving the purity of bromine, that substance being readily contaminated by lead, which it attacks. The bromine is collected in stoneware receivers, is separated from the water that accompanies it, and is put up in stoppered bottles holding 6½ lb. Sixteen of these are packed in a case, which is the usual package.

The above describes the general process of obtaining bromine in this country, with the exception of that employed at Midland and Mount Pleasant, where the electrolytic process is used, which is as follows:

Electrolytic Process.—The brine at Midland and Mount Pleasant is high in bromine, containing about 0.1 per cent. This brine is treated with the electric current in a specially constructed cell. The sodium chloride is disassociated into sodium hydroxide and chlorine, which frees the bromine. The liquor containing free bromine is then pumped into towers and spread over a great surface. Air is then forced through the towers removing the bromine. The bromine-laden air is conducted to another tower, where it comes in contact with sodium or potassium hydroxide, which take up the bromine as bromides and bromates. Those salts are crystallized, dried, calcined to remove the bromate, then recrystallized and are ready for the market. This process differs from the other in that the bromine is recovered not as bromine but as bromides, and if bromine is wanted it must be freed again from the bromides.

Bleach and Calcium Chloride.—The salt left in the brine after the extraction of bromine is further electrolyzed until practically all the chlorine is removed. This chlorine is absorbed in lime, forming chloride of lime, or bleach. In all the bromine-bearing brines of this country, there is always associated calcium and magnesium chlorides and in connection with the bromine industry there is often found a plant for the production of calcium chloride. Such plants exist at Mount Pleasant and at Allegheny and at the plants along the Ohio river. The process consists in neutralizing the acid liquors of the still with lime, driving off most of the water and running into drums of about 600 lb. capacity.

This calcium chloride is used for the production of solutions used extensively in connection with refrigeration plants.

Occurrence of Bromine in Michigan.—The bromine-bearing brines of Michigan are obtained from the Marshall sandstone and are the highest in bromine content of any found naturally. They occur in a natural basin, the deeper wells containing the highest bromine content; they are also highest in calcium and magnesium salts.

Along the shores of Lake Huron, Detroit river and Lake Michigan occur the well-known beds of rock salt and the brines of this Saginaw valley basin are no doubt the mother liquors of those rock-salt formations, containing the more soluble salts of calcium and magnesium, the bromides and iodides.

Market Conditions.—The market conditions of the industry have been very unsatisfactory of late, bromide of potash selling as low as 13c. per lb. and bromine at 9c. These unusual prices have been brought about by importation of German bromides which were sent in here as a retaliatory measure, as undoubtedly some of the American manufacturers have invaded fields which were formerly supplied by the German makers. But fortunately within the last few weeks[1] there has been evidence of some agreement inasmuch as the Germans have withdrawn from our market.

The future of the industry will look brighter with the advent of better market conditions. The curtailment of production that took place during the summer of 1908 will perhaps lead to a prosperous year in 1909. It is now hoped that the ruinous competition between foreign and American makers is over for all time.

Uses.—Liquid bromine is used as a disinfectant, as a solvent in the production of gold in connection with the cyanide process, and in the manufacture of certain dyes, such as eosine and Hoffman's blue. In combination with sodium, potassium and ammonium, bromine is used medicinally and in photography for the manufacture of sensitive plates.

[1] This MS. was received Feb. 1, 1909.—*Editor.*

CADMIUM.

This metal continues to be produced in the United States by but one concern, viz., the Grasselli Chemical Company, of Cleveland, Ohio. Its production in 1908 was about 10,000 lb., compared to 15,000 lb., valued at $1.25 per lb. in 1907. The production in Upper Silesia in 1908 was 68,200 lb., against 70,550 in 1907. There was also a production in Great Britain, for which no statistics are available. Besides the production of metal, there is now a considerable utilization of cadmiferous residues, which are employed for the manufacture of cadmium yellow. This utilization appears to be increasing, naturally with an adverse effect upon the market for the metal.

At the beginning of 1908 the price for the metal in the United States was $1.25 per lb. Later it was reduced to 75c. At the end it was 80c. These were the quotations for round lots. In Germany the price at the beginning of the year was 800 marks per 100 kg. Under various adverse influences it fell off and at the close cadmium bars warranted to contain 99.5 per cent. Cd. were offered at 525@575 marks per 100 kg., according to size of order.

The production of cadmium could be hugely increased if there were demand for it. Speaking of this, the *Eng. and Min. Journ.* said in an editorial, July 11, 1908, that "the Joplin blende contains a comparatively high percentage of cadmium, indeed more than any other ore produced in large quantities, but unfortunately the uses of cadmium are few.[1] Although under present conditions cadmium fetches a high price, it is one of the substances in limited demand for which a high value is maintained only so long as the production is small. The inauguration of competition in such industries invariably leads to a tumble in price that is disconcerting and dispelling of hopes of immense profits. Recent experiences in the production of bismuth, cobalt oxide, lithia and other substances are illuminating in this respect.

"With respect to cadmium, the amount that can be recovered as metal is only a tithe of what exists in the ore. But even so, the present producers could furnish a vastly greater quantity if they could sell it, and moreover the smelters of silver-lead ore could also furnish it.

[1] However, they are growing in extent and importance, and would no doubt increase materially if cadmium should become available at a lower price.

"The producer of zinc ore is not refused anything to which he is entitled when he is not paid for the cadmium in his ore. On the contrary the smelter is a good deal more likely to object to its presence. He has to be constantly on his guard to avoid paying for cadmium as zinc, because with the careless chemist cadmium may be reckoned as zinc in the assay; and, moreover, a considerable portion of the cadmium goes into the spelter, to which some consumers of spelter object; unreasonably we think, but still they do object."

(By Paul Speier.)—The editorial in the *Eng. and Min. Jour.* of July 11, p. 92, describes the situation in the cadmium industry perfectly. Lately considerable cadmium has been produced, not as a product in the preparation of crude zinc, but as a by-product in purifying zinc solutions in the manufacture of lithophone. In the preparation of lithophone the impure zinc solutions are purified by boiling with zinc dust which precipitates foreign metals, chiefly cadmium, after iron and manganese have been removed. In a new process at Marienhütte, Upper Silesia, the crude solutions are subjected to electrolysis in lead-lined tanks. For anodes plates or bars of metallic zinc suspended in the middle of the tank are used. The zinc dissolves in the neutral liquor while the foreign metals are deposited as a slime on the lead lining which serves as cathode. The electrolysis is complete as soon as a sample shows the absence of cadmium. This complete removal of cadmium also serves to prevent the development of a yellow tint in the finished lithophone.

At present residues containing cadmium are utilized in Germany as well as in Great Britain for the preparation of metal containing about 99.5 per cent. cadmium. Cadmium yellow is also produced directly from the residues, which naturally affects the consumption of the metal.

American cadmium was offered at 6.75 marks per kg. Hamburg (about 70c. per lb.) in the summer of 1908. Metal from the Silesian furnaces was held at about the same price. The price naturally varies with production and consumption, and it has been subject to marked fluctuations. In 1874 cadmium brought 15 to 16 marks per kg., while in 1884 the price had fallen to 8.50 @ 9 marks. Between 1888 and 1893, quotations fell to 4 @ 5 marks per kg. In April, 1896, the Government Feuerwerk Laboratorien and the Königl. Sächs. Artillerie-Direction took considerable quantities out of the market for special purposes, so that at the end of December, 1897, the price had risen to 21 marks per kg. When the special demand ceased, a rapid decline occurred. The price varied between 5.50 marks and 6.50 marks per kg. until the close of 1905. Soon after the beginning of 1905 active inquiry advanced quotations to 15 marks per kg., the high point of that year. This high price stimulated the manufacturers to increased production, which the consumption could not absorb, and quotations fell to 6.75 marks per kilogram.

Alloys of Cadmium and Tin.—The *Brass World* has made some experiments upon the alloys of cadmium and tin, reported in its issue for November, 1908. Tin and cadmium are of practically the same color, so it was reasonable to suppose that the alloys would retain their white color. This was found to be true. The following results were obtained:

90 per cent. cadmium and 10 per cent. tin.—Slightly harder than pure cadmium and of practically the same color. Soft, and could be rolled into sheet. Casts well in metal molds. Other properties the same as cadmium.

80 per cent cadmium and 20 per cent. tin.—Slightly harder than the 90 per cent. cadmium mixture. Color practically the same as cadmium. Casts well and rolled into sheet easily.

70 per cent. cadmium and 30 per cent. tin.—This alloy is harder than either of the previous ones, but it is still a soft metal. The castings, however, bent with more difficulty than either the 90 per cent. or the 80 per cent. cadmium. The casting qualities of this mixture are excellent and the metal runs freely and sharply. Rolled into sheet without difficulty. Color was somewhat whiter than cadmium although not as good as tin.

60 per cent. cadmium and 40 per cent. tin.—This mixture has the whitest color of any of the alloys and is superior to tin in this respect although much inferior to silver. The cast bars were stiffer than those of the previous mixtures. This mixture, and the one containing 60 per cent. tin and the 50 per cent. alloy, were the stiffest and hardest of the whole series. Rolled well into sheet.

50 per cent. cadmium and 50 per cent. tin.—This mixture has all of the properties of the 60 per cent. cadmium alloy except it is not as white. The difference, however, is slight. The bars were equally as stiff and they rolled into sheet without cracking. While the term "hard" will have to be applied to these mixtures which have the maximum hardness, the metal is, in reality, soft; it rolls well and dents easily. In hardness this mixture closely resembled the best britannia metal or hard babbitt.

40 per cent. cadmium and 60 per cent. tin.—Not as white as the 50 per cent. mixture. It is somewhat whiter than pure tin or cadmium. About as hard and stiff as the 50 per cent. mixture. Readily rolled into sheet.

30 per cent. cadmium and 70 per cent. tin.—Softer than the three preceding alloys and resembled the 70 per cent. cadmium mixture very closely. In color it is a trifle whiter than tin. Rolled into sheet as readily as pure tin or cadmium.

20 per cent. cadmium and 80 per cent. tin.—A little whiter than tin, and also slightly harder. Rolled into sheet without difficulty.

10 per cent. cadmium and 90 per cent. tin.—In color it is the same as tin, and is almost as soft. Rolls into sheet as well as the purest tin, and is practically the same in its properties. The slight amount of cadmium serves to harden the tin in a small degree, but otherwise the mixture has all of the properties of tin.

A summary of these experiments is as follows: Whitest mixture.—60 per cent. cadmium and 40 per cent. tin. Hardest mixtures.—Those containing 60, 50 and 40 per cent. cadmium. There seemed to be little difference among them. All of the mixtures roll into sheet without difficulty. They cast well and the hardest mixtures have about the same hardness as britannia metal or hard babbitt. They have no "ring" when struck, but remain "dead" like tin or solder. The casting qualities of the 60, 50 and 40 per cent. cadmium mixtures are excellent and they might be used for metal mold work should the price admit. They do not shrink much. Owing to the cost of the cadmium, these alloys could not be used to replace the britannia metals or similar mixtures. Owing to the whiteness, however, the mixtures containing from 20 to 30 per cent. of cadmium are now used for anodes in electroplating the interior of telephone parts. They are used to replace nickel and are said to be cheaper than that metal in actual practice. The deposit is nearly as white as silver. Undoubtedly they will find use in other lines of electroplating. The 60 per cent. cadmium alloy would be the best, as it is the whitest, but its cost is against it. With the exception of use in electroplating, the alloys of cadmium and tin, as far as known, are not used in the arts.

CALCIUM.

The only producer of this metal in the United States is the Virginia Electrolytic Company of Holcomb's Rock, Va. The production in 1907 was 350 lb., valued at $613. In 1908 it was about the same as to quantity. The metal sold in 1908 at $1.50 per lb. There is believed to be an important future for this metal. The manufacturer has received numerous inquiries for calcium hydride, which is made by heating calcium in an atmosphere of hydrogen. Calcium hydride reacts with water much like calcium carbide, but gives off pure hydrogen instead of acetylene. This is already used to some extent abroad for the inflation of balloons. Calcium is also used as a deoxidizing agent and in the preparation of certain alloys.

Calcium is a silvery white metal, which oxidizes quickly in moist air. It is very light (sp. gr. 1.72), fairly malleable, has a high specific heat, and is a good conductor of electricity. It is about as hard as aluminum, but at 400 deg C. becomes as soft as lead. It melts at about 800 deg. It can be sublimed in vacuo between 700 and 800 deg. C. It is a powerful reducing agent. A promising application is for the refining of metals. It acts in three ways: by reducing oxides and sulphides; by eliminating occluded gases; and by forming compounds with certain impurities, thus rendering them less deleterious. A suitable addition of calcium will remedy "dry" or "sulphury" copper, giving a sound casting and a soft, tough ingot, besides restoring ordinary overpoled copper to tough pitch. If excess of calcium is used, however, it produces brittleness. For further data as to the properties, production and uses of this metal reference should be made to THE MINERAL INDUSTRY, XVI.

CALCIUM CARBIDE.

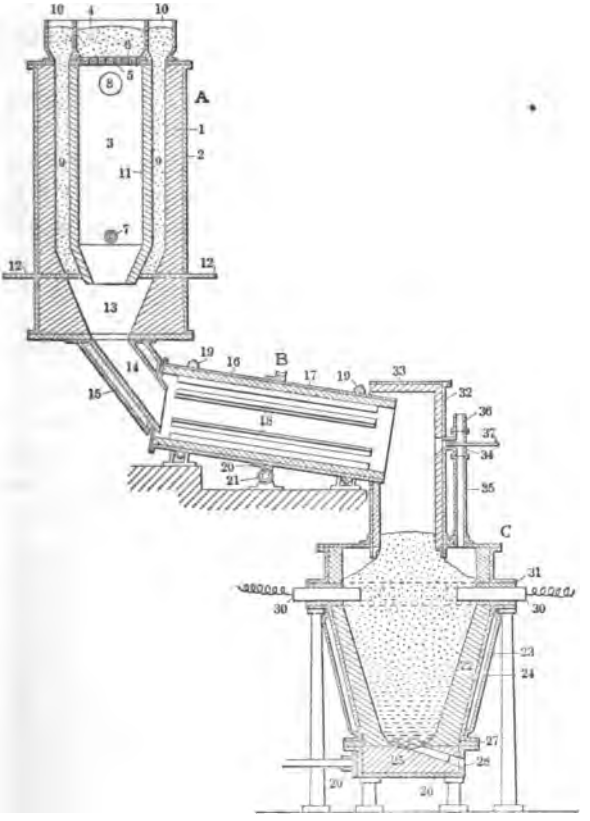
The Union Carbide Company, with works at Niagara Falls, N. Y., and Sault Ste. Marie, Mich., is the only producer of this substance in the United States. It being unwilling to report its production, no precise statistics can be given. According to *La Nature*, Feb. 13, 1909, the world's production of calcium carbide is about 200,000 metric tons. The United States and Canada furnish 38,000, Italy 32,000, France 27,000, Norway 25,000, Switzerland 20,000, Austria-Hungary 20,000, Sweden 12,000, and Germany 9000. The figure for Norway is obviously too low. According to another authority the world's consumption of calcium carbide is about 180,000 tons.

In Norway there are now five calcium-carbide works, viz., at Notodden, of about 4000 h.p.; at Hafslund, of about 20,000 h.p.; at Borregaard, of about 2000 h.p.; Meraker, of about 4000 h.p.; Ihlen, in Trondhjem, of approximately 3000 h.p. Two others will be completed in 1909, viz., at Odda, of about 20,000 h.p., and at Kragerö, of about 4000 h.p. The productive capacity of these works, including those under construction, is 80,000 to 90,000 tons yearly. The export of carbide in 1907 was 22,591 tons, valued at four million kroner; and in 1908 it was 36,666 tons, at a value of about six and a half million kroner. The home consumption was insignificant. During 1908 the price fell from about 190 kroner to 160 kroner per ton, in Norwegian ports.

The great development of carbide works in Norway, as well as in Italy, France, and Switzerland, has led to over-production, and the near future is regarded with anxiety. It seems to be unavoidable that the works must agree to some restriction of output in order to establish again a balance between production and consumption. In France, the Bullier patent expired on Feb. 9, 1909, and the manufacture has now become free in that country. This expiry of patent rights is considered to be a menace to French carbide manufacturers, who will have to compete with other countries, and they demand a rather high import duty to enable the industry to compete successfully. At present the duty is 5 per cent. ad valorem.

New Furnace.—E. F. Price, general manager of the Union Carbide Company has patented (U. S. 904,991, Nov. 24, 1908) a calcium carbide furnace, shown in the accompanying illustration, the special feature

being the method of preheating and feeding the charge. The preheating apparatus *A* is a rectangular tower (1) of firebrick, enclosing the lime chamber (3). The lime and carbon are charged separately. The lime falls from the hopper (4) and through the inner chamber (3) in form of a shower and is thereby heated by the hot products of combustion rising from the burners (7) and escaping through a port (8). The carbon is fed from the hopper (10) and moves downward through passages (9)



CALCIUM CARBIDE FURNACE.

in the wall of the tower. The preheated materials pass into the revolving cylinder *B* which acts as mixer, and then into the electric furnace *C*, in which the mass acts as resistor, the bottom (25) of solid carbon acting as one electrode and carbon rods (30) arranged radially at the top, as the other electrodes. Molten calcium carbide collects on the bottom and is tapped through 28. A burner (34) opens into the chamber (32) opposite the end of the mixer and is fed with waste carbon monoxide from the

furnace through the pipe 35, while air is supplied by pipe 37. In this way a high temperature is maintained in the mixer.

Bibliography.—The following are a few of the important technical contributions to the subject in 1908.

"Test of a Low-Voltage Alternator for Calcium Carbide Furnace." (*Elec. Rev.*, Lond., Jan. 17, 1908.) This gives the results of a test on a 430-kw. alternator, installed for making calcium carbide in Ireland, and discusses the results.

"Ueber die Fortschritte in der Verwendung grosser elektrischer Oefen zur Fabrikatich von Kalziumkarbid und Hochprozentigem Ferrosilizium." Walter Conrad. (*Stahl u. Eisen*, June 10, 1908.) Describes the latest European improvements in furnaces for calcium carbide manufacture.

"Electric Furnaces for the Manufacture of Calcium Carbide and Ferrosilicon." (*Electrochem. and Met. Ind.*, Oct., 1908.) Translation of a paper by Dr. Walter Conrad, before the Verein Deutscher Eisenhüttenleute in Düsseldorf, describing numerous electric furnaces from the oldest to the most modern.

CALCIUM CHLORIDE.

This interesting industry was treated comprehensively in Vol. XVI, and is touched upon briefly in the article on "Bromine" on an earlier page of this volume. The principal producers are the Solvay Process Company, of Syracuse, N. Y.; Rhodes & Co. and P. Van Schaack & Son, of Mt. Pleasant, Mich.; and the salt manufacturers of the Ohio valley and Michigan. The production in 1907 was estimated at 45,000 tons, containing 40 per cent. $CaCl_2$, of which 15,000 tons was produced by salt manufacturers in the Ohio valley. The production in 1908 was about 32,000 tons (10,000 in Ohio, Pennsylvania and West Virginia). The average price in Ohio in 1907 was $9 per ton, f.o.b. cars; in 1908, it was about $7, having been $9 at the beginning of the year, but later on selling as low as $5.

The Solvay Process Company, of Syracuse, N. Y., is the largest manufacturer of calcium chloride in this country, if not in the world. The capacity of its plant for the manufacture of calcium chloride in its various forms is limited only by the amount that the market will absorb.

Experiments have been made with calcium chloride, as a preventive of dust on roads. The U. S. Office of Public Roads has used a solution prepared by mixing 300 lb. of commercial calcium chloride with 300 gal. of water. Each application consisted of 600 gal. of the solution over an area of 1582 sq.yd., or 0.38 gal. per sq.yd. The texture of the road surface was completely changed after the application of the calcium chloride. Before treatment, raveling was excessive in spots and the whole surface seemed loosely knit together. After the application this condition changed and the road surface became smooth, compact, and resilient. Such auxiliary sprinkling as was necessary consisted in the application of about 0.2 gal. of water per square yard at a time. The calcium chloride was charged at the rate of $16 per ton, f.o.b. cars at Baltimore. A freight charge of 13c. per 100 lb. weight is added to place the material on the ground, making the total cost of the chloride $18.60 per ton. The total cost of application was: 600 lb. calcium chloride, $5.586; 3 men, 1½ hours, $0.675; 1 horse sprinkling wagon, 1½ hours, $0.525; total (1582 sq.yd.) $6.786. Total cost of five applications was $33.90, or $0.0235 per sq.yd. Labor was paid 15c. per hour and 35c. per hour was paid for the sprinkling wagon.

The specific gravity of the solutions that were employed ranged from 1.053 to 1.060. Some variation was unavoidable, as the calcium chloride in certain of the barrels had absorbed a large amount of moisture from the atmosphere.

The addition of 2 per cent. of calcium chloride to a cement concrete is said to prevent its freezing while setting during cold weather without impairing its strength. More of the salt weakens the concrete.

CALCIUM CYANAMIDE AND CALCIUM NITRATE.

During 1908 the calcium cyanamide industry was introduced in North America. The American Cyanamid Company, with offices at 100 Broadway, New York City, is the exclusive owner of the rights for the manufacture and distribution of this material in the United States. The initial plant of this company is in course of construction at Niagara Falls, Ontario, and it is anticipated that production will begin early in the spring of 1909. Pending the completion of its plant, the company is filling orders with material imported from Norwegian works. On the basis of 10 per cent. nitrogen content, the initial capacity of the half-unit plant at Niagara Falls will be 20,000 tons per annum. This cyanamide will yield from 12 to 15 per cent. ammonia. It is the intention of the company to follow this plant with others as the demand requires.

The Niagara Falls plant will differ from some works now operating in Europe, in that it will make its product complete instead of obtaining calcium carbide from outside sources. The price of the finished product will be the same as that of Chile nitrate; in Europe the prevailing price is from 5 to 10 per cent. less.

The cyanamide which will be manufactured by the American Cyanamid Company is expected to have about the following analysis: Nitrogen (expressed as ammonia) 12 to 15 per cent.; free carbon, 10 per cent.; calcium sulphate (land plaster), 25 per cent.; sulphides, phosphides and carbonate, 2 per cent.; iron oxide, 2 per cent. In deference to the requirements of the American trade, free lime is to be eliminated, and the nitrogen content reduced as compared with the foreign product.

Cyanamide in Foreign Countries.

In Europe progress in the manufacture of calcium cyanamide has been rapid and satisfactory. There are 11 plants in process of construction, or in operation, the aggregate annual capacity of which is 166,000 tons. The important works at Piano d'Orta, Italy, is being enlarged to increase its production from 4000 to 10,000 tons. In France, the Société Française des Produits Azotés is producing 4000 tons per year at its works at Notre Dame de Briançon. Other works exist in Italy, Switzerland, Germany and England.

The following abstract of a paper on "Chemical Industry in Relation to Agriculture," by Prof. Adolf Frank, which was printed in full in *Journ. of the Soc. of Chem. Ind.* for Nov. 30, 1908, comprises a complete history of the chemistry and industrial status of calcium cyanamide and calcium nitrate.

Early Discoveries and Deductions.

After a long series of experiments, it was found that the whole amount of nitrogen contained in calcium cyanamide, as also that contained in the pure cyanamide, produced therefrom, could be easily converted into ammonia, by heating with water under high pressure; this is expressed by the following equations:

$$CaCN_2+3H_2O=CaCO_3+2NH_3$$

$$CN_2H_2+3H_2O=(NH_4)_2CO_3$$

The consideration of this simple reaction led my son, Dr. Albert Frank, to the conclusion that, under suitable conditions, crude calcium cyanamide might in itself represent a suitable fertilizer that could be used directly for the nutrition of plants. As this supposition was only based on theoretical observations, it became necessary to prove its correctness by practical tests. A great number of tests were made, under varying conditions, in pots, as well as in the open field, and the results proved that crude calcium cyanamide containing 20 per cent. of nitrogen could be considered equivalent to ammonium sulphate and therefore, if used in the right place, might serve as a suitable fertilizer for plants. These trials having been successfully repeated in agricultural practice, we had to develop the manufacture of the product—which we termed "nitrolime"—so as to enable it to enter into successful competition with ammonium sulphate and Chile saltpeter.

Raw Materials.—The raw materials required, viz., lime, coal and atmospheric nitrogen, are easily obtainable everywhere, but the production of pure nitrogen in such large quantities required further investigations. The process first used by us to separate oxygen from nitrogen by leading the air over heated copper turnings proved to have many drawbacks, because, in consequence of the necessary reduction of the oxide formed, work could only be carried on intermittently. We therefore turned our attention to the Linde method of separating the two constituents of our atmosphere by liquefaction and fractional distillation. This method also presented difficulties, for we had to devise first an expensive plant for the supply of the large quantity of pure nitrogen (300 cu.m. per hour)

which we required. In the course of the last years we have, however, successfully developed both methods of producing nitrogen on a large scale, so that both are applicable, the choice of the method depending in each case on local circumstances.

The second raw material required for the manufacture of calcium cyanamide is calcium carbide. One hundred kilograms of commercial calcium carbide—of 80 per cent. purity—can, theoretically, absorb 35 kg. of nitrogen. In practical working, the absorption amounts to 25 kg. only, so that the resulting product is 125 kg. of nitrolime containing 20 per cent. of nitrogen. Thus, in order to fix one ton of nitrogen, four tons of calcium carbide are required. The production of this quantity of calcium carbide forms, of course, the basis of the subsequent manufacturing operations. Carbide factories now produce two tons of carbide per kilowatt-year; consequently we require two kilowatts-year, equaling 2.66 h.p.-year for the four tons of carbide necessary to fix one ton of nitrogen. Adding to this amount 0.34 h.p.-year for other mechanical work, etc., we require a total of 3 h.p.-year. It is, therefore, evident that the cost of electric energy represents an important factor in the competitive struggle of the new product with the old nitrogenous manures, ammonia and saltpeter. In order to introduce nitrolime into agricultural practice with success, it is necessary to supply the unit of nitrogen at a price at least equal to, and if possible, lower than that of the unit contained in ammonia and saltpeter.

Installations.—Since water power, necessary for the production of energy at a low price, is scarce and can only be obtained at a high cost in Germany, the Cyanidgesellschaft combined with the largest Italian carbide works to be found, the Società Generale per la Cianamide, in Rome, to which all patents were transferred. The first plant equipped for production on a large scale was, at the end of 1905, started at Piano d'Orta, utilizing the large hydraulic power of the Pescara river; the production of this plant, laid out first for 4000 tons a year, has already been increased to 10,000 tons a year. Further large plants in Italy have since been erected at Terni, in connection with the carbide works which already existed there, and near San Michele in the Aosta valley. On the other side of the Adriatic, at Sebenico in Dalmatia, at Fiume, and especially at Almissa, where water power of 50,000 h.p. is available, some factories are being constructed, while others are already working. Licenses for France have been granted to the Société Française des Produits Azotés, which has erected a plant for 4000 tons a year, at Notre Dame de Briançon. In Switzerland, the Société Suisse des Produits Azotés has begun manufacturing in works situated in the Rhone valley. In the United States, the American Cyanamid Company has erected, on the

Canadian side of the Niagara Falls, a factory for an output of 5000 to 6000 tons, which when the installation is complete, will be increased to 10,000 tons a year. I may also mention that a plant is now in course of construction on the Kiuskzu islands, in Japan; it is designed to produce, for the present, 4000 tons a year. In Great Britain, the North Western Cyanamide Company, Ltd., was founded in 1906. This company acquired from the Birkeland-Eyde concern water power, to the extent of 50,000 h.p., near Odde on the Hardanger Fjord in Norway and established there a plant for the production of 12,500 tons of nitrolime to begin with, an extension to 50,000 tons a year being provided for. In Germany, owing to the want of cheap water power, only two small factories, for a production of 2000 tons and 6000 tons respectively, have been started; there will, however, soon be added a third plant for 15,000 tons a year, now in the course of erection in Bavaria, which will utilize the water power of the Alz. When all the above plants are at work, their total production will correspond to the nitrogen contained in 250,000 tons of Chile saltpeter.

Utilization of Bogs.—The thought that Germany, the largest consumer of nitrogenous manure, would have to import nitrogenous manure, as she is now importing Chile saltpeter, acted as a strong stimulus to discover other hitherto unused sources of power. The solution of this problem, so important in many respects from the economic standpoint, has now been found by utilizing the large stores of energy accumulated in the German bog areas. As Germany owns an area of 22,000 sq.km. (or 8500 sq. miles) of bogs, she can not only produce the nitrogenous manure required for her agricultural requirements, but she can, in addition, supply large quantities of power for other industrial and agricultural purposes. Similar conditions obtain in Great Britain, especially in the Irish bogs. Ireland, for her area, owns more peat deposits than any other country in the world; almost one-seventh of her surface, viz., 4250 sq. miles, is covered with peat deposits of a quality rarely found elsewhere in Europe, and the percentage of nitrogen in the peat is often very considerable. The zealous endeavors made for years to improve Irish agriculture and industry, can therefore be assisted to a great extent by the rational utilization of the bogs.

Nitric Acid and Nitrates.—More important still than the fixation of nitrogen in the shape of calcium cyanide, is the application of large and cheap sources of energy for the production of nitric acid and nitrates direct from the atmospheric nitrogen by means of an electric arc. This process is a thermal one, but while the combination of nitrogen with calcium carbide represents an exothermic process (i.e., heat is set free), the combustion of nitrogen with oxygen is an endothermic process. Conse-

quently, the consumption of energy is considerably larger in the latter process, amounting theoretically to nearly 28 times the consumption of the former process per unit of nitrogen fixed. In practice, according to Eyde's own statement, the consumption is even higher. The product obtained by the Birkeland-Eyde process, namely, nitrate of lime, has proved itself to be, for the same percentage of nitrogen, a manure fully equal to nitrate of soda. The employment of neutral nitrate of lime is, however, attended with some difficulty on account of the hygroscopic property of this salt. Hence, it became necessary to manufacture the less hygroscopic basic nitrate, which in its commercial form contains only 11 to 12 per cent. of nitrogen. Other methods for the production of nitric acid, by combustion of ammonia with oxygen, wherein the necessary heat is produced by the oxidation of the hydrogen, have lately been worked out by Prof. Ostwald of Leipzig, as also by Doctor Caro and myself. Ostwald's process is based upon the catalytic action of platinum, while Caro and myself obtain the same effect by using a mixture of thorium and cerium oxides. The products obtained in the experiments we have carried out, on a small scale, have utilized as much as 85 per cent. of the nitrogen contained in the ammonia. Large plants are said to have been erected to work Ostwald's process, but reports as to their practical results are, so far, wanting. Considering that, as mentioned already, by treating nitrolime with steam under high pressure, pure ammonia can be easily produced with a yield of 98 per cent., it is quite within the limits of possibility that our process may also become a useful one for the production of nitric acid. Meanwhile the production of ammonia on a large scale with the aid of nitrolime is carried out only in the Piano d'Orta works for the manufacture of ammonium sulphate. As a matter of fact, the fertilizing effect is brought about by the transformation in the soil of the amide into nitric acid, this process of nitrification being effected, as has been ascertained by Löhnis and Perotti, by certain bacteria which abound in all soils that are rich in humus.

Use in Agriculture.—The calcium cyanamide supplied for agricultural purposes contains an average of 20 to 21 per cent. of nitrogen and 40 to 42 per cent. of calcium, corresponding to 56 to 59 per cent. of caustic lime; further, 17 to 18 per cent. of carbon and some impurities consisting of mixtures of silica, clay, traces of phosphoric acid, etc., originally contained in the crude carbide. When this material is mixed with the cultivated soil, the calcium cyanamide, under the influence of the carbon dioxide and of the moisture contained in the soil, is decomposed first into calcium carbonate and free cyanamide. The latter then, with absorption of moisture, is probably transformed into urea: $CN_2H_2+H_2O=(NH_2)_2CO$, from which ammonia, and by subsequent nitrification, nitric

acid is formed. The chemistry of this process of transformation has not yet been fully worked out, but nitric acid is acknowledged by all investigators to be the final product. Thus, while nitrate of soda, and nitrate of lime offer the nitric acid to the plant ready for absorption, nitrolime must first be transformed into nitric acid in the soil, in the same manner as are the ammonia salts and nitrogenous manures of animal origin. This feature constitutes a guarantee that both kinds of nitrogen manures, nitrates as well as amides, will be used in future side by side. Saltpeter will always be used, especially as top dressing, in all cases where vegetation must be rapidly assisted, i.e., especially in spring, while nitrolime must be introduced into the soil before the seed is sown in order to enable it to show an effect equal to that produced by ammonium sulphate. Moreover, it offers the advantage that its decomposition products are absorbed and retained by the soil, whereas in moist years the nitrates are washed out and carried away. Only prolonged practical experience can supply a knowledge of the most favorable conditions for the use of this new manure in different soils and for the different plants cultivated. The same thing occurred in connection with the introduction of potash manures, when their high chlorine content at first gave rise to apprehensions.

The Calcium Nitrate Industry in Norway.

Sam Eyde, in a recent lecture, described the status of the "saltpeter" industry at Notodden, in Telemarken, Norway. He traced the undertaking from its conception, and, throughout the experimental stages, to its present maturity under the auspices of the Badische Anilin und Sodafabrik, whose rival method, together with capital requirements by the Norwegian promoters, necessitated an assimilation of interests. The Birkeland-Eyde nitrate industry at Notodden is epoch-making, in that it is the pioneer of its kind; from beginning to end it has been planned and carried out by Norwegians. Owing to breakdowns at the power station and factory, an average use of 13,000 kw. only was obtained in 1908, whereas the normal should have been 22,000 kw. In the last six months, allowing for occasional shortages of water, the average was 19,500 kw., with a revenue of about 1,500,000 kroner, and a cost of 970,000 kroner, which for the six months gives a net profit of 530,000 kroner. In experimental operations 1,000,000 kroner has been expended. The total capital in connection with the Notodden and Svölgtos construction was 11,000,000 kroner. The revenues will be further increased as soon as arrangements for the special production of ammonia nitrate and lime saltpeter as well as the better regulation of the watersheds, are completed. Herr Eyde

anticipates that with the further damming of the auxiliary falls, which would occupy 10 years, at a cost of 150,000,000 kroner, the yearly output of Norwegian nitrate would amount to 300,000 tons. Chile at present produces 1,800,000 tons annually—say, in 10 years' time (1920), 2,500,000 tons. As Norway's production in 1920 will have reached about 300,000 tons, to the value of 45,000,000 kroner yearly, it will only amount to about 12 per cent. of Chile's output in the same period. Chilean nitrate has to undergo a chemical process before it reaches the consumer, and climatic and labor drawbacks will in a high degree tend to prevent prices falling to a level that would threaten the Norwegian industry. Experiments with the practical use of Norway saltpeter, have been carried out by authorities on agriculture in Norway, Sweden, Denmark, Germany, France, England and Scotland, Austria, Hungary and Italy, and results bear out the favorable opinion expressed by specialists on the first appearance of the fertilizer, viz., Norway saltpeter is equally as good as that from Chile; and in soil poor in lime it is even better.

The three companies connected with the Notodden-Rjukanfos power and nitrate production are: the Norwegian Power Company, capital 16,000,000 kroner; the Norwegian Saltpeter Works, capital 18,000,000 kroner; and the so-called Norwegian Transport Company, capital 3,000,000 kroner, which was formed to connect Rjukanfos, Notodden and Skein by rail and water.

The companies interested in the Birkeland-Eyde process have reached an understanding with the Badische Anilin und Soda Fabrik which has developed another process. The results which will be obtained at an experimental plant now in course of erection at Notodden will decide whether the furnace of the Badische company or the furnace of Birkeland and Eyde will be used for the large plant at Rjukanfos for which money has been procured by the co-operation of the two companies.

CARBORUNDUM.

By F. J. Tone.

The production of carborundum in the United States in 1908 was 4,907,170 lb., the Carborundum Company, of Niagara Falls, being the only producer. The considerable reduction in the output as compared with that of 1907 was due to the general depression in the metal and manufacturing trades, in which the demand for grinding materials of all classes was subject to large curtailment. The relative progress of the industry may be seen from the accompanying table:

PRODUCTION OF CARBORUNDUM IN THE UNITED STATES.

Year.	Pounds.	Metric Tons.	Value.
1891	50		
1892	2,145	1	
1893	15,200	7	
1894	52,190	24	
1895	225,930	102	
1896	1,190,600	540	$365,612
1897	1,242,929	564	153,812
1898	1,594,152	724	151,444
1899	1,741,245	791	156,712
1900	2,401,000	1,089	168,070
1901	3,838,175	1,742	268,672
1902	3,741,500	1,698	261,905
1903	4,760,000	2,160	333,200
1904	7,060,380	3,203	494,227
1905	5,596,280	2,539	391,740
1906	6,225,280	2,824	435,770
1907	7,532,670	3,418	451,960
1908	4,907,170	2,226	294,430

During 1908 no increase was made in the plant nor were any changes of importance introduced in the methods of manufacture. The present type of furnace is 30 ft. in length and 12 ft. in width. The end walls containing the electrode terminals are built of concrete and the side walls are composed of several sectional units, each section being built of fire brick set in iron frames. This facilitates the discharging and reloading of the furnaces and saves considerable labor. The furnaces are loaded by means of an overhead conveyer. Each furnace consumes 1600 k.w. with a maximum current of about 20,000 amperes and produces about 15,000 lb. of crystalline carborundum in each run. The transformers receive the primary current at 2200 volts and 25 cycles and are provided with regulators which vary the secondary voltage from 200 to 75 volts.

The abrasive uses of carborundum are constantly extending, although the manufacture of wheels and sharpening stones constitute the main portion of the industry. It enters into an infinite variety of grinding operations upon almost every substance which requires to be shaped or polished. The diversified properties of carborundum are illustrated by the new and striking uses which constantly arise for it. Non-slipping stair treads in public stairways and building entrances subjected to heavy travel are widely used, the method in this country being to insert in iron or steel plates strips of carborundum grains set in a matrix of cement. In France carborundum grains are now used as a top dressing in cement blocks and walks; a careful observation of three steps in the Gare-de-Lyon, a Paris railway station, shows no perceptible wear in 14 months after the passage of fourteen million travelers.

Electrical Uses.—Carborundum crystals are largely used as detectors in wireless telegraph apparatus. A crystal clamped between two metal surfaces forms a very sensitive responder and one which is always in operative position. A French engineer, M. Rouind, has recently devised a new form of carborundum receiver of still greater sensitiveness in which light carborundum powder is placed in a cylindrical glass tube between two cylinders of copper, which are connected one to the aerial and one to the ground.

The electrical conductivity of carborundum through a wide temperature range has never been determined in an exact manner. It is classed with conductors of the second class, its conductivity rising with increased temperature. Compacted carborundum for use as resistance units is made by several processes. Rods[1] of molded grains are made and given a second furnace treatment in which intersticial carborundum is formed, closing up the pores and producing a dense conductive rod. Silundum[2] is a name given by a German engineer to a variety of this material. It is stated that it is successfully applied for electric cooking and heating purposes and replaces platinum in a large number of similar appliances. The rods are formed in lengths up to 800 mm. These resist action of the air up to temperatures of 1600 deg. C. In the electric cooking and heating industries, which have developed rapidly in recent years there is required a concentration of electrical energy to be converted into heat, which at present is difficult to obtain without overheating the resistors. Ranges using compacted carborundum resistors have been made, that have the glowing heat of a coal fire.

Refractory Uses.—A study of the heat conductivity of refractory materials has been made by S. Wologdine of the Société d'Encourage-

[1] F. J. Tone, U. S. Patent, 913,324.
[2] *Electrochem. and Met. Ind.*, January, 1908.

ment pour l'Industrie Nationale.[1] Expressing the conductive coefficient in the number of gram calories passing through a cubic centimeter of the material in one second with a variation of one degree per linear centimeter at a temperature of 1000 deg. C., he finds the coefficient of carborundum to be 0.0150; that of crucible graphite 0.00145; and that of clay 0.0003. The porosity of the samples was not stated. This high value having been determined for carborundum it is proposed to utilize the material in the construction of muffle furnaces where great advantage is gained both in the thinness of the walls and the rapidity with which conditions of equilibrium can be attained between the temperature of the hearth and the walls of the muffle.

Carborundum is not acted upon in a neutral or reducing atmosphere up to the temperature of dissociation which is 2220 deg. C. On the contrary, oxidizing gases, such as air, as well as carbon dioxide and water vapor above the temperatures at which they dissociate, exert on carborundum an oxidizing action, the intensity of which depends on the temperature. A research on the oxidation of carborundum has shown that no action in pure oxygen occurs up to 1000 deg. C. When further heated in air from 1000 to 1350 deg., a slightly noticeable superficial oxidation was shown by the hardening of the walls of the crucible which constituted the test piece. At 1500 deg. a thin transparent layer of fused silica is formed. This becomes more regular at 1700 deg. and at 1750 deg. it flows freely and oxidation proceeds rapidly. Notwithstanding these facts, carborundum bricks may be used in certain places under oxidizing conditions with great success. This is due to the fact that in oil-burning or coke furnaces, operating at temperatures of 1600 to 1800 deg. C., suspended particles of silica, alumina, iron and other bases adhere to the surface of the brick and saturate the superficial layer of carborundum, while reacting slightly with it. A brown enamel is produced, forming a layer impervious to gas. The under layer of carborundum thus protected retains its high refractory nature as it would in a reducing atmosphere. The enamel by reason of a particular reaction with the carborundum has its fusion point raised and remains stable when the ashes forming it completely fuse.

Technology.—The formation of carborundum in the electric furnace under high temperatures has recently been studied by Dr. R. S. Hutton and J. E. Petavel.[2] The pressures employed were varied, the highest being about 100 atmospheres. It was found that the formation of crystalline carborundum was considerably impeded at the highest pressures and that of amorphous carborundum was almost negligible. This may be

[1] *Revue de Métallurgie*, February, 1909.
[2] Philosophical Transactions, Royal Society of London, Series A, Vol. 207, pp. 421-462.

explained by considering the important part which silica vapors play in the reactions in the carborundum furnace. Silica has very high vapor tension. Its vapor tension is appreciable even below its melting point and when pressures were attained which lowered the vapor tension of silica appreciably the speed of the reaction was correspondingly reduced, or in some cases the reaction was altogether prevented.

H. C. Greenwood[1] has studied the action of carbon on silica working in a vacuum. This method enabled him to determine the exact point at which reduction began, which was simultaneous with the evolution of carbon monoxide. He observed that reduction begins at 1460 deg. C., producing silicon carbide and oxy-carbide of silicon.

In a study of the compressibility of the elements and their periodical relations by T. W. Richards and W. N. Stull,[2] the compressibility of silicon was found to be not over 0.16×10^{-6}, silicon being the least compressible of the 35 elements thus far investigated. Carbon in the state of graphite was found to be filled with minute pores which interfered with the exact determination of its compressibility. Nevertheless, this quantity may safely be said to be not greater than 3×10^{-6}. The compressibility of carborundum (SiC) was found to be 0.21×10^{-6}, but little greater than that of silicon. From these results and from other compressibilities it is concluded that the compressibility of diamond can not greatly exceed 0.5×10^{-6}.

A recent investigation by J. N. Pring[3] on the formation of carbides, discusses the formation of carbide of silicon. The direct union of pure silicon and carbon in a vacuum commences between 1250 and 1300 deg. C., the reaction proceeding rapidly above 1400 deg. Commercial silicon containing 5 per cent. of iron and 0.7 per cent. of aluminum reacts with carbon at all temperatures above 1200 deg. The presence of iron does not apparently facilitate the reaction, nor does carbon monoxide exert any influence between the limits of atmospheric pressure and 0.03 mm.; hence, already below its melting point, silicon is proved to combine with carbon.

Foreign Producers.—In Europe the oldest plant producing carborundum is operated by the Austrian Carborundum Company at Benateck, Bohemia; it was started in 1893. This company has also since 1894, operated a plant at La Bathie, Savoy, France. The French plant announces that its production is to be considerably increased. A German company manufacturing carborundum at Badisch-Rheinfelden has constructed a plant of 3000 h.p. capacity. In Italy a plant is under construction at Bodio, Turin. A small amount of carborundum is also manufactured in Sweden. The American company for the past two years has had in

[1] Trans. (British) Chemical Society, 1908, Vol. 93.
[2] Carnegie Institution of Washington, Pub. No. 76.
[3] Trans. (British) Chemical Society, 1908, Vol. 93.

operation a plant at Düsseldorf-Reisholz, Germany, for the manufacture of grinding wheels and sharpening stones and from this factory supplies most of the Continental trade.

Recent Patents.—Robert C. Totten (U. S., 881,038) patents a new composition of chilled cast iron consisting of coke or coal iron and silicon carbide. The process is designed to utilize cheap pig iron in place of the more expensive charcoal iron for obtaining grades suitable for chilled rolls and other chilled castings.

E. F. Price and F. M. Becket (U. S., 891,565) describe a process for the production of ferro-chrome and similar ferro-alloys, in which there is used as reducing agent a mixture of silicon carbide and calcium carbide. Great advantage lies in the fact that the oxidized silicon and calcium unite to form a fusible slag.

Henry S. Hatfield and Frank M. Lewis (U. S., 904,532) patent an arc lamp electrode in which 5 to 10 per cent. of carborundum and 25 per cent. calcium fluoride are incorporated in the electrode to improve the steadiness of the light.

F. M. Becket (U. S., 906,854) covers a process of producing ferro-chromium, ferro-vanadium and other ferro-alloys in the electric furnace using silicon carbide as a reducing agent. A molten bath is formed of the ore with silicon carbide and the bath is heated by the passage of the current.

F. J. Tone (U. S., 908,357) describes an improvement in the production of silicon carbide in which the charge is heated by combustion gases or other means previous to its treatment in the electric furnace. The object of the process is to utilize the cheaper form of combustion heating during the initial period of operation and to complete the reaction with electrical heating where the very high temperatures are required. As the cost of electrical energy constitutes a very large item in the total cost of carborundum a large decrease in the cost of production should be obtained by a successful application of this method.

Frederick Bölling (German patent, 204,123, March 24, 1907) patents an electric heating element composed of carbon, carborundum, carbide, or the like. Resistance units are made in the form of a bar chain, the several parts being united at the enlarged ends by a binder fused in place of the oxyhydrogen flame and composed of silicon, 90 per cent.; titanium oxide, 3 per cent.; kaolin, 2 per cent.; feldspar, 1 per cent.; boric acid, 2 per cent.; lime, 1 per cent.; and silicon carbide, 1 per cent.; powdered and mixed with water.

F. J. Tone (U. S., 913,324) describes the manufacture of dense silicon carbide for use as electrical resistance units. A porous mass of silicon carbide is heated in the electric furnace and subjected to the action of vapors of silicon and carbon.

CEMENT.

By Robert W. Lesley.

Commercial depression was the characteristic of 1908, especially with respect to many of the materials of construction. Consequently it was with considerable surprise that the preliminary figures of the U. S. Geological Survey, covering the cement production for 1908, were received.

Figures show an actual increase in the output of portland cement in 1908 over 1907. Early returns based upon estimates from eastern and southern mills indicated a decrease in the output, but the prosperity in the West and the increasing demand for portland cement caused a larger production in that territory.

From the results of 1908, it would seem fair to say that the extensive publicity given to the numerous new uses of portland cement and the widespread desire on the part of consumers and contractors all over the United States to try the new material of construction and to find new outlets for its consumption, have borne fruit.

The increase in the portland cement production of 1908 was barely 5 per cent., a considerably lower rate of growth than has marked the figures of previous years. A decrease is noticeable in Pennsylvania, New Jersey and Michigan, which was offset by remarkable gains in the middle West and on the Pacific Coast, principally in Indiana, Illinois and California. It is an interesting fact that while the Lehigh district, which has been the Pittsburg of the industry, showed in 1907 (a very busy year in the trade) a slightly increased percentage over 1906, when for the first time it suffered a retrogression, that in 1908, when the industry showed a gain in the country at large, the Lehigh-Pennsylvania-New Jersey district showed a falling off in percentage with respect to the total output of the country. This seems to point to the conclusion that with the growth of new plants elsewhere in the country, the distributing area of the Lehigh district is gradually becoming restricted, and that while its excellent material, its admirable labor conditions and the reputation and quality of its product may for many years keep it in the forefront of cement production, yet in view of the fact that the price of cement is the cost at the mill plus the freight to the consumer, it is not an unlikely thing to expect that the area in which cements from the

Lehigh district can be distributed will be gradually lessened from year to year.

The same condition will in time apply also to the Kansas field, where a large group of works has been established at points where natural gas and good raw materials are found. The erection of works further West is certain to bear harshly upon this particular field.

Another interesting point in the statistics for 1908 is that the average price at the mill of the entire output of portland cement in the United States was only 85c. per bbl., or 36c. below the average of 1907. The lowest price previously reached was in 1904, when the net return was only 88c. As a result of this great reduction in price, promotions of

PRODUCTION OF CEMENT IN THE UNITED STATES. (a)
(In barrels.)

Year	Portland.			Natural Hydraulic.			Puzzolan Cement.			Total.	
	Barrels.	Value.	Per bbl.	Barrels.	Value.	Per bbl.	Barrels.	Value.	Per bbl.	Barrels.	Value.
1898	3,584,586	$ 6,168,106	$1.72	8,168,106	$3,819,995	$0.47	157,662	$ 235,721	$1.50	11,903,326	$10,223,822
1899	5,805,620	10,441,431	1.80	9,686,447	5,058,500	0.52	244,757	360,800	1.47	15,736,824	15,780,789
1900	8,482,020	9,280,525	1.09	8,383,519	3,728,848	0.45	446,609	567,193	1.27	17,312,148	13,576,566
1901	12,711,225	12,532,360	0.98	7,084,823	3,056,278	0.43	272,689	198,151	0.73	20,068,737	15,860,731
1902	17,230,644	20,864,078	1.21	8,044,305	4,076,630	0.50	478,555	425,672	0.81	25,753,504	25,366,380
1903	22,342,973	27,713,319	1.19	7,030,271	3,675,520	0.50	525,896	542,502	1.03	29,899,140	31,931,341
1904	26,505,881	23,355,119	0.90	4,866,331	2,450,150	0.50	303,045	226,651	0.75	31,675,257	26,031,920
1905	35,246,812	33,245,867	0.94	4,473,049	2,413,052	0.54	382,447	272,614	0.71	40,102,308	35,931,533
1906	46,610,822	51,240,652	1.10	3,935,151	2,362,140	0.60	481,224	412,921	0.86	51,027,321	54,015,713
1907	48,785,390	53,992,551	1.10	2,887,700	1,467,302	0.51	557,252	443,998	0.79	52,230,342	55,903,851
1908	51,002,612	43,472,679	0.85	1,621,862	808,509	0.49	151,451	95,468	0.63	52,775,925	44,376,656

(a) Statistics of production for 1900 and subsequent years are as reported by the U. S. Geological Survey. The barrel of portland cement contains 380 lb. of the material; of natural cement, 265 lb.; of slag cement, 330 lb.

STATISTICS OF CEMENT IN THE UNITED STATES.
(In barrels.)

Year.	Production.		Imports.		Exports.		Consumption.	
	Barrels.	Value.	Barrels.(a)	Value.	Barrels.	Value.	Barrels.	Value.
1898..	11,903,326	$10,223,822	2,119,880	$2,624,228	55,969	$ 98,121	13,967,237	$12,749,929
1899..	15,736,824	15,860,731	2,219,246	2,858,286	116,079	213,457	17,839,991	18,505,560
1900..	17,312,148	13,576,566	2,512,300	3,330,445	147,305	289,186	19,677,143	16,617,825
1901..	20,068,737	15,786,789	994,624	1,305,092	303,380	752,057	20,759,981	16,340,424
1902..	25,735,504	25,366,380	2,100,513	2,582,281	367,521	575,268	27,486,496	27,373,393
1903..	29,899,140	31,931,341	2,439,948	3,027,111	312,163	466,140	32,026,925	34,492,312
1904..	31,675,257	26,031,020	1,101,361	1,383,044	816,640	1,158,572	31,959,978	26,256,392
1905..	40,102,308	35,931,533	891,134	1,102,041	1,060,054	1,428,489	39,933,308	35,605,085
1906..	51,027,321	54,015,713	2,321,803	2,950,268	600,386	964,373	52,748,738	56,001,608
1907..	52,230,342	55,903,551	2,006,228	2,637,424	900,550	1,450,841	53,336,020	57,090,434
1908..	52,775,925	44,376,656	839,247	1,189,560	846,528	1,249,229	52,768,644	44,316,987

(a) Barrels of 400 lb.

cement mills were materially lessened. The method of quotation for deliveries of cement, practiced generally throughout the trade, of including in the selling price the price of the bags, gives a selling price which is frequently used by promoters to indicate the figure cement to be produced in a new plant can be sold for. Thus the selling price of cement during

1908 averaged about $1.25 per bbl. at the mill, including the bags. The bags are charged at 40c.; therefore, the $1.25 price indicates a price of 85c. at the mill. This condition is prevalent all over the United States, cement being usually sold upon a basis of 376 lb. to the barrel, in bulk, at the mill.

The result of the very low prices for portland cement is shown in the figures for natural cement, which fell off from 2,887,700 bbl. in 1907 to 1,621,862 bbl. in 1908, and in the figures of puzzolan cement, which show a falling off from 557,252 bbl. in 1907 to 151,451 in 1908. Both of these cements have proper and legitimate markets, if not a large field, in good times, but in view of the extremely low price of the higher grade portland cements, they found few customers in 1908, as the average buyer was always ready to purchase portland cement when he could obtain it at the low figures prevailing. Similar conditions are shown to have governed the imports of cement. Some companies devoting attention to the foreign trade found during the year 1908 a fairly good market for some of their surplus, but other companies, not so fortunate, were forced to shut down some of their plants during a considerable period of the year.

Industrial Conditions.—The Association of Licensed Cement Manufacturers, which was organized in New York in January, 1908, for the general betterment of the mechanical and chemical processes used in making cement, the improvement of the quality of cement, dealing with matters of traffic and shipment and the establishment of an association laboratory for technical tests and experiments, held many meetings during the year with a view to carrying out the purposes for which it was organized. Under its auspices a great number of publications were circulated, covering the manifold new uses of cement and other subjects of general interest to the industry. The Association represents the licensees of the North American Portland Cement Company, the holder of the Hurry & Seaman patents for burning pulverized coal in rotary kilns, as stated in THE MINERAL INDUSTRY for 1907. During 1908 there was considerable litigation under the patents referred to against alleged infringers.

New Plants.—In 1908 there was but a small development in the construction of new plants. Business conditions were such that not only were plants already under construction greatly delayed, but new ones were made practically impossible by reason of the existing financial depression. Several works in California were started early in 1908 but construction was abandoned. New owners took over the Devil Slide works in Utah. These works were brought into operation in 1907. The rapid construction of projected works in Iowa was held over during

1908, and many of the proposed enterprises in Kansas and Oklahoma were also delayed in construction. In the East several projected enterprises in the Lehigh district were held up, one large plant being at the present time nearly two years in construction. On the Hudson river also two plants have been delayed so far as rapid building is concerned.

Materials.—In the development of the industry a tendency has been shown to increase the magnesia content, following the German specification, and this has finally been fixed at 4 per cent. by the specification of the American Society of Civil Engineers to correspond with that of the American Society for Testing Materials. The German specification also fixes the magnesia content at 4 per cent. The modifications in the German specification for portland cement change the sulphur trioxide content from 1.75 to 2.5 per cent. This is due to the introduction of the rotary kiln in Germany, and the larger percentage of sulphate of lime which the German manufacturers find is required to make their cement slow-setting. An increase in sulphur trioxide from 1.75 to 2 per cent. has been proposed to the committee having charge of the standard specification of the American Society for Testing Materials.

Machinery.—The length and diameter of rotary kilns seem to show no retrogression in either respect. Longer kilns than were ever contemplated are now in the course of erection, and some with diameters as high as 12 ft. are either in use or projected in modern cement mills. The question of coal consumption in these large kilns is one for the engineer of the future to figure out, but there seems to be no doubt as to the enormous capacity, in barrels that one of these kilns is capable of producing. The Fuller mill has been developed in the East and made a much more efficient machine for grinding. A new form of Kent mill has also been put on the market. The grinding of raw materials is attracting more and more attention in view of the development of so many new deposits with varying chemical and physical characteristics in the materials to be used. Tube and ball mills are still being largely used both in the preparation of the raw material and on the finishing side.

Engineering Investigations.—Congress appropriated the sum of $100,000 for the further investigation of structural materials under the auspices of the Geological Survey and National Advisory Board on Fuels and Structural Materials. The laboratory, which has been at St. Louis for many years, was removed in the latter part of 1908 to Pittsburg, Penn. The addition to the laboratory of an immense testing machine costing nearly $100,000 will prove an important aid to the development, on a large scale, of all kinds of concrete structures. Another interesting feature in connection with the Government work is the establishment

on Young's Pier, at Atlantic City, of a salt water testing station, under the same auspices as the testing laboratory at Pittsburg above mentioned. Both of these laboratories are in charge of Richard L. Humphrey. Many bulletins have been issued giving records of the work done, and many more are in course of preparation, which will add materially to the world's knowledge on the subject of the proper use of ordinary and reinforced concrete. For 1909 Congress appropriated another $100,000 for the work of these laboratories.

A Progress Report has been made by the Committee on Concrete and Reinforced Concrete, composed of members of the American Society of Civil Engineers, the American Society for Testing Materials, the Association of American Portland Cement Manufacturers, the American Railway Engineering and Maintenance of Way Association, and the American Institute of Architects. This report, which is in the nature of a preliminary statement, embodies the work of the committee for several years past, and gives a summary of many of the principles governing proper construction in concrete and the proper treatment of that material.

Market Conditions.—Emerging from the panic and facing the presidential election, the country was in no state to offer any great promise to the cement manufacturer during 1908. The figures showing the slight increase in the output were, as already stated, surprising to both producer and consumer. The market opened with a considerably reduced price and a largely reduced demand. Many of the mills found the demand so slack and the stocks on hand so large that shut-downs were forced upon them in the early spring. The impending presidential election and the slow recovery of confidence postponed building operations and no manufacturing enterprise had business enough to warrant any increase of its capacity; consequently the usual development of increased business in the fall of 1908 did not appear in the East, though a successful crop season in the West and the accumulation of large sums of money in the hands of the farmers, did cause a spurt in the autumn in that territory. There seems to be no question that 1908 demonstrated, practically, what the figures in all the cement directories had already set forth, namely, the fact of a large excess of producing capacity in the portland cement industry. This condition is likely to influence 1909. The year 1908 closed with large stocks of cement on hand and a very limited demand. The early months of 1909 show little, if any, increase in the consumption of cement.

Review by States.

Alabama. (By Eugene A. Smith.)—During 1908 only two plants making portland cement were in operation in Alabama, viz., the Stand-

ard Portland Cement Company at Leeds, in Jefferson county, using Trenton limestone and carboniferous shales; and the Alabama Portland Cement Company at Demopolis, Marengo county, using the Selma chalk and a residual clay from the same. Both plants suffered from the general business depression, and their total production was not more than 135,000 bbl. The projected portland cement plants at Village Springs, in Blount county, and at St. Stephens, in Washington county, have not yet begun operations. The puzzolan cement industry, like others, suffered from the effects of the panic. The production for 1908 in Alabama will hardly exceed 60,000 barrels.

Illinois. (By Frank W. DeWolf.)—The increase of 57 per cent. in the cement production of Illinois during 1908 is surprising when the general business depression is considered. It is in part explained, however, by the decrease of 35 per cent. in average price per barrel, and it is evident that certain necessary construction work was continued and stimulated by the general low price of building materials. The cement industry has changed but little from the conditions described in the report for 1907. The increased production and the strength of the industry lie in the manufacture of portland cement from limestone and shale, with coal for fuel. The use of limestone and blast-furnace slag, however, is continued successfully by the Universal Portland Cement Company in Chicago. Natural or hydraulic cement was produced by one company while two similar plants were idle and production declined.

The production of portland cement for 1908, according to statistics collected by the State Geological Survey and the U. S. Geological Survey, exceeded 3,200,000 bbl., at an average value of 85c. at the plants. The output of the previous year was 2,036,093 bbl., and the average price, $1.29. The increase was in part due to the operation after March 1 of the new Sandusky plant, at Dixon, and the enlarged equipment of the Chicago Portland Cement Company's plant at Lasalle. This latter company now operates six kilns 136 ft. long. Each of the other Illinois plants has six kilns measuring 60 ft. The limestone used at the Dixon plant is of Trenton age, while that of the other operators is a bed near the middle of the Coal Measures. The following list names the operating companies, and indicates the approximate daily capacities of the plants: Marquette Cement Manufacturing Company, 3000 bbl.; Chicago Portland Cement Company, 3000 bbl.; Sandusky Portland Cement Company, 2000 bbl.; German-American Portland Cement Company, 1800 bbl.; Universal Portland Cement Company, 1500 bbl.

Hydraulic or natural cement has been manufactured many years at Utica, and the interesting process is described in a recent report[1] by

[1] *Bull.* No. 8, State Geol. Surv. of Illinois

G. H. Cady. The Lower Magnesian limestone, which is obtained by underground mining, has the proper composition for cement manufacture. The industry has declined of recent years. Two of the plants changed hands in 1908 and only that of the Utica Cement Manufacturing Company continued in operation. This company appears to market its product with profit, and expects a reasonable demand for hydraulic cement to continue.

The undeveloped cement resources of Illinois are immense and their favorable location with respect to fuel, to transportation, and to large city markets promises a tremendous growth of the industry. The temporary postponement, however, of a number of projects will doubtless follow the present reduction in price of output and the increased competition in the Chicago and St. Louis markets. A report of the State Geological Survey on cement materials of Illinois is now ready for the printer. It is based on field and laboratory studies during which samples from 120 promising localities were collected and analyzed. Favorable materials were found in half of the places.

In the geologic succession of rocks in Illinois, from oldest to youngest, the limestones become less and less prominent. Limestone composes nearly half of the total thickness of the Ordovician rocks, all of the Silurian, and considerable portions of the Devonian and Mississippian, but in the Pennsylvanian they are inconspicuous. The Ordovician and Silurian limestones, being mostly magnesian, are not suited to portland cement manufacture. These occupy the northern counties of the State. The Devonian occupies limited areas in the West and South. The Mississippian beds have good average purity and outcrop extensively in the western and southern counties. The Pennsylvanian limestones are thin, but are locally of much commercial value, as at Lasalle and Fairmont. According to the report of E. F. Lines, high-grade materials will be available very commonly throughout the State, and the ultimate location of plants will depend chiefly on transportation facilities to the commercial centers.

Mississippi.—The first bulletin of the Mississippi State Geological Survey, Albert F. Crider, director, deals with the cement materials of the State. The bulletin contains many analyses, and, in conclusion, discussing advantageous situations for cement plants, says:

"The Southern Railway from Memphis to Chattanooga passes near the northern outcrop of the oolitic limestone to Tishimingo county near where the road crosses Bear creek. At this point it is only eight miles to the Tennessee river. The largest boats of that river run as far as the mouth of Bear creek. A cement plant built at this point would have an outlet to the north by boat and a railway connection to the East, West

and South. Coal could be obtained by river or from the nearby Alabama fields at a minimum cost. A portland cement plant at Mingo bridge could use the oolitic limestone and the overlying shale. Bear creek furnishes sufficient water to run a mill by water-power. The newly-constructed line of the Illinois Central Railway, connecting Birmingham, Ala., and Jackson, Tenn., with an outlet to the North and South, runs within three miles of this place.

"The Selma limestone and Porter's creek clay are in proximity along the Mobile & Ohio Railway in Kemper county, along the Illinois Central in Oktibbeha county, and along the Southern in Southern Clay county. The Mobile & Ohio furnishes an outlet to the North and South. The Southern line from Greenville, Miss., to Birmingham, Ala., offers an outlet to the East and West. The Aberdeen branch of the Illinois Central connects with the main line from Louisville to New Orleans at Durant, thus giving an outlet into a new territory.

"The Vicksburg limestone outcrops in the banks of Pearl river at Bryam, in the railway cut one and one-third miles south of Plain, and again at the Robinson quarry near Rankin. All of these outcrops are on railway lines and within a radius of 14 miles from Jackson. The limestones at all of these places have been analyzed and found to be desirable materials for portland cement. Jackson is a good distributing point, with seven railway lines radiating to the North, East, South, and West. Two railway lines, the Illinois Central and the Gulf & Ship Island, connect Jackson with deep-water routes to the Gulf.

"Vicksburg offers more natural advantages for the establishment of a cement plant than any other city in the State. Raw material of limestone and marl are found in the bluffs facing the river. The Mississippi river and the Yazoo & Mississippi Valley Railway afford transportation to the North and South; the Alabama & Vicksburg Railway affords transportation to the East and West. Coal could be obtained by river from Pittsburg, by the Yazoo & Mississippi Valley Railway from Illinois and Western Kentucky, and by the Alabama & Vicksburg Railway from the Alabama coal field."

New Jersey. (By Henry B. Kümmel.)—The portland cement industry of New Jersey during 1908 showed a considerable decrease in production as compared to 1907 and a very marked falling off in prices. The three active plants situated in the New Jersey extension of the Lehigh district reported a total production of 3,208,446 bbl., valued in bulk at the mill at $2,420,907, or 75c. per bbl. These figures show a production of 71 per cent., and a valuation of 56 per cent. as compared to the previous year. All the mills were shut down part of the time, or ran at a greatly reduced output, and 1908 closed with large stocks on hand.

The reported daily capacity of these three mills and of a small mill at Perth Amboy, which has not been a producer for several years, is 17,550 bbl. The actual production for 1908 was, therefore, almost exactly 50 per cent. of the total capacity.

A year ago I took occasion to utter a word of protest against the misleading and highly exaggerated statements made by promoters regarding profits in the portland cement industry.[1] The ratio between the actual production and capacity of present plants and the prices prevailing during 1908 only serve to emphasize the assertions then made. Conditions during 1908 were naturally unfavorable for promoting enterprises of this kind, but with a revival of activity in all lines of construction, many new cement promotions will unquestionably be put before the public. Some of these may possess a fair chance of commercial success. It is safe to say, however, that many of them will serve only as a means of enriching the promoters at the expense of unwary investors. Too much emphasis cannot, therefore, be laid upon a few fundamental facts.

The present capacity of the mills of this State and of the country as a whole is vastly in excess of the present consumption of cement, and is more than able to provide for any probable increase in the immediate future. The margin of profit in cement manufacture is very small at the prices which now prevail and will inevitably prevail during periods of business depression. New companies capitalized far in excess of the actual cost of land, mill and working capital have small chance of success. Finally, it is not easy to find a market for a new brand of cement in competition with well-known brands of established reputations.

New York. (By D. H. Newland.)—The cement business in New York was far from prosperous in 1908. With demand smaller than for a number of years and prices at about the lowest level ever reached, the local plants were put to a severe test as regards economy of manufacture and financial stability. Production was kept down in an effort to maintain some relation with the needs of the market, but the close of the year found most plants holding stocks for which there was no ready sale.

In the portland cement trade conditions were better than in the natural rock industry. The manufacture of natural rock cement, though still carried on rather extensively in the famous Rosendale district, has declined steadily. The total output in 1908 was a little over 600,000 bbl. as compared with an annual production of from 4,000,000 to 5,000,000 bbl. 10 years ago. On the other hand, the production of portland cement showed a falling off of less than 200,000 bbl. from the

[1] Ann. Report of State Geologist of New Jersey for 1907, pp. 15-19.

output of 1907, which was 2,108,450 bbl. At the average selling price of portland cement in 1908, which was about 90c. per bbl., there was but a small margin as regards cost in favor of the natural rock cement.

Notwithstanding the adverse market conditions, there has been an unusual amount of interest shown in the development of the cement resources of New York, and an enlargement of the portland cement industry can be anticipated for the near future. An event of importance in this connection has been the entrance of one of the large Pennsylvania manufacturers—the Atlas Portland Cement Company—into the Hudson River region, where it has secured the property formerly owned by the Hudson Portland Cement Company. The present plant in the city of Hudson, it is understood, will be dismantled and new works with a capacity of 5000 bbl. a day will be erected at Greenport. Two new organizations—the Seaboard Cement Company and the Knickerbocker Portland Cement Company—were formed for the purpose of engaging in cement manufacture in this region. In the western part of the State the plant of the Iroquois Portland Cement Company, at Caledonia, has been taken over by the Marengo Portland Cement Company, which will begin operations in 1909 after making extensive improvements.

North Dakota. (By A. G. Leonard.)—The cement rock, which is a highly calcareous shale, outcrops in the northeastern corner of North Dakota, where the Niobrara beds are exposed at intervals along the western border of the Red River valley for a distance of about 30 miles south of the Canadian line. The outcrops are confined to the deep valleys and ravines of the wooded Pembina Mountain escarpment, in eastern Cavalier county.

The plant of the Northern Cement and Plaster Company is situated at the new town of Concrete, and a branch railroad has recently been built which connects it with the line of the Great Northern, 20 miles to the south. The company manufactures hydraulic cement, cement plaster and stucco. Recent improvements have given the plant a capacity of 500 bbl. per day. It has been in operation seven years.

CHROMIUM AND CHROME ORE.

California is the sole producer of chrome ore in the United States. Deposits of the mineral are known and have been worked in various counties in the State, but the chief activity in 1908 was in Shasta county. The production, however, was insignificant as compared with the total consumption in the United States. The accompanying table shows the production, imports and consumption of chrome ore in the United States for a period of years.

STATISTICS OF CHROME ORE IN THE UNITED STATES.
(In tons of 2240 lb.)

Year.	Production (a)			Imports.			Consumption.	
	Long Tons.	Value.	Value per Ton.	Long Tons.	Value.	Value per Ton.	Long Tons.	Value
1897	*Nil.*	.....	.....	11,566	$186,313	$16.11	11,566	$186,313
1898	*Nil.*	.....	.....	16,304	272,234	16.70	16,304	272,234
1899	*Nil.*	.....	.....	15,793	284,825	18.03	15,793	284,825
1900	140	$1,400	$10.00	17,542	305,001	17.39	17,682	306,401
1901	130	1,950	15.00	20,112	363,108	18.05	20,242	365,058
1902	315	4,725	15.00	39,570	582,597	14.73	39,885	587 322
1903	150	2,250	15.00	22,931	302,025	13.13	23,081	304,275
1904	123	1,845	15.00	24,227	348,527	14.38	24,350	350,372
1905	40	600	15.00	54,434	725,301	13.32	54,874	725,901
1906	317	2,859	9.00	43,441	557,594	12.84	43,758	560,453
1907	335	5,620	20.00	41,999	491,925	11.71	42,333	498,605
1908	280	5,600	20.00	27,876	345,960	12.40	28,156	351,560

(a) As reported by the California State Mining Bureau except for 1907 and 1908, for which years the statistics are compiled from our own reports.

The chief market for chrome ore in the United States is in the manufacturing States of the East. Here the material is used chiefly in the manufacture of bichromates, ferro-chrome alloys and for hardening steel; a considerable quantity is used in the manufacture of chrome brick. The bulk of the supply of raw material is imported from Asiatic Turkey and New Caledonia.

In October a New Jersey charter was granted to the Mutual Chemical Company of America. This company absorbed the American Chrome Works, of Boston, and the Baltimore Chrome Works. These are all close corporations. The new company controls a large amount of the output of chrome salts in the United States.

Prices.—There was practically no change in the price of imported chrome ore or chrome brick during 1908. New Caledonia ore, 50 per

cent., ex ship, New York, brought $17.50@20 per 2240 lb. Bricks were steady at $175 per M., f.o.b. Pittsburg.

Chrome Ore in Foreign Countries.

Africa.—The firm of A. Dickson, operating in the vicinity of Johannesburg, developed its deposits during 1908 to such an extent that it is now prepared to produce as much as 3000 tons per month. A branch line of railway is to be constructed to the chrome mines at Selukwe, Rhodesia.

Asiatic Turkey.—Consul Ernest L. Harris, of Smyrna, reports that chrome ore has dropped so much in price that there is scarcely any profit in mining it, and certainly not in districts far distant from the railways. The most important mines formerly worked were those of Daghardi, in Broussa. The district which has produced and exported the greatest quantity of chrome ore is that termed the basin of Macri, of which the port of that name was the chief point of shipment. The last shipments of chrome ore from Smyrna were 1500 tons in 1906. The largest firm which ever handled chrome at Smyrna has returned all its mines to the Turkish government, as the annual tax upon the ownership of these mines amounted to more than they could be worked for at a profit.

Canada.—The principal chrome deposits are in the Eastern Townships of Quebec. The bulk of the production in 1908 was supplied by the Black Lake Chrome Asbestos Company, which operated extensive deposits in this district. Operations during 1908 were carried on more with a view to developing thoroughly the known resources than to prospects or search for new deposits. The Canadian production during 1908 was 7225 tons, valued at $82,008; this was practically the same as the production of 1907.

THE PRINCIPAL SUPPLIES OF CHROME ORE. (a)
(In metric tons.)

	1898	1899	1900	1901	1902	1903	1904	1905	1906	1907	1908
Bosnia	458	200	100	505	270	147	279	186	320	164	(c)
Canada	1,834	1,824	2,119	1,156	817	3,184	5,512	7,781	7,936	6,528	6,553
Greece	1,367	4,386	5,600	4,580	11,680	8,478	15,430	8,900	11,530	11,730	(c)
India	.....	.....	.....	.....	.....	260	3,654	2,751	4,445	7,325	(c)
New Caledonia (b)	14,300	12,480	10,474	17,451	10,281	21,437	42,197	51,374	57,367	31,552	46,309
Newfoundland	657	717	*Nil.*	*Nil.*	*Nil.*	*Nil.*	*Nil.*	*Nil.*	(c)	(c)	(c)
New South Wales	2,145	5,327	3,338	2,523	454	1,982	403	53	15	30	(c)
Norway	*Nil.*	41	165	85	22	*Nil.*	154	*Nil.*	*Nil.*	*Nil.*	(c)
Russia	15,467	19,146	18,233	22,169	19,655	16,421	26,575	27,051	(c)	(c)	(c)
United States	*Nil.*	*Nil.*	142	132	320	152	125	40	322	339	284

(a) From the official reports of the respective countries. No statistics are available from Turkey. (b) Exports. (c) Statistics not yet available.

COAL AND COKE.

By Floyd W. Parsons.

The production of coal in the United States in 1908 showed less of a falling off than was anticipated. The output of coke, however, was rather less than expected, being only about 61 per cent. of that in 1907.

PRODUCTION OF COAL IN THE UNITED STATES.
(In Tons of 2000 lb.)

	1907.			1908.		
	Short Tons.	Value at Mines.		Short Tons.	Value at Mines.	
		Total.	Per Ton		Total.	Per Ton
Bituminous.						
Alabama	14,424,863	$19,040,819	$1.32	11,523,299	$14,404,124	$1.25
Arkansas	2,550,764	4,336,299	1.70	1,866,565	2,893,176	1.55
California and Idaho	14,500	46,400	3.20	21,760	66,368	3.05
Colorado	10,965,640	15,242,240	1.39	9,703,567	13,099,815	1.35
Georgia	363,463	407,247	1.12	301,640	392,132	1.30
Illinois	(a)51,317,416	54,687,382	1.07	51,507,991	52,538,151	1.02
Indiana	13,985,713	15,114,300	1.08	10,987,419	11,317,041	1.03
Iowa	7,574,322	12,258,012	1.62	(b)7,490,000	11,984,000	1.60
Kansas	7,137,040	10,848,301	1.52	5,960,417	8,940,625	1.50
Kentucky	10,207,060	11,431,907	1.12	9,805,777	10,394,124	1.06
Maryland	5,529,663	6,617,354	1.20	4,377,094	5,116,378	1.20
Michigan	(a)2,035,858	3,660,833	1.80	1,639,927	2,943,883	1.60
Missouri	(a)3,997,936	6,540,709	1.64	(b)3,547,000	5,675,200	1.60
Montana	2,020,910	3,839,729	1.90	1,979,417	3,561,921	1.85
New Mexico	2,628,959	3,832,128	1.46	2,772,586	3,881,620	1.40
North Dakota	298,640	567,416	1.90	317,840	556,220	1.75
Ohio	32,365,949	35,926,203	1.11	28,101,949	30,631,124	1.09
Oklahoma	3,642,658	7,433,914	2.04	3,633,108	7,447,871	2.05
Oregon	(a)70,981	166,304	2.34	(b)60,000	135,000	2.25
Pennsylvania	149,759,089	172,222,952	1.15	118,309,680	130,140,648	1.10
Tennessee	6,760,017	8,179,621	1.21	6,082,851	6,961,393	1.14
Texas	(a)1,648,089	2,778,811	1.69	(b)1,280,490	2,048,784	1.60
Utah	1,967,621	3,364,632	1.71	1,786,204	2,911,513	1.63
Virginia	4,570,341	4,890,265	1.07	4,224,821	3,881,448	0.92
Washington	3,713,824	7,427,648	2.00	2,977,490	6,054,002	2.03
West Virginia	47,906,458	47,427,393	0.99	(b)41,360,500	39,292,475	0.95
Wyoming	6,218,859	10,883,003	1.75	6,100,000	10,370,000	1.70
Alaska and Nevada	15,500	60,450	3.90	10,240	39,936	3.90
Total Bituminous	393,692,113	$468,932,572	$1.19	337,929,632	$387,678,972	$1.15
Anthracite.						
Colorado	45,113	121,805	2.70	69,440	187,488	2.70
New Mexico	(b)17,000	52,700	3.10	(b)20,000	60,000	3.00
Pennsylvania	86,279,719	171,696,641	1.99	80,240,138	158,875,473	1.98
Total Anthracite	86,341,832	$171,871,146	$1.99	80,329,578	$159,122,961	$1.98
Total Coal (Short tons)	480,033,945	640,803 718	1.33	418,259,210	546,801,933	1.31
Total Coal (Metric tons)	435,483,938		1.47	379,361,103		1.44

(a) As reported by the U. S. Geological Survey. (b) Estimated.

The coal production in 1908 was approximately 88 per cent. of the 1907 output, anthracite showing a falling off of about 7 per cent. as

compared with a decrease of approximately 14 per cent. in the bituminous production. One factor that probably acted to prevent a greater falling off in the production of anthracite was the possibility of labor troubles in the spring of 1909, when the present three-year contract with the miners expires. In anticipation of such difficulties, the anthracite operators undoubtedly stored considerable coal.

The close of 1908 showed great stagnation in all branches of the coal industry. The Eastern operators were more severely affected by the slackness in business than the owners of coal mines in the South and West. Likewise, the coke industry felt the depression in business con-

PRODUCTION OF COKE IN THE UNITED STATES.
(In tons of 2000 lb.)

States.	1907.			1908.		
	Short Tons.	Value.		Short Tons.	Value.	
		Total.	Per Ton		Total.	Per Ton
Alabama	$3,096,722	$9,290,166	$3.00	2,336,602	$6,425,656	$2.75
Colorado	1,097,051	3,510,563	3.20	854,662	2,606,719	3.05
Georgia and North Carolina	71,460	278,694	3.90	(a)60,000	210,000	3.50
Illinois	281,400	1,252,230	4.45	310,540	1,335,322	4.30
Kansas	10,500	38,850	3.70	(a)10,000	37,000	3.70
Kentucky	77,055	177,227	2.30	54,515	122,659	2.25
Missouri	7,800	24,258	3.11	(a)5,000	15,500	3.10
Montana	31,400	204,100	6.50	29,482	176,892	6.00
New Mexico	203,437	711,925	3.50	353,240	1,095,044	3.10
Ohio	310,640	913,282	2.94	(a)240,000	672,000	2.80
Oklahoma	57,600	239,616	4.16	24,580	101,270	4.12
Pennsylvania	26,209,422	65,523,555	2.50	12,287,828	23,961,264	1.95
Tennessee	495,200	1,386,560	2.80	250,491	688,850	2.75
Utah	467,284	1,495,309	3.20	321,200	995,720	3.10
Virginia	1,622,734	3,850,620	2.37	1,219,927	2,781,433	2.28
Washington	61,400	276,300	4.50	(a)37,381	205,595	5.50
West Virginia	4,078,222	8,849,742	2.17	(a)3,107,000	6,586,840	2.12
Other States (b)	2,415,911	9,422,053	3.90	1,994,218	7,578,028	3.80
Total	$40,595,238	$107,445,050	$2.65	23,496,666	$55,595,792	$2.36

(a) Estimated. (b) Includes output of by-product coke for Massachusetts, Maryland, Minnesota, New York, Michigan, Wisconsin.

siderably more than did the producers of coal. The production of coke in the Connellsville region, which district affords more than half of the total output of our country, showed a decrease in production of more than 10,000,000 tons, or about 50 per cent. of the 1907 output. This was the natural result of the dullness in the iron and steel industries. In the Western States the demand for coke was about equal to the output during the latter half of the year.

Several important coal companies were organized during the year, but little new development was undertaken. It was expected that during 1908, the Hepburn law would become effective, and that, as a result of the enforcement of this new legislation, many of the larger anthracite and bituminous companies would be obliged to reorganize and operate

independently of the controlling railroad company. The test case against the anthracite coal companies was decided early in the year in favor of the railroads; the case was appealed to the Supreme Court and a final decision is expected in the spring of 1909.

The coal production in the southwestern States—Arkansas, Oklahoma, Kansas and Texas—was influenced less by the financial conditions than by the competition of coal with fuel oil and natural gas. Two other

EXPORTS FROM THE UNITED STATES. (a)
(In tons of 2240 lb.)

	1904	1905	1906	1907	1908
Anthracite	2,228,302	2,229,983	2,216,969	2,698,072	2,752,358
Bituminous	6,345,126	6,959,265	7,704,850	10,448,676	9,100,819
Total Coal	8,573,518	9,189,248	9,921,819	13,146,748	11,853,177
Coke	523,090	599,054	765,190	874,689	622,228
Total	9,096,608	9,788,302	10,687,009	14,021,437	12,475,405

(a) These figures do not include coal bunkered, or sold to steamships engaged in foreign trade.

DESTINATION OF EXPORTS. (a)
(In tons of 2240 lb.)

	1904	1905	1906	1907	1908
Canada	6,577,954	6,964,630	7,533,346	9,843,315	9,252,943
Mexico	880,747	927,170	1,084,319	1,066,502	694,099
Cuba	519,227	564,385	689,333	804,310	690,867
Other West Indies	253,585	200,776	319,339	474,382	374,699
Europe	144,354	101,277	81,734	220,479	234,581
Other countries	197,651	331,010	212,748	737,760	606,988
Total	8,573,518	9,189,248	9,921,819	13,146,748	11,853,177

(a) The European exports in 1908 were chiefly to Italy, that country receiving 189,607 tons. Other countries are chiefly the South American republics. The Canadian shipments were 75.9 per cent. of the total in 1906; 74.9 in 1907, and 78 per cent. in 1908.

factors that tended to cause a decreased demand for coal in this Southwestern territory were the smaller cotton crops, which caused lessened activity at many ginning establishments, and a period of idleness from April 1 to June 15, pending a settlement of the weight scale.

The most serious labor troubles during 1908 occurred in the Alabama field. For a number of years the coal mines of this State were operated on a sliding scale, based on the price of pig iron. The cause of the disagreement was the proposal of the operators to reduce wages 10c. per ton; the union offered as a compromise a decrease of 2½c. only. This was rejected and the strike was ordered. Considerable violence marked this strike, and as a consequence general sympathy was with the operators, who were successful in the outcome.

The financial statements of the coal companies for 1908 strongly reflect the dullness in business. A fair idea may be obtained from the reports of the Reading Coal and Iron Company and the Pittsburg Coal Company, each typical of its branch of the industry. The Reading Coal Company's report for the year ended June 30, 1908, shows a decrease of nearly one-half million tons in coal sold. The production, however, amounted to nearly 12,000,000 tons, an increase over 1907 of 2.2 per cent. The total earnings of the company were $39,215,348, equal to $3.57 per ton of production. The net earnings of the company were $3,592,371, equal to 33c. per ton of production. After deducting total charges, there remained a surplus of $207,515, equal to 2c. per ton of production. The total expenses of the company in 1908 were made up as follows: Cost of coal at mines, 64.6 per cent.; transportation, 29.9 per cent.; selling and general, 5.5 per cent. The total working expenses were 90.8 per cent. of

IMPORTS OF COAL AND COKE INTO THE UNITED STATES. (a)
(In tons of 2240 lb.)

	1904	1905	1906	1907	1908
Canada	1,211,304	1,331,292	1,427,731	1,398,194	1,107,737
Great Britain	135,292	94,600	106,771	42,830	36,969
Australia	235,069	184,426	191,758	552,918	327,441
Japan	45,429	41,956	11,996	123,720	31,792
Other countries	759	569	6,251	8,356	340
Total coal	1,628,675	1,652,843	1,744,507	2,126,018	1,504,299
Coke	.....	181,376	128,461	132,355	129,591
Total	1,628,675	1,834,219	1,872,968	2,258,373	1,633,890

(a) Of the coal imported in 1908, there were 16,483 tons classed as anthracite. Nearly all the imports were for the Pacific coast. The features of 1908 are found in the large falling off of imports from Australia and Japan. The unusual increase in the receipts of Australian and Japanese coal in 1907 was due to the expectation of a fuel famine in the West. The coke received is from British Columbia with the exception of a few thousand tons from Germany.

gross earnings. As compared with the previous year, the gross receipts showed a decrease of $733,149, while the expenses decreased $970,964, leaving a net gain of $237,815. The statement of the Pittsburg Coal Company for the year ending Dec. 31, 1908, showed that the earnings, after deduction of all expenses of operation, interest on bonds of subsidiary companies and current losses, were $3,024,921, which compares with $5,731,983 in 1907. The total deductions in 1908 were $2,559,205, leaving for net earnings, $465,716, which compares with net earnings of $2,958,592 in 1907. The last dividend on the preferred stock of this company was paid in 1905.

The year 1908 was a period of much agitation for the purpose of forcing the enactment of various restrictive measures to govern the coal industry. In several States the so-called reform movement has been carried beyond reasonable limits and if not curbed great injury to the mining industry will result. The legislature of one State is now consid-

ering a bill that will prohibit the sale of fuel in which there is any foreign substance. Should this become a law producers would be held liable if any of their product were dirty. Among other important matters affecting the coal industry is the lack of any just or uniform basis for the assessment of mining property. A number of cases have recently been heard, in both the anthracite and bituminous fields, which have involved the value of coal lands. These cases have generally resulted in litigation, because many coal companies have appealed from the assessment for taxation made on their lands by county Boards. Valuations placed on coal lands are now much higher than in former years, and the companies claim they are entirely too high. In one case the county assessors' figures placed a value on several coal tracts equal to six times the best price paid for coal acreage in the same region.

Other important cases upon which verdicts were rendered, dealt with the responsibility of an employer in case of injuries to an employee. In a number of such cases, damages were awarded to miners who had received injury through falling roof where the mining company had failed to furnish props. Other decisions dealt with the question of discrimination in the allotment of cars by railroad companies. The most important case of this kind was decided by the United States Circuit Court of Appeals at Richmond, Va., and concerned the distribution of coal cars, where part of them were owned by the company whose coal was carried by the railroad company. The action was brought against the Baltimore & Ohio Railroad Company, and certain coal companies said to be allied to it. The charge was that the railroad company in the distribution of coal cars had arbitrarily assigned those owned by coal companies to their owners, and arbitrarily assigned other cars in a way to give the car-owning mining companies a distinct preference and advantage. The decision of the court was that, "the fuel cars of the carrier, its regular equipment of cars, the cars of other roads sent in for fuel, and the private or individual cars of mine operators, should be placed absolutely on the same basis." The court further declared, "any other rule would make it possible for wealthy mine owners to purchase car equipment and utilize car equipment to such an extent as practically to deprive other mine owners of the means of transportation."

The legislative measures most feared by coal operators at the present time are the numerous bills that have been introduced to regulate the use of electricity and the firing of shots underground. There is undoubtedly room for improvement along these lines, but if such bills are not carefully prepared much hardship will be thrust upon coal operators. There is so much coal legislation really necessary that the passage of ill-advised laws is to be deprecated.

Coal Resources.—According to the most recent estimates, the accessible coal supplies of the United States are 1,463,800,000,000 tons. It is stated that this supply will be so depleted as to approach exhaustion before the middle of the next century. The commission making this estimate further states that for every ton produced since coal mining was undertaken in this country, half a ton has been lost or wasted. The first step recommended in extending the life of our fuel supply is to lessen the waste in mining, handling and transporting coal. Recent investigations show that Alaska has a greater coal reserve than generally supposed. The coal of Alaska ranges in quality from lignite to a fuel that compares favorably with the famous Pocahontas coal of West Virginia. Some of the bituminous coals make good coke. Several of the coal seams of this Territory attain great thickness—a total of 60 ft. at many points. The present known coal area in Alaska is placed at 15,000 square miles. The largest coalfield in the United States is that which extends from Casper and Douglas, Wyo., northward to the Canadian boundary. All of the coal in this field lies nearly flat, and in what are commonly known as "blanket seams." All of the coal in this field seems to be low-grade subbituminous, and the land is, therefore, given the minimum value, $20 per acre if within 15 miles of a railroad and $10 per acre beyond that limit.

Review of Coal Mining by States.

Alabama. (By L. W. Friedman.)—The production of coal in Alabama in 1908 was 11,523,299 tons, against 14,424,863 in 1907; 8,754,429 tons in 1908 were run-of-mine. There were 18,783 employees in and about the mines. One hundred and eight men were killed in the mines in 1908. The coke production in 1908 was 2,336,602 tons, as compared with 3,096,722 tons in 1907. During 1908 the United Mine Workers of America carried on a big strike from July to September. The militia was called out for more than a month. The laborers brought in to replace the union men had much trouble in remaining at work, and the production of coal, except at mines where convict labor was employed, remained unsteady and unprofitable.

Arkansas.—The decrease in the production of coal in Arkansas in 1908 was due more to the mild winter and to the increased consumption of oil and natural gas in the adjoining States, than to the depression in business. While for several years previous the Arkansas operators were handicapped by a shortage of railroad cars, that trouble did not exist during 1908. The outlook for 1909 is somewhat brighter, however, as long as oil and gas can be obtained by consumers in the Southwestern States at the present low cost, the demand for coal will remain restricted.

All of the coals of Arkansas are more or less brittle, and when mined yield from 20 to 40 per cent. of slack. Where the coal is mined by blasting or shooting off the solid, the percentage of slack is higher than when produced by hand or pick mining. The coals of Spadra and Russellville seem to be of nearly equal hardness, but a much higher percentage of slack is produced in the former district, owing to the use of powder in mining; the coal in the latter district is won by pick mining. The hard coals, even though they produce a less amount of heat, are better adapted to domestic use than the soft ones. Although some of the Arkansas coals, especially those from the western part of the field, seem to coke in laboratory experiments, none of them has yet been successfully coked on a commercial scale. Coal was first mined in Arkansas by stripping, and in some districts this method of mining is still followed; however, most of the coal now produced is mined from deeper workings, which are reached either by slopes or shafts, the deepest mine in the State being a 480-ft. shaft in the Shinn basin in the eastern part of the field. The actual mining of the coal is generally done by shooting or blasting it off the solid face of the beds. In some of the smaller mines, where the coal is of a high grade and commands a good price, undercutting by hand is the method used. No mining machines of any kind are used in Arkansas, and such machines are not likely to be used in the immediate future. The general reason for machines being so much in disfavor is due to the fact that the beds are usually thin and the percentage of slack from the cuttings of machines would probably be as great as that made by shooting from the solid.

Colorado. (By John D. Jones.)—The coal production of Colorado in 1908 was 9,773,007 tons; this was 1,188,633 tons less than the tonnage of 1907. This relapse in the production is accounted for by the general industrial depression and the mildness of the winters of 1907 and 1908. In consequence numerous coal mines were obliged to suspend operations and some did not resume business until late in 1908. There were but few mines working to their full capacities. The depression in the trade did not affect the wages of the miners, nor the price of coal. No labor troubles occurred to interfere in any noticeable way with the production of coal. The supply of railroad cars was unusually regular, although at times there were shortages, caused more through the inability of the companies to furnish motive power than to a lack of cars.

What makes 1908 a memorable year is the fact that in spite of the poor demand for the commodity the operators directed their efforts toward placing their mines on a safer working basis. In many instances attention was directed also toward developing larger capacities. A number of new mines were opened, not yet producing, which will be equipped with

modern and costly appliances. The following summary gives a fairly accurate digest of the industry during 1908: Number of producing mines, 190; number of new mines opened, 8; tons of lignite coal produced, 1,991,234; tons of semi-bituminous coal produced, 818,216; tons of bituminous coal produced, 6,824,117; tons of anthracite coal produced, 69,440; tons of unclassified coal produced, estimated, 70,000; total tonnage produced, 9,773,007; total tonnage of coke manufactured, 854,662; total number of coke ovens, 2,811; number of employees at the coke ovens, 1,104; number of employees in and about the mines, 14,354.

Idaho. (By Robert N. Bell.)—In the Fremont county coalfields, the Brown Bear mine was developed to a depth of 200 ft. during 1908. The coal is found to be of even better quality and of greater value than in the shallow levels above. This district is producing a good deal of high-grade bituminous coal and is supplying the local farmers; with railway transportation, the district would probably be capable of supplying the demands of the State for high-grade fuel. Unsuspected deposits that promise a big tonnage in the future were developed.

Illinois.—The production of coal in Illinois in 1908 showed but little change. Labor troubles in the central fields early in the year caused the closing down of many mines. The demand was so slack, however, that probably this made no difference. The depression in business caused many consumers to use the cheaper grades of bituminous coals in preference to the more expensive anthracite product. According to the report of the State mine inspector for the year ending June 30, 1908, there were 54 counties in Illinois producing coal. The number of mines operated totaled 922, of which 97 were new developments. There were 108 mines closed down. The total output was 49,272,452 tons. Of this production, 47,809,730 tons were produced by 407 mines, while the remaining 1,462,723 tons were produced by 515 mines engaged in local trade only. The average value per ton for all grades of coal at the shipping mines was $1.02. The aggregate home value of the total product was $50,989,082. There were 105 mines in which machines were used, which compares with 101 machine mines in the preceding year. Of the total production, 15,210,423 tons were produced by mining machines. The total number of employees is estimated at 70,841, of which number, 64,424 work underground. The average price paid per gross ton for hand-mining was 59.3c.; for machine mining, 46.7c. In blasting the coal the miners used 1,328,545 kegs of powder. The number of men killed by accidents totaled 175. For each life lost, 281,557 tons were mined. There were 2.5 deaths per 1000 employed; and 3.6 for each million tons produced.

(By Frank W. DeWolf.)—The most important features of mining in Illinois in 1908 were:—First, the general dullness of the market, and second, the complete shutdown of mining beginning with the April conference of miners and operators. Production before and after the shutdown was stimulated, and thus the public felt no shortage of fuel. The decline in manufacturing and in transportation was not fully taken into consideration by the operators, and the sluggishness of the market, together with overproduction, caused a fall in prices. In the Chicago market, especially, where Illinois and Eastern coals come into competition, prices went by the board.

Judging of the year's production from the best evidence now available, the output may be expected to equal, or possibly exceed, the record-breaking figures of 1907. During the first half year, production fell behind that of the corresponding period in 1907 by two and one-half million tons, but the gains of the second half probably overcame the early losses and carried the total output to approximately 52,000,000 tons.

There has not been much change in the relative production of different districts, though the rapid increase in Williamson and Franklin counties has continued. Expansion in Vermilion county is promised by recent consolidations and by the organization of the new Clinton, Danville & Peoria Railroad. The Dering Coal Company, which has been the largest producer in this field, was placed in the hands of a receiver early in 1909, and this fact, together with the activity of the steel interests in acquiring coal land in this vicinity promises changes of great magnitude. Eighteen counties in the State produced over one million tons each. When the final figures are in, it is likely that Williamson will again lead with a production approximating six million tons. Sangamon, Saint Clair, Macoupin, and Saline counties probably produced from five to three millions each and ranked in the order named. The bulk of the production of the State comes from three seams. The output from Coal No. 6 probably equalled the combined output of Coals 5 and 7, which in turn precede that of No. 2.

Accidents in Illinois mines showed a slight increase over the previous year. Explosions were frequent in southern counties where the coal is gassy and lies at considerable depth. It is timely that the U. S. Geological Survey has established a sub-station of the Technologic Branch at Urbana, where oxygen helmets and other apparatus have already been placed for use in case of explosions or mine fires. The apparatus has already been put to use a number of times, and it seems likely that this station will stimulate the organization of a number of others by the operators of the mines or by the State officials.

Indiana.—The total coal production in Indiana in 1907 was 13,985,713 tons, valued at $15,114,300, which was an increase of 1,893,153 tons over 1906. The total production for 1908 was only 10,987,419 tons. After the month of March, two months were required to adjust satisfactory working conditions, wages, etc., with the mine workers. When mining was resumed it was under a new agreement to last until April 1, 1910. The drought had some influence; in some districts operators hauled water from 10 to 100 miles by railroad to feed their boilers. There were a few strikes and labor troubles, but the principal, controlling feature was the lack of orders, due to the industrial depression. Heretofore, Indiana has furnished a large amount of steam coal to manufacturers throughout the central States. This demand was curtailed nearly one-half. The Ohio river market, because of the low stage of water, was cut out, and this loss was felt by the producers of southern Indiana. The usual demand for domestic coal from the Northwest was tardy and not up to volume, while the lake demand was by no means equal to previous seasons.

About 18,000 miners were employed in the Indiana mines during 1908. The average number of days worked were less than in the previous year. The operators in many instances took occasion to improve their mines during the slack time and employed the idle miners to do the work. In addition to putting the plants in good condition, several new mines were opened and a number of old and abandoned operations were repaired and worked in a limited way. Coal has been found in several localities in the State heretofore unknown to contain workable seams. In each case the work of developing the new field is being pushed successfully.

The special session of the legislature amended the laws so as to make the use of a 3¼-in. bit permissible, instead of the 2½-in. bit prescribed by the law of 1907. The operators are not inclined to furnish the larger bit because of the increased danger, and this has precipitated serious trouble in the Clinton district. There were fewer accidents and explosions in the mines than in 1907; the fatalities from this source did not exceed six or eight lives.

Iowa. (By James H. Lees.)—The coal-mining industry in Iowa was not so active in 1908 as in 1907. This manifested itself in a reduction in the number of days the miners worked, and a consequent diminution in the output. In 1907, according to data collected by the Iowa Geological Survey, the output amounted to 7,574,322 short tons. In 1908 the production was about 7,490,000 short tons. There were a number of causes contributing to this decrease. One of these was the falling off in railroad business, which was especially noticeable in the early part of 1908, notwithstanding the increased passenger traffic due to the 2c. fare. Another

cause was the suspension of mining operations during April, pending the adoption of the working agreement by the operators and the miners. Still another reason was the mild weather prevailing during most of the fall months. Even in December many of the mines worked only four or five days a week.

One of the most important factors was the shutting down of numerous worked-out mines and the delay in opening others, together with the gradually decreasing output of some of the old mines. This factor is partially offset by increasing activity in prospecting and exploitation in certain counties in the central coalfields, notably Boone and Dallas, where there was more prospecting and development than for years past. This is due to the finding of several good seams of coal at greater depths than had been prospected before, the depths varying from 170 to 270 ft. One firm has recently secured control of some 7200 acres of coal land in these two counties. Polk county, in the same general field, has also received an impetus through the opening of about 10 new mines; the larger size and superior equipment of these mines will result in increased output in the near future.

In spite of the slight decrease of production, there was an increase both in the number of mines operated and of the employees. Over 330 mines were in operation, giving employment to 17,312 men. During the fiscal year ending June 30, 1908, there were 38 fatal accidents, or 2.2 per 1000 employees, in addition to 100 serious, non-fatal accidents. This means the loss of one life for each 188,301 tons mined, and serious injury to an employee for each 71,544 tons raised. During the last half of 1908, fewer accidents were reported than for some time previous. The use of mining machines is not extensive in Iowa, and the same may be said of electric haulage, etc. Many mines are not large enough to use them profitably, and the broken nature of the deposits also tends to discourage the installation of expensive equipment except in the larger fields.

Kansas.—The output of coal in Kansas in 1908 showed a falling off of nearly 20 per cent. This, like other central western States, was affected by the general depression in business; however, the lessened demand for coal was due as much to the increased consumption of oil and natural gas throughout the State, as to the stagnation in business. There is small likelihood that there will be any increased demand for coal as long as oil and gas can be obtained at the prevailing low prices.

Maryland.—There is little to be said concerning the coal industry in Maryland during 1908. Labor conditions were quiet and generally satisfactory throughout the year. The car supply was generally good, so that no trouble occurred because of inadequate transportation facilities. One of the most important acts affecting the coal industry in Maryland

in 1908 was the removal Aug. 15, of the 15c. differential on small vein coal. This differential was charged by the Baltimore & Ohio Railroad for hauling coal from the Georges Creek region to the seaboard, in excess of the rate charged for hauling West Virginia and Pennsylvania coal. The differential was ordered removed by the Interstate Commerce Commission. The Baltimore & Ohio Railroad has not yet removed the differential from Georges Creek "big vein" coal. However, the Interstate Commerce Commission strongly implied that this discrimination would have gone the way of the "small vein" if it had been specifically asked.

(By William Bullock Clark.)—The production of coal in Maryland in 1908 was 3,908,120 long tons, valued at $5,116,378. These figures show a decrease of 1,031,726 long tons as compared with the production of 1907, while the value decreased by $1,507,319. This is an unusually heavy reduction in the annual production, representing the lowest valuation since 1901. The average price per ton was the same as in 1907, viz., $1.34 per long ton. The coal from the "Big Vein," or Pittsburg seam of the Georges Creek Valley sold at prices from $1 to $1.80 per long ton, the average being about $1.38. The relative production of coal from the different seams remains about the same, but there appears to be an increasing interest in and development of the "small veins," as the "Big Vein" approaches exhaustion in some of the well-known mines. The conditions during 1908 were not favorable to the development of new properties in the "small vein" coals.

Michigan.—There was much activity in the operation of coal mines in Michigan in 1908. Thirty-five mines were worked. The average number of employees was about 3200. The average daily earning was $3.05, and the aggregate paid in wages was nearly $1,200,000. The average cost of each ton of coal produced was approximately $1.60. There was a surplus of cars during the year, so that transportation facilities were satisfactory. No strikes occurred and labor was plentiful. The decrease in production was almost entirely due to the depressed condition of business in the manufacturing cities along the lake front. Most of Michigan's coal output is used by this local trade.

Missouri.—The output of coal in Missouri during 1908 was about 15 per cent. less than the production of 1907. The year 1903 was the banner year in the history of coal mining in the State, and at no time since has the production been up to the standard then set. Some of the coal produced from the mines at Leavenworth, Kansas, should be properly credited to Missouri, since the workings of these mines extend under the Missouri river into Missouri territory. During 1908, the average daily production per man was 2.16 tons. Mining machines are used only in the thinner seams, where the general scheme of development is on the long-

wall plan. Only about 12 per cent. of the total production is machine-mined. According to the U. S. Geological Survey, the total production of coal up to the close of 1908 was considerably less than 1 per cent. of the coal originally estimated as being available.

Montana.—The production of coal in Montana in 1908 was 1,979,417 tons, as compared with 2,020,910 tons in 1907. In 1907 there was over-production, due principally to a fear on the part of both producers and consumers that a fuel scarcity similar to that which occurred in 1906 might again take place. At the end of 1907 a considerable percentage of the year's coal production remained unconsumed and was carried forward to be disposed of in 1908. During 1908, there were 1589 pick-miners employed, 802 inside day-men, and 750 day-men outside. The mines also employed 176 machine-men and 325 loaders. The total number of men employed was 3642. The only company producing coke was the Montana Coal and Coke Company whose plant is at Electric. This company mined 65,015 tons of coal, which was made into 29,482 tons of coke, constituting the coke production of the entire State. The most important producers in 1908 were the Northwestern Improvement Company, a subsidiary of the Northern Pacific Railroad Company, which produced nearly 650,000 tons. The Cottonwood Coal Company, at Stockett, produced 433,166 tons, the Anaconda Copper Mining Company produced 186,299 tons, and the Nelson-Jenks Coal Company at Sand Coulee produced 113,612 tons. Carefully collected data show that for each pound of powder used 2.81 tons of coal were produced. Ordinary black powder was the kind generally used.

(By H. K. Welch.)—Difficulties between the miners and operators about the wage scale, and the resulting strikes occurring during the last six months, had an effect upon the coal production of the State in 1908. At Roundup, in Yellowstone county, the Chicago, Milwaukee & St. Paul Railroad developed some mines which will eventually become large producers. The Washoe Copper Company's mines at Bear Creek, Carbon county, were brought to a producing state and thoroughly modernized, although of late they have not produced to their full capacity, which is about 700 tons daily. At Red Lodge, in Carbon county, the Northwestern Improvement Company mines held first place in the State in the amount of production. Almost the entire output was used by the Northern Pacific Railway Company. At Chimney Rock some new properties of Evans & Anderson were opened; they give every indication of becoming large producers. The mines of the Great Northern Railroad at Sand Coulee, in Cascade county, were in almost constant operation throughout the year.

Nebraska. (By Erwin H. Barbour.)—The opinion of geologists that coal does not occur in Nebraska in commercial quantities holds good despite the fact that the Honey Creek coal mine is in operation. This at best is local. According to measurements and estimates of Roy V. Pepperberg, of the University of Nebraska, there are probably 250,000 tons of workable coal here, which at $3.50 a ton, the market price of coal at the mine, represents about $875,000, and, since the lessor receives a royalty of 25c. a ton, he hopes to realize about $60,000. The total output for 1906 was 200 tons, worth $839; for 1907, 646 tons, worth $2260; the output in 1908 was 221 tons, worth $773. Owing to fire and subsequent flooding the mine was closed from April to December, 1908. At the outset, the wasteful room-and-pillar method was adopted, but now mining is done by the longwall system. Though the Honey Creek coal is below the standard of the best grades of coal from Iowa or Kansas, it is quite as good as much of the coal shipped from those States. Eleven samples analyzed by Mr. Pepperberg give the following averages: Moisture, 24.4 per cent.; volatile combustible matter, 32.4; fixed carbon, 33.4; ash, 9.7; fixed carbon percent of combustible, 50.7; volatile matter percent of combustible, 49.3; sulphur, 6.22. When freshly mined this coal is compact, but tends to slack readily. It burns well and has good heating qualities, though leaving a large amount of red ash. Its specific gravity is 1.28. The company is incorporated as the Honey Creek Mining Company, with a capital of $15,000. Nebraska is just at the western edge of the great Carboniferous coal beds and at the eastern edge of the Cretaceous coalfields, hence each is pinched out, and coal is not to be expected in the State. Since the outstanding bounty of $4000 for the discovery of a workable bed of coal 26 in. thick was claimed by this company, a committee from the State Legislature visited the Honey Creek mine, and upon their recommendation the House acted favorably to the award.

New Mexico. (By Reinold V. Smith.)—Prof. A. K. Adams, of the department of geology in the New Mexico School of Mines, says regarding coal in the Territory: "The vast coalfields of New Mexico, about ten million acres, have scarcely been scratched, although the annual production is more than 2,500,000 tons." Each of the coalfields has from two to six seams of workable coal, and in many cases the coal mined is more than 5 ft. thick, practically all of it being Cretaceous. Most of the coal (two-thirds of that now mined) comes from the northern part of the Territory, near Raton, and it is in this region that most systematic and scientific mining operations are carried on. This coalfield is the southern extension of the Trinidad field of Colorado and its coal produces excellent coke. The largest producer is the Phelps-Dodge Company, which

operates coal mines and coke ovens at Dawson. The Hagan coalfield, just north of Albuquerque was developed during 1908, and plans are well under way for a branch railway to connect this district with the Santa Fe railroad, and thus to make the coal more marketable. The Cerrillos district produced the only anthracite in the Territory; one seam of coal there is anthracized by a sheet of andesite near it. There were no special developments in this field, mining continuing as usual. At Carthage a short branch connects the Allaire mine with the railroad. The Western coalfields near Gallup were in active operation throughout 1908. Surveys were made for a new railroad in the western part of the Territory; this will furnish transportation to the mines near Silver City, Cooney, and Mogollon, and also the large coalfields south and north of Gallup. If this line is completed it will open up a large area that has great mineral wealth but is as yet little known. The Bernal mine at Carthage had a dust explosion early in January which killed 12 men, that being the only serious accident in the Territory in 1908.

(By Jo. E. Sheridan.)—The total output of New Mexico in 1908, as reported to the U. S. Geological Survey, was approximately 2,772,586 short tons, of which 71,450 tons were used in the operation of the mines and for domestic purposes at the camps, 572,400 tons were consumed in the manufacture of coke, and 2,128,736 tons were shipped from the mines to market. The gross value of the product at the mines was $3,881,620; that of the coal shipped to market was $2,980,230. Slack work at coal mines in a number of States gave New Mexico a full quota of miners for the first time in several years, and 3900 men and 120 boys were employed at the mines. Accidents caused 24 deaths during the year. The percentage of fatal accidents was therefore 5.9 per thousand employees. The enforcement of discipline among the miners by means of more stringent and specific laws and the employment of more careful methods in the operation of the mines would greatly reduce the list of casualties. The coking branch of the industry received so great an impetus in the later months of 1908 that all the ovens in the Territory, half of which had been idle during the first half of the year, owing to lack of demand for the product, were put into operation. The coke production for the year is reported at 318,000 short tons.

Nevada.—The principal supply of fuel used in the State of Nevada has heretofore been shipped from Utah and Wyoming; however, the new coal operations at Coaldale, Nev., are showing up so well that it is probable the State will soon be able to supply part of its needs. At one of the Coaldale operations a shaft is down 208 ft., and at the bottom shows about 42 in. of coal. It is stated that this coal shows 57 per cent. carbon, 0.5 sulphur, and about 25 per cent. volatile matter. The seams are not

uniform in composition, since there are several streaks of good coal varying in thickness from 3 to 5 ft., separated by partings of less desirable coal. On the surface in this same district, there are seven parallel seams which average in thickness from 8 to 80 feet. It is estimated that the Coaldale mines can transport coal to Blair Junction, the nearest railway point, for not more than $2 per ton; the railroad will haul the coal to Tonopah and Goldfield for $1.50 per ton. This would, therefore, enable the consumers to purchase coal at a price of approximately $10 per ton, which would be considerably less than the average price now paid. There are many indications of coal throughout the entire region, and when the many oil concerns that are now drilling in the district have finished their borings, it is almost certain that their development work will disclose good coal seams of considerable area. It is stated by those who have visited the region that the coal here found will make a good grade of coke, and that the product will not slack when exposed to the elements.

North Dakota. (By A. G. Leonard.)—The coalfields of North Dakota are confined to the western part of the State, where the area underlain by lignite-bearing beds is estimated at about 32,000 square miles. The coal seams are distributed through a thickness of from 1000 to 1300 ft. of Fort Union strata. The detailed work carried on in the southwestern portion of the State during the past few years by the North Dakota Geological Survey and U. S. Geological Survey has shown that many of the individual coal beds cover large areas. In the badlands of the region, where outcrops are numerous, it is possible to trace these beds of coal almost continuously for many miles in the bluffs, buttes, and ridges. One bed, with a thickness varying from 5 to 16 ft., has a known extent of 20 miles in one direction and 25 in another, with an area of at least 500 square miles, and probably much greater. Another bed of coal was traced 36 miles north and south and 24 miles east and west, and while its known area as shown from outcrops is about 900 square miles, it undoubtedly has an extent of 1000 to 1500 square miles. This coal bed, which has a thickness ranging from 9 to 15 ft. and over, has been largely burned out or removed by erosion, but still underlies many townships. Other beds of coal are less persistent and the area covered by them is comparatively small. The Consolidated Coal Company opened a new mine during 1908 at Scranton, on the Chicago, Milwaukee & St. Paul railroad. The thickness of the coal bed is 22 ft., with no clay seams. Its depth below the surface varies from 30 to 140 ft. Much of the output of the mine is shipped to South Dakota points. Nearly 92 per cent. of the coal production of the State comes from three counties, viz., Ward, Burleigh,

and Stark, named in the order of their importance. The other 8 per cent. comes chiefly from Morton, McLean and Williams counties.

Ohio.—The decrease in the production of Ohio in 1908 was due wholly to the depression in business. Both cars and labor were plentiful. There were no important strikes, and taken as a whole, 1908 showed fewer labor troubles than for many years previous. Some of the smaller operators, whose output is consumed locally, held their production up to the standard of 1907. The larger producers, however, showed a falling off of from 10 to 20 per cent. in their output. Although during 1908 coal was produced in 29 counties in the State, the greater part of the production came from the following counties: Jefferson, with a production of nearly 4,000,000 tons; Belmont, with a production of more than 5,000,000 tons; Athens, upward of 4,000,000 tons; Guernsey, approximately 3,500,000 tons; Perry, 2,500,000 tons; Hocking, about 1,100,000; Jackson and Tuscarawas, each about 1,000,000 tons. Ohio continues to lead all other coal-producing States in the percentage of the total product mined by machines. Statistics for 1908 show that there were approximately 1400 machines in use. About 92 per cent. of all of the machines used are of the chain-breast type. The famous Pittsburg coal seam extends into Ohio, and the area of workable coal in this seam in Ohio is estimated at 1250 square miles. In Ohio, this seam shows marked variation in structure. Along the river in Jefferson and Belmont counties, where the seam is at its best, the structure is normal; farther west, along the western border of Belmont county and near the western margin of the eastern field, the structure is less constant.

Belmont county not only contains the largest quantity of Pittsburg coal of any county in Ohio, but also contains more coal than any other county in the State. Belmont county has the greatest future prospects, for at present its southern two-fifths do not produce a ton of coal from the Pittsburg bed. All that is needed in that territory is transportation, and the valley of Captina creek furnishes an easy line for railroad construction. Next to Belmont county, Jefferson contains the largest quantity of Pittsburg coal. Coal mining in Jefferson county did not attain any importance until the construction of the Wheeling & Lake Erie Railroad, about 15 years ago. Since that time other transportation lines have tapped the field, making it one of the largest producers of coal in the State. The coal-bearing formation of Ohio contains at least 16 different seams. Of these beds, six are important and have been developed on a large scale, while the other 10 have been developed principally by small mines, whose output has been sold for local consumption. The important productive seams are the Block or No. 1; Wellston, or No. 2; Lower Kittanning, No. 5; Middle Kittanning, No. 6; Upper Freeport, No.

7; and Pittsburg, No. 8. Some of the coals of Ohio are celebrated for certain uses. That of the Hocking Valley region is an open-burning coal, and is most popular as a furnace fuel. The Upper Freeport, or No. 7 seam, is a high-grade steam fuel and would make good coke were it not for its high sulphur content. The first record of production of coal in Ohio was in 1838, when 119,952 short tons were produced. The estimate of the U. S. Geological Survey states that there were 86,028,000,000 short tons of coal in the original supply of Ohio. The total output to the close of 1908 represents an exhaustion of approximately 770,000,000 tons, or less than 1 per cent. of the estimated original supply.

(By J. A. Bownocker.)—Coal mining in Ohio has advanced to the stage where some once important fields are approaching the end. This is well shown in the Wellston field of Jackson county. The coal has usually been considered the best in the State, and of course, development has been rapid. Notwithstanding that the coal output of the State has greatly increased during the past ten years, that of Jackson county has decreased. The output in 1907 was about 1,000,000 tons less than in 1900. Many mines have been abandoned and only an occasional one opened.

Another field with a similar history is the Massillon. There too the coal is of unusual excellence in Ohio. In 1900 the field yielded about 1,000,000 tons, but in 1907 it had dropped to approximately 700,000 tons. A new mine is occasionally opened, but these do not begin to keep pace with the abandoned. A few years more will see the end of both the Wellston and Massillon fields as large producers.

Within the past ten years a persistent effort has been made to extend the Hocking Valley field to the east. The result of the drilling has been to disclose basin shaped deposits of coal, but not a continuous bed. Naturally this discourages development and no great change may be expected for years. It is worthy of note, in this connection, that quite recently a mine has been opened at Canaanville in the eastern part of Athens county, where the coal is about 450 ft. under cover. This is by far the deepest mine in the State.

In the Hocking Valley field proper, few changes have occurred in the past year. An occasional mine is still being opened, but perhaps an equal or greater number is being abandoned. The field as a whole seems now to be at its zenith. In fact, the production in Hocking county is decreasing while Perry county is slowly increasing. Athens county, however, has greatly enlarged its production in the past few years, and still more may be expected whenever there is suitable demand. The same seam is extensively worked in other places, especially in Coshocton and Tuscarawas counties. All in all, this seam is the most important in Ohio.

The Cambridge field has not materially changed in the past few years, and the limits of the territory seem to be pretty definitely established. However, recent tests to the east are claimed to demonstrate some extensions, but these appear to be as pockets rather than as a continuous bed.

The great development in the past few years has been in the Pittsburg field of the eastern part of the State, especially Belmont and Jefferson counties. A few years ago this was rated the third largest field in the State. It has since advanced to the first position, the production of the two counties named having been about 11,000,000 tons in 1907, or more than that of the entire Hocking Valley field.

Moreover, a large part of this field has not yet been touched. This applies to much of the southern half of Belmont county where the coal is under cover and the shipping facilities inadequate. With the opening of this territory, Belmont county, which is now the largest producer in the State, will be in a class by itself. The Pittsburg is the last of the great seams in Ohio to be developed and it promises to be the most lasting in production, and from this time on the most bounteous in supply.

Two important areas of the Pittsburg coal exist in the southern part of the State. One of these is at its best along Federal creek in Athens county, where it has long been mined on a rather small scale and where it has been coked. The calorific value of this field is lower than in the larger one reviewed in the last paragraph. This fact coupled with the poor shipping facilities will probably retard development for some years. The third area of this seam is found in Gallia county and is known as the Swan creek coal. It is shipped from one place by river. In structure and other properties it resembles the Federal-creek field.

Meigs county contains another field whose development has been notably extended within the past few years. This has been regarded as the Pittsburg seam, but recent work by the State Geological Survey has shown it to be the equivalent of the Redstone coal of Pennsylvania and West Virginia. Only one seam, important by reason of the quantity of coal, remains to be developed in the State, and that is the Meigs creek or Sewickley. It underlies the larger part of Belmont, much of Noble, and the eastern half of Morgan, and smaller parts of several other counties. Its proximity to the great Pittsburg seam will continue as a handicap, but eventually it will be the base of extensive mining. At present it is shipped by rail from Flushing, Belmont county and from one place in Noble county.

Oklahoma. (By Charles N. Gould.)—The coal fields of Oklahoma cover about 20,000 square miles in the eastern part of the State and fall naturally into two divisions, namely, one north of the Canadian river, which is usually known as the Cherokee-creek field, and one south of that

river, or the Choctaw field. Before the Choctaw and Chickasaw Indians took their allotments of land, they made a treaty with the Government whereby the land that was underlaid with coal, was segregated or set apart for the benefit of the nations as a whole. The total amount of land segregated is 437,743 acres, to be exact. Joseph A. Taff, of the U. S. Geological Survey, who segregated the coal land, estimated that this area was underlaid with veins of coal, the combined thickness of which approximated 7 ft., and that the average output for the entire region may be placed at 7000 tons per acre. This gives a total of approximately 3,000,000,000 tons of coal on the segregated land. It is probable that this is not half the coal in the State. Conservative estimates place the amount of workable coal in Oklahoma at somewhere between 6,000,000,000 and 10,000,000,000 tons.

The first mining of coal on a commercial scale in the Indian Territory was carried on near McAlester in 1872. Since that time a number of railroads have been built through the coal region and with their aid the coal industry has developed rapidly. In 1907 Oklahoma produced 3,642,658 short tons of coal, valued at $7,433,914. The number of men employed was 8398. The average production per man was 434 tons. Up to October 1, 1908, the coal production in Oklahoma was 2,724,832 tons, the estimated value of which was $6,130,872, or $2.25 per ton, mine run.

There was a general suspension of operation in all the coal mines from April 1 to May 25, 1908, and under present conditions it seems probable that the production of coal in Oklahoma will not increase to any marked extent for several years. This is due largely to the fact that the oil and gas fields of Oklahoma are being developed so rapidly. Beginning near the Kansas line the oil fields have steadily developed southward until at the present time the city of McAlester, which is the center of the coal industry, is preparing to pipe in natural gas. Such towns as Tulsa, Bartlesville, Sapulpa, Claremore, Muskogee, Oklahoma City, Guthrie, and Chandler, where formerly coal was used exclusively for fuel, now secure natural gas at prices varying from 3c. to 25c. per 1000 cu.ft. The oil and gas industry seems to be yet in its infancy, and if the rapid production of the last few years increases, there will probably continue to be a corresponding decrease in the production of coal in Oklahoma.

Pennsylvania.—The production of anthracite coal in Pennsylvania during 1908 held up remarkably well considering the general business depression, and the fact that comparison is made with 1907, which was the banner year. Practically four-fifths of all the anthracite produced is consumed in the New England States, Pennsylvania, New York and New Jersey. Prices for anthracite remained practically unchanged from

those that ruled in 1907. The total production of anthracite coal from Pennsylvania since its initial discovery in 1808 is estimated at 1,800,000,000 tons. The estimated anthracite coal in reserve is as follows: Lackawanna and Wyoming region, 1,857,000,000 tons; in all other districts, 4,427,000,000 tons. This makes the total available anthracite coal in Pennsylvania 6,284,000,000 tons. There is an increasing demand for the smaller sizes of anthracite for steam purposes, and when normal conditions return, it is doubtful if the anthracite operators will be able to meet it. Where thousands of tons of small anthracite coal and boney material was discarded a few years ago, the same grades are now handled with the greatest care. Of the 337,000 men employed in the coal mines of Pennsylvania in 1908, 170,000 were in the anthracite mines. The most astonishing feature of the anthracite industry is the great increase in the proportion of small or steam sizes that are now sold. In the last 20 years the proportion of pea coal and smaller sizes has increased from 22 to nearly 42 per cent. A considerable part of this increase in small-size coal is due to the washery product. It should also be noted that pea coal, which was for years a steam coal, is now used extensively for domestic purposes.

In the bituminous regions of Pennsylvania 1908 showed one great change over conditions in 1907, and that was with regard to labor and transportation facilities. During 1907, the greatest handicap to coal mining in Pennsylvania was the unprecedented shortage of miners, in addition to a shortage of railroad cars. It is probable, however, that if there had been no scarcity of miners and cars many properties would have been compelled to run on reduced time. The inadequate transportation facilities in 1907 enabled the operators to maintain prices at a high level. All of this was reversed in 1908, and as a result the bituminous companies fared poorly. Of the total production in 1908, about 41 per cent. was undercut by machines. Approximately 5000 machines were used in 1908, of which about 70 per cent. were of the pick or puncher type. The year 1908, as usual, had its list of serious disasters. Perhaps the most unfortunate was the explosion which occurred in the Marianna mine near Pittsburg. This mine was owned by the Pittsburg-Buffalo company, which had attempted to make it a model plant, sparing no expense. The accident destroyed 134 men. In connection with this disaster there were many points that should have been cleared up. The coroner's inquest, as usual, was farcical.

The bituminous coalfield of Pennsylvania includes an area of about 14,150 square miles in the western part of the State. The bulk of the coal mined comes from the Allegheny and Monongahela formations. Below the Allegheny formation is the Pottsville containing the Sharon and the

Mercer coal. The Allegheny formation yields about 40 per cent. of the total output of bituminous coal in Pennsylvania. The wonderful increase in the production of bituminous coal in Pennsylvania is shown by the fact that in 1840 the production for the State amounted to less than one-half million tons. In 1860 the production was under three million tons; and in 1870 was less than eight million tons. According to the U. S. Geological Survey there remained unmined approximately 17,000,000,000 tons of anthracite coal. If all of this coal could be recovered, the supply would last nearly 200 years at the rate of production in 1908. The bituminous coal seams of Pennsylvania are estimated to contain about 110,000,000,000 short tons, or more than 700 times the production of 1908. Allowing for a considerable waste in mining, it is probable that the bituminous coal in Pennsylvania will not be mined for at least 400 years.

Tennessee.—The slackness of business throughout Tennessee during 1908 caused a large falling off in the production of coal. Several new properties were opened, but practically none of the mines worked full time. Labor and cars were plentiful. A considerable proportion of the coal mined in Tennessee is made into coke to supply the furnaces at Rockwood and Chattanooga. The falling off in production would probably have been greater had it not been for the strike in Alabama, which temporarily curtailed the coke production in that State. Mining in Tennessee is greatly handicapped by the fact that the markets for the coal are held in common by the operators of Alabama and Kentucky. The workable seams are confined principally to the eastern part of the State; nearly one-half of Tennessee's production is used for railroad fuel. In 1908 there were 145 machines in use, which produced about 14 per cent. of the total. Four-fifths of the machines used were of the pick or puncher type. About 10 per cent. of the total production in 1908 was washed.

Texas. (By J. K. Prather.)—Coal is mined in Webb, Maverick, Houston, Shelby, Wood, Hopkins, Wise, Parker and Erath counties. Lignite is mined in Milam and Robertson counties. Some of the companies are the Rio Grande Coal and Irrigation Company, at Minera, Webb county; the Cannel Coal Company, at Darwin, Webb county; the Carr Wood and Coal Company, near Lytle, Medina county; the Maverick County Coal Company, near Eagle Pass, Maverick county; the Bertetti Coal Company, near Lytle, Medina county; the Rio Bravo Coal Company, near Eagle Pass, Maverick county; the Glenn-Belto Coal Company, Bishop, Bastrop county; the Worley mine, Rockdale, Milam county; the Black Diamond Coal Company, Rockdale, Milam county; the Lignite Eggette and Coal Company, Rockdale, Milam

county; the Central Texas Mining, Manufacturing and Land Company, near Calvert, Robertson county; the Houston County Coal and Manufacturing Company, at Crockett, Houston county; the Timpson Coal Company, near Timpson, Shelby county; the North Texas Coal Company, near Alba, Wood county; the Como Coal Company, near Como, Hopkins county; the Wise County Coal Company, near Bridgeport, Wise county; the Bridgeport Coal Company, near Bridgeport, Wise county; the Texas Coal and Fuel Company, Rock Creek, Parker county; the J. S. Young mine, Keller, Parker county; the Texas & Pacific Coal Company, Thurber, Erath county; the Strawn Coal Mining Company, near Strawn, Palo Pinto county, and the Smith-Lee mine, near Cisco, Eastland county.

The lignite beds in Texas are Eocene in age and extend over a belt 200 miles long. The Cretaceous coals are found in the Eagle Pass coalfields which extend into Mexico, and for eight miles north of Eagle Pass. The Carboniferous coals are found in a small area in El Paso county and in a much larger area in north-central Texas, stretching from a few miles south of the Colorado river in Concho, McCulloch, and San Saba counties in a northeasterly direction to the Red river.

Utah.—That portion of the coal production of Utah which is used for domestic purposes held up remarkably well considering the general dullness in all lines of business. The coke production in Utah, however, showed a falling off of about 60 per cent., which was due to the curtailment of production at the metal mines. Labor was plentiful throughout the year and no serious strike occurred. The closing months of 1908 showed a slight increase in demand, and the outlook for 1909 is more promising. The coal industry in Utah is largely dependent upon conditions existing in the metalliferous trades and the production of coal is largely governed thereby. Practically all of the coal produced in Utah comes from Carbon and Uinta counties. The former produces about 90 per cent. of the total. It is estimated that Utah contains about 13,000 square miles of workable coal beds. These seams are said to contain nearly 200,000,000,000 short tons. Mining began in 1870 and the coal production up to date is less than 20,000,000 tons.

Washington. (By D. C. Botting.)—The decrease in the production of coal in 1908 was largely the result of adverse conditions beginning in 1907, which were not improved until April, 1908. Throughout 1908 the supply of railroad cars was unusually good. The labor supply also exceeded the demand. In summing up the results perhaps none is more gratifying than the great desire manifested on the part of mine owners and officials to comply with the demand for safety measures to protect the lives of employees. The relation between operators and employees

in general was harmonious, and no labor trouble occurred which in any way interferred with the production of coal. It is also gratifying to report that the number of fatalities for the year was 32½ per cent. less than in 1907.

The total production of coal in 1908 was 2,977,490 tons, valued at $6,054,001.61, an average of $2.03 per ton. The only coke-producing county, Pierce, manufactured 37,381 tons; value at oven, $5.50 per ton; total value, $205,595.50. There was 5341 men employed, averaging 201.5 days per year, with an average daily production per employee of 2.77 tons.

A new mine was developed by the Green River Coal Company at Bayne. This mine is expected to produce from 150 to 200 tons per day in the near future. The Yakima-Roslyn Coal Company purchased the Kittitas & Ellensburg company's mine at Roslyn and is operating the same successfully. The year's production averages about 58 tons per day. This average will be largely increased during 1909.

West Virginia.—The number and seriousness of the accidents in West Virginia during 1908 showed a great falling off from the appalling figures of 1907. The year's record did not record any such disaster as the one that occurred at the Monongah mines in 1907. Every effort has been made by the State Department of Mines, as well as by the operators, to reduce the number of fatalities to a minimum.

With the possible exception of Pennsylvania, it is probable that West Virginia felt the depression in the coal business to a greater extent than any other State. There were no labor troubles of consequence, and the miners, as a rule, still remain in the ranks of unorganized labor. Chief Mine Inspector Paul has severed his connection with the State to accept a position with the U. S. Geological Survey experimental station corps. John Laing, a mine superintendent of experience, has been appointed chief inspector to succeed Mr. Paul. At the close of 1908, the outlook for the coal industry in West Virginia is a little brighter, although it is not expected that the output in 1909 will equal the record production of 1907.

(By I. C. White.)—The coal industry of West Virginia, like that in all other States of the Union, has languished during 1908. Production, instead of increasing at the rate of 4,000,000 tons annually, as had practically been the case for the six previous years, decreased about 10 to 15 per cent. so far as can be estimated in advance of exact figures. The accompanying table gives the production in West Virginia during all years since 1873, the date from which statistics have been gathered with more or less exactness. The estimate of 41,360,500 short tons for 1908 may be a million tons in error, one way or the other, when the final

returns can be compiled. This decrease in production was not equally distributed among the several districts, the very pure coals along the Chesapeake & Ohio and Norfolk & Western railways appearing to show a slightly smaller decline in shipments than those from other fields.

It is frequently stated in publications of every description that West Virginia has 17,000 square miles of coal, and in a publication of the

COAL PRODUCTION OF WEST VIRGINIA SINCE 1873.
(In tons of 2000 lb.)

Year	Quantity.	Year.	Quantity.	Year.	Quantity.	Year.	Quantity.	Year.	Quantity.	Year.	Quantity.
1873.	672,000	1879..	1,400,000	1885..	3,369,062	1891..	9,220,665	1897..	14,248,159	1903..	29,337,241
1874.	1,120,000	1880..	1,568,000	1886..	4,005,796	1892..	9,738,755	1898..	16,700,999	1904..	32,406,752
1875.	1,120,000	1881..	1,680,000	1887..	4,881,620	1893..	10,708,578	1899..	19,252,995	1905..	37,791,580
1876.	896,000	1882..	2,240,000	1888..	5,498,800	1894..	11,627,757	1900..	22,647,207	1906..	43,290,350
1877.	1,120,000	1883..	2,335,833	1889..	6,231,880	1895..	11,387,961	1901..	24,068,402	1907..	48,091,583
1878.	1,120,000	1884..	3,360,000	1890..	7,394,654	1896..	12,876,296	1902..	24,570,826	1908..	(*e*)41,360,500

(*e*) Estimated.

U. S. Geological Survey, as late as May, 1908, these same figures are given, together with an estimate of the total amount of coal in the State, which is placed at 231,000,000,000 tons. Regarding these figures as largely guess work, and entirely too great, the State geologist in his new Volume, II(A) on West Virginia coals published under date of Sept. 15, 1908, went carefully over the matter, county by county, and reached the accompanying detailed estimates for the number of square miles of coal lands in each county of the State, and the total available commercial coal. The total shown in this table is 9500 square miles (6,080,000

COAL RESOURCES OF WEST VIRGINIA.

Counties.	Coal Area. Sq. Miles.	Counties.	Coal Area. Sq. Miles.	Counties.	Coal Area. Sq. Miles.
Barbour	300	Kanawha	640	Preston	350
Berkeley, (a)	*Nil.*	Lewis	200	Putnam	20
Boone	500	Lincoln	400	Raleigh	480
Braxton	310	Logan	400	Randolph	100
Brooke	75	Marion	250	Ritchie	10
Cabell	20	Marshall	240	Roane	20
Calhoun	10	Mason	30	Summers	20
Clay	195	Mercer	80	Taylor	100
Doddridge	30	Mineral	60	Tucker	50
Fayette	700	Mingo	350	Tyler	50
Gilmer	100	Monongalia	300	Upshur	150
Grant	100	Monroe	*Nil.*	Wayne	150
Greenbrier	100	Morgan	(a)	Webster	150
Hampshire	*Nil.*	McDowell	700	Wetzel	200
Hancock	75	Nicholas	500	Wirt	10
Hardy	*Nil.*	Ohio	120	Wood	10
Harrison	300	Pendleton	*Nil.*	Wyoming	500
Jackson	20	Pleasants	10		
Jefferson	*Nil.*	Pocahontas	25		

(a) Has a little impure anthracite.

acres). If we estimate the total available good coal in this area that can be recovered under commercial conditions at 10,000 tons to the acre

we will certainly have included all of it and probably more. This estimate would yield 60,800,000,000 tons as the available quantity of good commercial coal that can be procured from surface and shaft mines from every portion of the State. The average of 10,000 tons to the acre will, of course, be much exceeded in some special areas, but it will not be equaled in other large ones, so that it is probably not an underestimate. Then, too, some counties, like Calhoun, Jackson, Ritchie, Wood, Wirt, Pleasants, and others, have been given credit for small coal areas, while the fact of the matter is they may not have any worth mentioning, if the drill of the petroleum seeker counts for anything. Hence it is safe to conclude that West Virginia will produce somewhere between 50 and 60 billion tons (of 2000 lb.) of commercial coal, and probably one-third as much more, or 20 billion tons, of impure and boney coal of practically the same grade in calorific value as that shown by the analysis of the Washington coal on page 689 of Volume II(A). This grade of fuel is, of course, not now marketable, since its heat units can best be rendered available through the agency of producer gas and the gas engine, which have not yet come into general use, although each ton of this impure coal when so utilized will produce more power, according to the determinations of the Technologic Branch of the United States Geological Survey, than a ton of the purest New River or Pocahontas coal, when the steam engine is the agency of conversion.

At the meeting of the National Conservation Commission in Washington December 1, 1908, as the result of discussion and criticism by me, Mr. Campbell reduced his estimate of West Virginia's total coal deposits by 81,000,000,000 tons, bringing it down to 150,000,000,000 instead of 231,000,000,000, as given at the Conference of Governors in May, 1908, thus making it in practical accord with the foregoing estimates of the State geologist of West Virginia, which allowing for mining waste would read as follows for the total quantity of coal in the State in round billions: Good commercial coal, 60,000,000,000; waste in mining same, 60,000,000,000; inferior coal, 20,000,000,000; waste in mining same, 20,000,000,000; total, 160,000,000,000.

The waste in mining has been estimated upon what has proved true in the past, viz: A ton of coal lost for each ton recovered, and on this basis the figures of the State geologist for the total quantity of coal in the State exceed the revised estimates of Mr. Campbell by only 10,000,000,000 tons, so that they are in practical agreement, and if the mining waste can be reduced to one-third of the total, instead of one-half as used in these estimates, then the available coal possible to bring to the surface would be as follows: Good commercial coal, 80,000,000,000; inferior coal, 27,000,000,000; total, 107,000,000,000.

The Virginian Railway just completed through the Kanawha, New River, and Pocahontas coal fields, and connecting them directly with tidewater at Norfolk, is certain to exert a great stimulating influence on the future developments of West Virginia's coal industry. Owing, however, to the fact that this railway skirts along the edge of the developed coal fields, and that spurs, a few miles in length, must be built in most cases to reach the richest coal areas, it will be probably a year or more in the future before shipments over the new route can be greatly increased. The contract for one of these most important spurs, involving an expenditure of $1,500,000, has already been let, and the work is in vigorous progress along Winding Gulf Branch of the upper Guyandotte river, where a vast area of the Beckley coal is available above water level and at convenient mining hight above the grade of the railway.

The E. F. Saxman Coal and Coke Company has begun the development of the Sewell coal, or highest member of the New River series, on the waters of Cherry river in Nicholas county near the town of Fenwick. This operation is on the Gauley Branch of the Baltimore & Ohio Railway, and the intention is to manufacture a high-grade coke for furnace and other metallurgical uses. This is the most northeastern point at which the New River coals have been opened up for commercial purposes.

Wyoming.—The greatest trouble experienced by the Wyoming coal operators in 1907 was due to the scarcity of labor and the lack of railway cars. In 1908 these difficulties were largely eliminated; however, the industry was confronted with a lessened demand, which resulted principally from the decrease in traffic in the larger trans-continental railways. The output in 1908 showed but little change from 1907. The total output sold for about $10,370,000 at the mines. This was an average value of approximately $1.70 per ton. In the fall of 1907 the coal operators of Wyoming entered into an agreement with the United Mine Workers, whereby the miners were to receive an average of $3.40 for eight hours and 60c. a ton for mining on a run-of-mine basis. This agreement terminated in September, 1908 and an attempt was made to reduce the wage scale, but as the miners refused to accept any reduction, a number of mines were closed down. After an idleness of about one month, most of the mines resumed operations at the old rate. A number of new mines were opened in 1908, and if consumption does not show a considerable increase, the industry will not flourish during 1909. The greatest uncertainty that hangs over the industry at present is the possibility of adverse tariff legislation, which would affect Montana and Wyoming operators to a greater extent than the coal producers in any other State. Even with the present tariff in effect, the coal operators in the Crow's Nest district of Canada are making a considerable bid for the

Northwestern markets. Approximately 20 per cent. of Wyoming's coal production in 1908 was machine-mined. The two types of machines generally used are the pick or puncher type, and the chain-breast, with preference slightly favoring the former type. Nearly one-half of Wyoming is underlaid by coal-bearing formations.

COAL PRODUCTION IN THE CHIEF COUNTRIES OF THE WORLD.
(In metric tons.)

Countries.	1903	1904	1905	1906	1907	1908
Asia:						
China					10,450,000	11,970,000
India	7,557,400	7,682,319	7,921,000	9,783,250	11,147,339	12,865,408
Japan	10,088,845	11,600,000	11,895,000	12,500,000	13,716,488	13,942,000
Australasia:						
New South Wales	6,456,524	6,116,126	6,035,250	7,748,384	7,850,000	(e)7,992,300
New Zealand	1,442,916	1,562,443	1,415,000	1,600,000	1,831,009	1,904,276
Other Australia	771,536	769,723	805,000	870,000	900,000	(e)870,000
Europe:						
Austria Hungary (c)	40,160,823	40,334,681	40,725,000	37,612,000	40,112,530	40,760,870
Belgium	23,913,240	23,380,025	21,844,200	23,610,740	23,705,190	23,678,150
France	34,906,418	34,502,289	36,048,264	34,313,645	36,753,027	37,622,556
Germany (c)	162,457,253	169,448,272	173,663,774	193,533,259	205,542,688	215,071,345
Italy	346,887	359,456	307,500	300,000	453,137	421,906
Russia	(f)17,500,000	19,318,000	17,120,000	16,990,000	21,207,500	22,943,794
Spain (c)	2,800,843	3,123,540	3,199,911	3,284,576	3,250,000	(e)3,871,480
Sweden	320,390	320,984	331,500	265,000	305,000	(e)300,000
United Kingdom	233,419,821	236,147,125	239,888,928	251,050,809	267,828,276	261,506,379
North America:						
Canada—						
Western	1,791,798	2,619,816	3,183,909	3,717,816	4,780,301	4,304,600
Eastern	4,700,645	4,194,939	4,775,802	6,196,360	5,730,660	6,599,866
United States	317,272,110	318,275,920	351,120,625	375,397,204	435,483,938	379,361,103
South Africa (a)	2,957,736	3,015,000	3,218,500	(e) 3,900,000	3,945,043	4,621,988
Other countries (e)	4,000,000	4,250,000	4,550,000	5,500,000	3,475,780	(e)4,106,000
Totals	882,865,185	867,020,658	928,049,163	988,173,043	1,098,468,506	1,054,714,021

(a) Transvaal, Natal and Cape of Good Hope. (c) Includes lignite. (e) Estimated. (f) Estimated by Minister of Finance.

Coal in Foreign Countries.

Belgium.—The report of the inspector-general of mines shows that in 1908 the production of coal amounted to 23,678,159 metric tons, a decrease of 27,040 tons, as compared with 1907. The production of 1907 showed a decrease when compared with 1906; this falling off was attributed to labor troubles. The average daily wages of the working force of the mines was 4.95 francs (95½c.), which was an increase of 44 centimes (8.49c.) over the wages of the previous year. While the coal industry was prosperous in 1908, the operators of the mines have for some time past been forced to make concessions; prices have dropped, and the conclusion seems to be reached that the industry has been face to face with a world-wide financial crisis. It is said that unusually large shipments of foreign coal have recently been imported into Belgium, and many interested observers of present conditions fear that an invasion of foreign coal will have a depressing effect upon the Belgium industry, which may result in a reduction in wages. The latest available statistics

show that the proportion of coal miners killed, out of every ten thousand workmen employed in the mines, was 15.10 in 1907, against 10.87 in 1906, and 9.97 in 1905.

CONSUMPTION OF COAL IN BELGIUM.

	1908. Tons.	1907. Tons.
Production	23,678,000	23,705,000
Stock at beginning of year	459,000	356,000
Imports (a)	5,918,000	5,894,000
Totals	30,055,000	29,955,000
Stocks at end of year	917,000	442,000
Exports (a)	6,389,000	6,240,000
Totals	7,306,000	6,682,000
Consumption	22,749,000	23,273,000

(a) The imports and exports include coke and briquettes, the amounts added for those commodities being calculated on the amount of coal involved in their manufacture.

The coke syndicate of Liége recently induced various independent coke oven owners to join the combination, and it is now stated that the syndicate has concluded negotiations with the pits for the establishment of a monopoly of the sale of small coking coal. Pig-iron producers who have coke ovens expect that this action will lead to increased purchases of German small coal.

Canada.—Each of the coal-mining provinces except British Columbia, made an increased output in 1908. Of the total, Nova Scotia contributed over 59 per cent., Saskatchewan and Alberta over 19, and British Columbia 21 per cent. The total production of oven coke in 1908 was approximately 865,257 short tons, valued at $3,668,974, being an increase of about 3 per cent. over the quantity produced in 1907. The coke was made in Nova Scotia, Alberta and British Columbia and entirely from Canadian coal. At the end of the year there were in Nova Scotia about 659 ovens in operation and 173 idle; in Alberta and British Columbia 916 in operation and 528 idle, not including the ovens at Hosmer and Comox in British Columbia, from which no returns were received.

(By E. Jacobs.)—In British Columbia, the output of about 1,700,000 long tons was less by 100,067 tons than that of 1907. This production was chiefly from Vancouver island collieries, of which the Wellington Colliery Company produced about 740,000 tons, and the Western Fuel Company 393,000 tons. On the mainland of the province, the Crow's Nest Coal Company produced 522,000 tons and the Nicola Valley Coal and Coke Company, which recently opened mines in the Nicola district, about 30,000 tons. Coke production was 13,000 tons at the Wellington Col-

liery Company's ovens at Union, Vancouver island, and 235,000 tons at the Crow's Nest Pass Company's ovens at Fernie and Michel. It is noteworthy that the larger companies in the Crow's Nest Pass region are substituting the most economical type of Corliss condensing engines for the simple slide-valve engine.

On Dec. 18, the Hosmer Mines, Ltd., commenced the shipment of coal from its newly opened colliery at Hosmer, on the Crow's Nest branch of the Canadian Pacific Railway. This company's property consists of six sections of coal lands and two sections of surface, on which the town of Hosmer is built. There are 13 coal seams on the property, varying in thickness from 4 to 30 ft. Bee-hive coke ovens, 240 in number, have been erected, sufficient to give an initial output of about 300 tons per day, while the installation of Belgian ovens, with by-product recovery, is contemplated.

COAL PRODUCTION IN CANADA.

Province.	1907		1908	
	Tons.	Value.	Tons.	Value.
Nova Scotia	6,354,133	$12,764,999	6,539,866	$13,138,124
New Brunswick	34,584	77,814	60,000	135,000
Saskatchewan	151,232	252,437	130,000	214,500
Alberta	1,591,579	3,836,286	1,845,000	4,899,611
British Columbia	2,364,898	7,390,306	2,329,600	7,280,000
Yukon	15,000	60,000	.....	.....
Total	10,511,426	$26,381,842	10,904,466	$25,567,235

Another coal-mining industry established in 1908 is that of the Corbin Coal Company, which has opened a mine in the Crow's Nest district and connected it by a 17-mile railway with the Canadian Pacific Company's Crow's Nest branch. Several other companies are preparing to mine coal.

The South Wellington Coal Mines, Ltd., has opened a mine on Vancouver island and is now installing a tipple of 1000 tons daily capacity, constructing a tramway seven miles to tide water, and building bunkers and a loading plant at the shipping point. This company expects to have its preparatory work completed by May, 1909.

China. (By T. T. Read.)—The annual output of coal in China amounts to about 12,000,000 tons. The production is chiefly anthracite and bituminous, but a small amount of lignite is also produced, the chief center for the last being the Fu-Shun mines in Manchuria, now worked by the Japanese; 356,000 tons is given as the output of the principal mines for 1908; native workings in the same vicinity produce a further indeterminate amount. Near Jehol, in the northern part of Chili

province, there is another lignite field, which is estimated to produce about 250,000 tons annually.

The production of anthracite and bituminous is about equal in amount, but the latter is much more widely distributed, occurring in nearly all the provinces; while anthracite is confined to the northern part, chiefly the provinces of Chili, Shansi, and Honan. All the production in Shansi, with one small exception, is from small native mines. Most of the production in Chili is also from native mines, but in Honan and Shantung, companies under foreign direction contribute a large part of the output.

PRODUCTION OF COAL IN CHINA IN 1908.

Province.	Anthracite.	Bituminous.	Lignite.
Fengtien (Manchuria)	.....	.....	500,000
Chili	750,000	1,750,000	250,000
Shansi	4,000,000	.....	.....
Shensi	.....	500,000	.....
Kansu	.....	500,000	.....
Shantung	300,000	300,000	.....
Honan	950,000	.....	.....
Ssu-chuan	.....	500,000	.....
Yun-nan	.....	300,000	.....
Chekiang	.....	10,000	.....
Kiang-si	.....	500,000	.....
Hunan	.....	200,000	.....
Kuangtung	.....	50,000	.....
Kuangsi	.....	100,000	.....
Other Provinces	.....	100,000	.....

The greatest amount of bituminous coal is produced in Chili province, the Chinese Engineering and Mining Company alone producing 1,120,000 tons during 1908. There are two other companies under foreign direction in this province, which, together with many native mines, add about 630,000 tons more to the total. In Shantung, the Shantung Bergbau Gesellschaft produces about 200,000 tons and native mines about half as much more. In Kiangsi the Han-Yeh Ping Iron and Coal Company operates a large mine under foreign superintendence, producing about 400,000 tons per year. Native mines furnish a considerable amount in addition. In the remaining provinces the production is, so far as I know, entirely confined to native workings. For more remote ones, such as Kansu, Ssu-chuan and Yun-nan, it is impossible to obtain any definite information. Above is given a table of the estimated production by provinces for 1908. I am indebted to K. Y. Kuang, Provincial Director of Mines, for the figures for Chili province, and to my colleague, Dr. N. F. Drake, for the rest.

The chief producer of coke is the Han-Yeh-Ping Iron and Coal Company, in Kiangsi. This produced 160,000 tons of coke in 1908, in modern retort ovens. The Chinese Engineering and Mining Company also produce a large amount of coke, using bee-hive ovens and moiler. In all

the bituminous fields throughout the Empire, large amounts of coke are produced in primitive native ovens, as the bituminous coals are commonly so friable that a good deal of loss is incurred in transporting them unless they are first coked.

France.—The coal mining industry in France in 1908 struggled against difficult conditions. Trade was indifferent, but Germany, Belgium and Great Britain suffered from the same complaint. Production in France was about 37,000,000 tons, a good total, but the stock on hand at the end of the year was rather excessive. At that time, however, the outlook was favorable to improved business in 1909. With coke 48c. per ton cheaper and a reduction of 58c. in the price of coal, there was a noticeable resumption of activity. Efforts are being made to promote the use of French anthracite in Paris, instead of the British. The latter commands $14.47 per ton, while the French product, which is claimed to be of practically equal quality, is obtainable at $12.50. Coal mining in France in 1908 was singularly free from labor troubles. This condition is believed to be too good to last, and strikes in 1909 are anticipated.

Germany.—The coal production in Germany during 1908 amounted to 215,071,345 metric tons, an increase of nearly 10,000,000 tons over the production of 1907. German operators are much concerned about the competition of British coal in their home market. During 1908 the business depression in Germany was acute, and the coal trust, so as not to accumulate too large a stock, made strenuous efforts to increase its sales of coal and coke in the foreign markets and to enlarge sales at home by stronger competition with English coal. However, in spite of the lessened consumption of coal in Germany, the imports of British coal scarcely diminished, the latter crowding out the product of the trust wherever it can be carried along waterways. Even the Government has at some points used English coal for its railroads. Of the average annual receipts of coal at Hamburg, more than two-thirds come from England, and less than 2,500,000 tons are native product. Berlin consumes 1,100,000 tons of English coal annually, against 480,000 tons furnished by Germany, and the furnaces and iron works along the German coast, get almost all of their coal from England. The German coal-mining interests desire that the government grant them preferential freight rates for railroads and canals so that they can successfully compete with the British coal. A sad event in coal mining in Germany in 1908 was the great catastrophe at the Radbod mine in Westphalia, which occurred on Nov. 12, 1908, whereby 300 miners lost their lives.

The Government collieries in the Saar district were active during 1908. The coal seams in this field usually consist of several bands of coal, separated by barren partings, which rarely contain a thickness of pure

coal exceeding 3¼ ft., and according to the fluctuating thickness of the partings are worked either as one seam or as distinct seams. The duration of the coal supply in the area reserved by the Prussian State is reckoned to be 500 years. There are 24 collieries at work in this field. The recently extended use of packing has greatly contributed to the diminution of accidents by falls of stone or coal in the Saar basin. Practically all of the coal that is brought to bank, has to undergo mechanical sorting and washing before it is marketable. About 20 per cent. of the output is converted into coke, the amount obtained from the undried coal being 53 or 57 per cent., as compared with 65 or 70 in the case of the Westphalian bituminous coal. The coke as sold contains about 12 per cent. of ash. The total number of persons employed above and below ground, averages 47,000; the daily work for the pitmen does not exceed 8½ hours; for persons employed at points where the average temperature exceeds 72 deg. F., the working day is limited to six hours.

India. (By Glen George.)—Coal occurs in India over a widespread area, ranging from the hill coalfields of Assam in the north to that of Hyderabad in the south. The coal measure formations are exposed over a large part of this area. There are some 37 fields having an estimated approximate area of 30,000 square miles. But of these, only eight are worked to any extent, while of the total output of India 85 per cent. is raised from the three coalfields in Bengal, those of Raneegunge, Jherria and Giridih.

The situation of the Bengal coalfields near Calcutta and the cost and the better quality of coal, have been factors which have caused their rapid development in the last decade. The total output of the Bengal fields in 1898, compares with the production in 1907 as follows: In 1898 Raneegunge produced 2,216,000 tons; Giridih, 653,000; Jherria, 750,000; total, 3,619,000. In 1907, Raneegunge produced 3,980,035 tons; Giridih, 750,374; Jherria, 5,179,185; total, 9,909,594.

The coal seams are thick on the whole, the majority of the beds worked being 12 to 18 ft. In a few cases beds 30 ft. thick are being developed, while in limited areas, seams 60 to 90 ft. thick exist, owing to the amalgamation of several beds. The seams are almost always inclined, but not at steep angles, the dip being, as a rule, 1:10 to 1:5. Faults are not so frequent as to be vexatious. Igneous dikes, on the other hand, give a great deal of trouble and a considerable area of one of the best seams has been spoiled by them.

As regards quality, the best Bengal coals are good. The defect of Indian coal is the rather high percentage of ash. On the other hand, the percentage of sulphur is low. The seams with the best reputations are the Sanctoria, Desherghur and Seebpore, of the Raneegunge field,

Giridih coal and Nos. 14 and 17, of Jherria. Then follow a considerable number of seams slightly inferior, but still of fairly good quality. Following is a typical analysis: Desherghur Seam.—Proximate analysis: Fixed carbon, 54.33; volatile carbon, 33.77; ash, 9.47; moisture, 2.43. Ultimate analysis: Carbon, 69.19; hydrogen, 5.10; oxygen, 11.83; nitrogen, 1.59; sulphur, 0.37; moisture, 2.43; ash, 9.47. Calorific value, 6580 B.t.u. Evaporating capacity, 8.18 lb. water per lb. of coal. Specific gravity, 1.321.

A considerable number of seams coke well, but the percentage of ash in the coke is high. The method of coke manufacture is crude and wasteful—burning in open kilns—and yields only 40 tons of coke per 100 tons of coal. A range of by-product ovens is being erected by the East Indian railway at its Giridih collieries. Although a great deal could be done to improve the sale and price of second-class coals by picking, so far no efficient system of cleaning has been adopted.

The usual method of working is simple. The poor quality of available labor, the high cost of timber, the thickness of the seams, render longwall out of the question, or even any method of taking out a very large proportion of the coal is difficult. The almost unvarying method consists in driving two sets of galleries at right angles, splitting up the coal into square pillars. The usual width of the galleries is 12 to 16 ft., while the size of the pillars varies from 30 to 70 ft. square, according to the depth. Roofs are good and timbering is the exception. On reaching a boundary, or in any favorably situated area, the pillars are split and worked as much as possible before the area collapses. The average percentage of extraction is about 60 to 70 per cent. of the seam.

The occurrence of firedamp is rare, and safety lamps are the exception. The large size of the galleries and the number open, make the drag of the air currents small and in almost all cases natural ventilation, or turning the exhaust steam from pumps into a shaft, is found sufficient. In eight cases only have fans been erected, but deep workings must soon lead to a large increase in their number. Water is usually met with in fair, but not great, quantities. In brief, it may be stated that coal mining in Bengal, so far, has presented no great difficulties from the engineering point of view. The death rate of persons employed is low—0.76 per thousand miners above and below ground. From the point of output, however, the death rate is high—7.47 per million tons. Methods in the past have not been economical, large areas of pillars having been abandoned, owing to unskilful working. A great improvement has taken place in recent years.

While Indian coal mining presents no great engineering difficulties as yet, the problem of securing intelligent labor is most perplexing. Labor

is obtained locally and consists chiefly of the two low castes—the Bowries and Santhals—who are an aboriginal race. The Indian native is an agriculturist and has an almost passionate attachment to the land, and regards coal mining only as a disagreeable necessity, in spite of the fact that he can earn four to six times as much at the latter as at the former. This has led to the absence of the formation of any craft skill and the Indian miner is as bad a workman today as he was 40 years ago. He cuts only 96 tons per year, as compared with 572 of the American, and the 279 of the English miner. Working costs are fairly low, ranging from 48 to 56c. per ton exclusive of royalty. Where royalty has to be paid, it is usually 4 annas (8c.) per ton. Prices until of late have been low—as low as 64c. per ton at the pit's mouth. During the past two years, prices have risen considerably and $1.60 at the pit's mouth is probably the average at present. The coal trade suffers greatly from the lack of rolling stock of the railway companies and the complaints in Calcutta are loud and insistent at present. To give an idea of the magnitude of the industry, it may be stated that the capital of the limited liability coal companies (excluding private concerns) is $17,000,000, while the present market value of their shares is nearly $90,000,000.

Japan.—Practically all the coal mined in Japan comes from Kiushiu, the most southern of the main group of islands, and the Hokkaido, the most northern. The principal fields in the latter district are being worked by the Hokkaido Colliery and Steamship Company, whose main office is in Tokyo. The seams in the Yubari field of the Hokkaido district are the most extensive, there being four, dipping at an angle of 15 to 20 deg., and measuring 4 to 25 ft. in thickness. The mines in Kiushiu produce two-thirds of the entire output of Japan. The Miike mine occupies about 14,000 acres. The Takashima coal mines occupy three small islands, about seven miles from Nagasaki; their output ranks as the best of the coal mined in Japan. All the working galleries are situated under the sea. The price of coal at Yokohama during 1908 varied from $3.95 to $5.70 per ton. Nearly 3,000,000 tons of coal were exported during 1908. Almost half of the exports went to China. The total quantity of coal remaining unmined in the Hokkaido field is estimated at more than 600,000,000 tons, or about as much as exists in the whole of the rest of Japan. The Kayanuma coal mines in Shiribeshi province, near Oparu, were opened as long ago as 1821, but little coal is now produced there. The production of coal in the Hokkaido province has more than tripled during the last 10 years.

Russia. (By I. I. Rogovin.)—Coal mining was carried on in 1908 in the same districts as in 1907, the Donetz basin, South Russia, being the most important both in respect to quantity and quality. The production

of coal in that district showed a small increase in 1908 and there was also an increase in the production of coke. The total production of coal in Russia in 1909 is expected to be 1,155,000 poods; and of coke 29,000,000 poods. In the Donetz mines 115,000 men were employed in 1908, against 122,602 in 1907. The coal mining district of next importance is the Dombrowa, in Poland. The output of this field in 1908 was 4,704,561 metric tons, against 4,486,475 in 1907. Bituminous mines to the number of 27, employing 21,480 men and brown coal mines to the number of five, employing 550 men, were in operation in 1908. The production of the Urals in 1908 was about 44,000,000 poods; of Siberia, about 100,000,000 poods.

Transvaal.—There was no great expansion of coal mining operations in South Africa during 1908, although the output increased somewhat. The efforts to maintain the price of coal were unsuccessful, so that during the year, despite a somewhat larger output, the total value realized did not advance at all. The value of coal raised in the Transvaal at the present day is considerably less than in 1904, although the average monthly output during 1908 was about 50,000 tons greater than in 1904. Transvaal coal mining companies distributed £94,380 in dividends during 1908. The Witbank Colliery Company, which is the leading coal mining venture in South Africa, accounted for one-half of this amount. Competition is now so keen in coal selling that there seems to be but little chance for new companies. Only the best organized and equipped propositions have been able to declare dividends. In some quarters it is believed that before 1909 is concluded the price of fuel will touch even lower figures than those quoted in 1908.

Turkey.—In the Vilayet of Smyrna there are about 60 coal mines being worked under Firmans and 75 under licenses. On the shores of the Black Sea the coalfields of Heraclea form a source of vast wealth to the Turkish Empire. The beds extend along the shores of the Black Sea for nearly 40 miles. The coal lies in thick seams, but contains so many impurities that it cannot successfully compete with English coal. In other parts of the empire there are also extensive deposits of coal, but want of proper communication with the outside world has thus far rendered it impracticable to exploit them.

United Kingdom.—According to returns for 1908, the exports of coal amounted to 62,547,175 tons, compared with 63,600,947 tons in 1907. The exports in 1908 were made up of the following kinds: Steam, 45,129,194 tons; gas, 10,541,658; anthracite, 2,274,792; household, 1,558,320; and other sorts, 3,043,211. The total value of coal, coke and patent fuel exports in 1908 is given as £41,615,923, against £42,118,994 in 1907. The production of coal in 1908 was 261,506,379 metric tons.

a decrease of 6,321,897 tons. When we consider that the production of coal in the United Kingdom in 1907 was the greatest ever recorded, showing an increase of nearly 17,000,000 tons over the production of 1906, the totals for the industry in 1908 are gratifying. Practically all of the coal companies whose financial years ended in June showed excellent results. Although the volume of business was greater in 1907, the product was supplied at relatively low figures, so that when new contracts were made in 1908, the average selling price was slightly higher. In the second half of 1907, the Welsh collieries did better under current than under contract sales; in the first half of 1908, however, and particularly during the second quarter, these same collieries made larger profits under contract than under current sales, so that in the 12 months ending with June, 1908, the South Wales collieries obtained their largest profit.

West Indies.—A considerable deposit of coal has been found in the Dominican Republic in the valley lying between the Gran Cordillera, and the Monte Cristi chains of mountains. Anthracite coal has also been found in the Tamboril province of Santiago, in the valley above mentioned.

The Coal Markets in 1908.

The general coal trade in the United States showed a falling off of from 20 to 25 per cent. in 1908. The anthracite markets were less affected than the bituminous trade, and prices in the former held firm. Anthracite shipments for the year showed a decrease of 2,444,379 tons, which falling off is very slight when it is recalled how large was the increase in 1907.

SHIPMENTS OF ANTHRACITE.
(Tons of 2000 lb.)

	1906		1907		1908	
	Tons.	Per ct.	Tons.	Per ct.	Tons.	Per ct.
Reading	11,258,295	20.2	14,018,795	20.9	12,578,883	19.4
Lehigh Valley	8,536,254	15.4	11,532,255	17.2	10,772,040	16.7
N. J. Central	6,983,217	12.5	8,714,113	13.0	8,495,425	13.2
Lackawanna	9,201,875	16.5	10,237,919	15.2	10,088,697	15.6
Del. & Hudson	5,346,695	9.6	6,562,768	9.8	6,461,666	9.9
Pennsylvania	4,856,004	8.8	6,203,171	9.2	6,019,457	9.3
Erie	5,636,537	10.2	7,151,683	10.7	7,450,175	11.5
N. Y. Ont. & Western	2,444,273	4.4	2,689,089	4.0	2,798,671	4.4
Del., Susq. & Schuylkill	1,435,445	2.4	(a)	.....	(a)	....
Total	55,698,595	100.0	67,109,393	100.0	64,665,014	100.0

(a) Shipments included in total of Lehigh Valley Railroad Company.

Two companies showed increases in 1908—the Erie and the New York, Ontario & Western. The largest proportional decrease was on the Reading; the smallest on the Delaware, Lackawanna & Western.

Pittsburg. (By S. F. Luty.)—Coal production in the Pittsburg district in 1908 did not exceed 42,000,000 tons, a decline of about 10,000,000 tons compared with 1907. The Pittsburg Coal Company produced about 13,000,000 tons as against a trifle over 15,000,000 tons in 1907. The Monongahela River Consolidated Coal and Coke Company produced in round numbers 6,000,000 tons, or 1,500,000 tons less than in 1907. Of this tonnage 2,250,000 tons went to lower river ports, and 1,500,000 tons was shipped by rail. The rest of the tonnage was distributed among the manufacturing interests along the Monongahela, Allegheny and Ohio rivers and in the home market. Other river-coal interests produced about 2,250,000 tons, or about 750,000 tons less than in 1907.

River shipments were very good for the first five months of 1908, but subsequently the rivers were not navigable. What coal was not consumed in the home market, or shipped by rail, was still in the pools and harbor at the close of the year, and amounted to 700,000 tons. When navigation was suspended in June there were enough empty coal boats and barges in the pools to keep the mines running for two months. After that operations were practically suspended, only rail shipments being made. About the middle of December the rains brought the rivers to a stage that permitted the return of empties and a partial resumption of the river-coal mines followed.

The lake-coal trade also was disappointing. In 1907 the Pittsburg Coal Company and other interests shipped 10,485,000 tons to the northwestern market; in 1908 shipments fell off to around 8,000,000 tons. One reason was the late opening of lake transportation. In former years ore was sent down the lakes as soon as navigation was opened, but owing to conditions in the iron and steel trade no ore was shipped until June 1, and as coal is shipped on the return trip of these boats, none went out before the middle of June. Shipments to lake ports began at the usual time, but the docks at Cleveland soon became congested, and there was considerable delay.

Prices remained throughout the year on the basis of $1.15 for mine-run coal at the mine for both river and rail shipments. Producers figured that as the demand was not normal no advantage would be gained by cutting prices as it would not bring any more business. Some of the small producers, however, did shade the price, but not sufficiently to disturb conditions. Slack fluctuated, the highest point being 80c. and the lowest 40c. a ton at the mine. A general suspension occurred on April 1, owing to the expiration of the two-year wage agreement on the mining rate with the United Mine Workers of America. The interstate agreement was dissolved when the joint convention, held at Indianapolis in January, failed to reach a settlement. The mine workers' organization

demanded a renewal of the wage scale, which was on the basis of 90c. per ton for pick mining over a 1¼-in. screen. The two-year agreement expired on March 31, and the miners wanted an agreement for a period of one year. At a conference held in Toledo in April, a settlement was reached and a contract for two years was made. For the first time in many years there was no complaint of a scarcity of railroad cars.

Coke production in the upper and lower Connellsville fields in 1908 was fully 5,000,000 tons less than in 1907. The output of the H. C. Frick Coke Company was, in round numbers, 7,000,000 tons; the independent interests produced about 6,000,000 tons. At times during the year less than 50 per cent. of the ovens were in use. At the opening of the year furnace coke had dropped to $2 and foundry coke to $2.50. Later, sales of furnace coke were made at $1.60@1.75, and foundry at from $1.90@2. On account of the low prices, the H. C. Frick Coke Company made a reduction in wages effective Jan. 1, 1908. No change was made in some classes of labor, and the reductions in others ranged from 5 to 12 per cent. The readjustment was not a surprise, as coke prices had declined fully $1 a ton compared with the prices in the first quarter of 1907. On March 1 of that year the Frick company voluntarily advanced the wages of its workers from 8 to 12 per cent. At the close of 1908 furnace coke for spot shipment was quoted at $1.85 and on contract at $2. Foundry coke late in December was quoted at $2.25@2.40 for both spot shipment and on contract.

Chicago. (By E. Morrison.)—From the point of view of the dealer in steam coals, particularly those from the Illinois and Indiana fields, 1908 was disappointing. The demand for such coals was low throughout the year; although it steadily improved in the last three months, the average was probably 20 per cent. lower than in 1907. Prices of Western bituminous were low, but continued without any material reduction from 1907, when keen competition brought them to a low level. Domestic coals were low-priced. The hard times aided the western producers somewhat by bringing into demand low-priced bituminous as a substitute for anthracite and high-priced eastern bituminous.

PRICES OF WESTERN COALS, CHICAGO.

(In tons of 2240 lb).

Month.	Lump and Egg.	Run-of-Mine.	Screenings.	Month.	Lump and Egg.	Run-of-Mine.	Screenings.
Jan ..	$2.00@$2.75	$1.60@$1.90	$1.20@$1.60	July	$1.75@$2.25	$1.50@$1.75	$1.40@$1.60
Feb	1.90@ 2.75	1.60@ 1.80	1.20@ 1.50	Aug	1.75@ 2.25	1.60@ 1.70	1 35@ 1.50
March ..	1.50@ 2.60	1.60@ 1.80	1.30@ 1.50	Sept.....	1.75@ 2.45	1.55@ 1.75	1.25@ 1.50
April ...	1.75@ 2.30	1.50@ 1.75	1.30@ 1.60	Oct.....	1.75@ 2.25	1.65@ 1.75	1.20@ 1.50
May	1.75@ 2.25	1.65@ 1.85	1.30@ 1.60	Nov	1.75@ 2.65	1 65@ 1.75	1.00@ 1.55
June . .	1.75@ 2.25	1.65@ 1.75	1.40@ 1.60	Dec.....	1.75@ 2.65	1.65@ 1.75	1.10@ 1.65

The demand for anthracite was quiet throughout 1908, car prices being the same as in 1907—$6.50 for egg, stove and chestnut, and $6.25 for grate. As for several years previous, the graduated monthly discount system was in use between April 1 and Sept. 1, giving 50c. off on April sales, 40c. on May sales and thus down to 10c. off on August sales.

Eastern bituminous coals suffered throughout 1908 from a tendency of steam users to seek low-priced coals, and throughout the summer and autumn from over-supply. High-grade smokeless coals sold fairly well as a result of persistent advertising; low-grade smokeless was moved with difficulty and was constantly sold at less than circular prices. Smokeless brought $3.05@3.30 for run-of-mine, and $3.85@4.05 for lump. Hocking shipments were well restricted and though the demand was not heavy it was fairly uniform, the price ranging $2.95@3.15. Youghiogheny was very quiet and steady throughout the year at $3.15@3.25 for ¾-in. gas. Many producers of bituminous coals learned greater caution about sending surplus products to the Chicago market, in 1908. The desirability of regulating the supply so as to prevent flood waves of shipments of one kind of coal or another has often been pointed out and was emphasized by the history of 1908. In default of such regulation, shippers found most uniform success in closely watching the market rather than in taking speculative chances.

Seaboard.—The Seaboard coal trade comes under two heads, anthracite and bituminous. The anthracite coal trade is usually fairly steady, the greater part going to the domestic trade, which is not affected to any great extent by business fluctuations. A mild winter will make more difference in this trade than the condition of business. A portion of the anthracite trade—about 25 per cent.—is in the small, or steam sizes, as they are generally called, and to some extent this is affected by trade conditions. So large a part of this coal, however, is consumed in the cities, chiefly in the heating of large buildings and in operating electric light plants, water-pumping stations and similar enterprises that the variation is, after all, not very much greater than in the domestic trade. As is shown by the accompanying table, the coastwise shipments, which are chiefly to New England ports, experienced a considerable decrease in 1908, but this was due mainly to the fact that two mild winters in succession reduced the consumption.

The Seaboard bituminous trade was dull throughout 1908, especially in New England, where the demand from manufacturing plants for steam coal was small for nearly the whole year. Summer shipments to that territory were light and it was a marked feature of the trade that what are known as the shoal water ports, i.e., those which are closed by ice in the winter, took very small stocks. One consequence of the light

trade was that the bituminous coal sold at tide-water at lower prices than had been recorded for several years.

COASTWISE COAL SHIPMENTS.

(In tons of 2240 lb.).

	1907.			1908.		
	Anthracite.	Bituminous.	Total.	Anthracite.	Bituminous.	Total.
New York (a)	16,753,914	11,691,101	28,445,015	15,069,981	10,247,014	25,316,995
Philadelphia	2,411,521	5,095,473	7,506,994	2,164,747	4,675,767	6,840,514
Baltimore	266,062	3,804,066	4,070,128	251,739	3,704,851	3,956,590
Newport News	.....	2,396,406	2,396,406	.....	2,742,294	2,742,294
Norfolk	.....	1,951,747	1,951,747	.....	1,651,093	1,651,093
Total	19,431,497	24,938,793	44,370,290	17,486,467	23,021,019	40,507,486

(a) New York includes all the New York harbor shipping ports.

Recent Practice in Coal Mining.

The matter of coal-mine accidents, their cause and prevention, was the all-important subject of discussion during 1908. The year brought forth many new, if not startling, ideas. In the expression of these opinions, our most prominent engineers have differed widely in their views. Several have asserted that we are sending too much air through our mine workings, and that before we adopted such elaborate systems of ventilation, explosions, especially those due to coal dust, were not so frequent. It has also been stated that the practice of sprinkling our mines in order to settle the dust and prevent dust explosions is in error; the opponents of this plan of introducing moisture into mines, base their opinion on the idea that excessive moisture in mines causes roof falls, and thereby adds largely to our list of fatalities. However, it is safe to say that the majority of mine managers do not agree with this opinion and believe that the introduction of a reasonable amount of moisture into mine air, as well as the practice of sprinkling the dust at the face of rooms and entries before shots are fired, is a precaution that insures greater safety to the miner.

There is no question that the great number of roof falls is a serious menace; however, it is doubtful if we can safely reduce the volume of air at present being used or neglect to sprinkle dusty working places. It is probable that when such problems are better understood we shall find that underground, as on the surface, rock disintegrates not so much from either dampness or from unusual dryness, as from the frequent change to one or the other of those conditions. If we could keep the moisture in our mine air at a certain standard all the year round, we should materially reduce the number of roof falls.

Coal Dust Explosions.—The problem of eliminating coal dust as a factor in mine explosions was the principal subject of attention in 1908. The first precaution is to remove as much of the dust as is practically possibly. After this has been done, there still remains an immense quantity of very fine dust that must be taken care of. This will not explode so readily or lend itself to a gas explosion so quickly when it is damp. Therefore, it seems that the proper thing to do is to dampen it. Experiments by English engineers have shown that when rock dust is mixed with coal dust, the latter is rendered inexplosive. Whether this can be brought to a point where it will be of practical benefit, remains to be determined. There are many other modern practices that are looked upon with disfavor by a few. Several operators have recently declared themselves as opposed to mining machines, and have stated that in solving our difficulties we must again return to pick mining. It is probable, however, that all of our advanced practice has come to stay.

Blown-Out Shots.—An important problem yet to be solved is that of shot-firing. In bituminous mines the majority of explosions are due to blown-out shots. The powder manufacturers, and the Federal Government, at its experiment station in Pittsburg, are rendering great service in providing us with a better knowledge of the correct kind of powder to use. The practice must eventually be adopted at every coal mine to have all the shots fired by special shot-firers. Furthermore, all holes should be fired when the miners, with the exception of the shot-firers themselves, are out of the mine at the end of the day's work. There are at least 100 important mines in the United States today where the continual shooting in the rooms makes the air so cloudy from smoke and dust that the surveyors cannot see a sufficient distance along the entries and up the chambers to put in their sights properly. Promiscuous shooting of this sort is dangerous and is sure to result in disaster. When it is suggested to operators that they should have all shots fired by regularly appointed men the reply is that the plan is impracticable because of the increased expense. However, there are mines in this country, as well as in Europe, where such shot-firing is carried on economically. I do not believe that conditions in any field in this country are such as to prohibit shot-firing by specially appointed men after the day's work has been done. It has been suggested that this practice is dangerous, especially if the miners charge their own holes, for when they do not fire the shots themselves, they are not so careful about loading and tamping the holes. This point is undoubtedly true to a small extent and cannot be entirely eliminated so long as we have to deal with the personal factor. It is better, however, if we must have explosions, that eight or 10 men lose their lives than 10 or 20 times that number.

Enforcement of Mine Rules.—We have examples every day where miners deliberately disobey rules, and endanger not only their own, but the lives of fellow workmen. This can only be avoided by stricter supervision, and if necessary, by having a number of additional police underground. The Marianna mine in Pennsylvania, where the worst disaster of 1908 occurred, was designed to be a model operation; it was, however, as dangerous as any poorly planned and badly managed property could be, because of the violation of rules by miners who carried black powder into the workings, and even tamped their holes with coal dust. We must hold the miners responsible and see that they are severely punished for any infraction of rules. On the other hand, operators should be held responsible if they do not properly police their mines, and thus prevent such reckless violations. Coroner's inquests are laughable comedies, where no one but the dead are held responsible, and where the defendants suggest the questions put to themselves.

The Cost of Safety Precautions.—We hear much about the perfection of coal mining in Europe, and are constantly reminded that in the United States our death rate is higher than in any other important country, which causes us to infer that our mine managers are either less intelligent or more negligent than operators in other countries. The whole question resolves itself into the simple fact that our mine owners have gone into this particular business with the hope of securing a fair profit on their capital, and conditions of the coal trade in this country have made it impossible to expend much money on improvements not absolutely necessary. European coal operators sell their coal at the mines at a much greater profit than can operators in this country. Our anthracite industry is well in hand, but the bituminous trade is without effective organization, which condition causes bitter competition among independent companies, and reduces the selling price of the product to a minimum. There is consequently but small margin available to be expended in safeguarding the men or mines.

The cost of installing an efficient system of sprinkling varies with the locality. At the Llwnytia colliery in Wales, there is a water main passing through all the principal entries; at a distance of 200 yd. from the shaft the entries are sprinkled with a hose and nozzle; beyond this distance the mains are furnished with jets from point to point, which spray water into the entries. The cost of installing this watering plant was as follows: 8.8 miles of 1½-in. pipe, $5000; junctions, $105; brackets and hangers, $165; ¾-in. nipples, $340; labor of installation, $480; hose, $60; making a total cost of $6150. At this mine the expense of watering amounts to 0.4c. per ton of coal hoisted; this figure includes interest on the cost of the plant, but not its depreciation, nor the cost of the water,

which might in some circumstances be considerable. At several other collieries in Wales, the cost of watering is estimated at 2c. per ton. In the United States, perhaps the best system of underground spraying is at the Sunnyside mine of the Utah Fuel Company in Utah. The installation of a complete spraying system at this mine cost approximately $7000, which expenditure, if distributed over a year, would amount to about 2½c. per ton, where the annual production was about 300,000 tons. If this same initial outlay were distributed over a period of five years, the cost would be less than 0.5c. per ton of product. The total operating cost of this same system amounts to less than 1c. per ton.

There is one other colliery safeguard that the future will see installed at all modern coal mines, and that is a shot-firing system where electricity will be the agent to explode the shots. To install an efficient shot-firing system at a mine having an output of not less than 250,000 tons per annum, costs about $1250, which if paid by one year's output, would add less than 0.5c. per ton of product. At one of our modern mines, the annual operating cost of a shot-firing system has not exceeded $1900, which would increase the cost of coal, if the plant produced 300,000 tons annually, about ⅔c. per ton. From the foregoing figures, it is evident that a mining company producing 500,000 tons annually could spray all its underground working and fire shots electrically by expending about $10,000 per annum. This added expense is not a great deal when we place a fair value on the lives and property that would be saved.

The Use of Electric Power in Coal Mining.—Electric power is now used not only for haulage, but also for a multitude of other purposes, such as driving coal-cutting machines, operating mine pumps, winding engines, etc. On the surface, the most modern screening plants recently constructed at European collieries are entirely driven by electric power from a main generating station; each unit should be provided with a separate motor, so that the break-down of any one part will not seriously affect the working of the plant as a whole. One special feature to be observed in constructing such plants is to see that ample room has been allowed around the various parts, so that all machinery can be easily repaired.

The increased use of electricity in mines has developed the idea of centralization in the generation of power. Transmission can be made in the form of electricity over a wire with far less loss than it can be through a steam pipe, by compressed air or in any mechanical form. As to the choice between direct current or alternating current, the former can be used economically where the distance does not exceed a certain limit, beyond which the cost of the wire becomes prohibitive. If it were desirable to transmit 200 kw. to a mine one mile distant from the power

house, the cost of copper for transmitting this at 250 volts direct current, with 10 per cent. loss in transmission, would be about $26,000; the wire would weigh about 130,000 lb. The cost of copper for transmitting the same power the same distance, with 10 per cent. loss at 2000 volts alternating current, would be about $400; the wire would weigh approximately 2000 lb. The cost of copper varies inversely as the square of the voltage. One of the most modern of central power plants has been erected by the H. C. Frick Coke Company, at Connellsville, Penn. This plant has a capacity of about 2000 kw.; the heat for the boilers is obtained from a number of bee-hive ovens, which are connected with flues, so that the waste heat from the ovens is utilized.

The Installation of Electric Power.—Electricity has often been blamed for accidents in coal mines, where many other causes might as consistently have been found. Of course, the proper installation of electric power in coal mines is most important. The latest ideas along this line, resulting from extended experience, may be briefly summed up as follows: Switches should be of an approved hight in boxes. Each motor and locomotive should have an inclosed copper wire fuse on the main circuit. All forms of stationary motors should have the frames grounded in an approved manner. Controller frames on electric locomotives should be grounded to locomotive frames. Lightning arresters should be placed at the entrance of each mine. Where mining machines are used in rooms the frames of motors should be grounded to the frames of machines, and the machines should be connected to the return circuit. Where it is necessary to install wires in rooms, have an approved automatic cut-out or switch, and install in the room, in view of the miners, two lamps in multiple. Where electric fans are used for ventilation, an alarm should be so arranged in the power house that when the fan stops the power-house engineer will get a signal. If the mine is gaseous, he should have instructions to cut power off the mine at once. As far as possible, fan circuits should be separate from all other circuits.

The extent to which electric motors are applicable in the case of engines, such as winding, pumping, and haulage engines, is a question requiring careful thought. As a general rule, the electric motor is applicable to all small engines, and to large engines which run continuously, such as fan engines and pumping engines; however, when applied to haulage at high speed, where there is a continual starting and stopping, the advantage of the electric motor is less apparent. If the haulage is done by endless ropes and is practically continuous for eight or 10 hours a day, at a uniform speed, then the electric motor is eminently suitable.

Electrically Operated Breakers.—The Lackawanna Coal Company, in Pennsylvania, was one of the first to experiment with electrically oper-

ated breakers. For several years an experimental breaker has been installed at the lower end of the Wyoming valley. Individual motors are used for driving the different parts of the breaker, and the results have so far proved satisfactory. The complicated machinery required to operate a breaker by steam has always necessitated a certain loss of energy and time. Indeed, the location of the gears, belts and shafts requires a nice designing to eliminate, so far as possible, loss of efficiency. The electric motor simplifies the designing and makes the operation of the different parts easy. One of the largest electrically operated breakers in the anthracite region has been installed by the Delaware & Hudson company; the saving obtained so far shows a considerable economy of operation. The saving is not only of fuel and time, but also of operative labor. The elimination of workmen is one of the greatest gains that electricity makes. One man can generally do the work of two or three where machinery is operated by electricity. The perfect control of the machine by one guiding hand always produces higher and better results.

Mine Haulage.—Underground haulage in coal mines is generally accomplished by mules, electric locomotives, compressed air locomotives, or oil-burning locomotives. Mules are generally used for short hauls from the face of rooms or entries to the nearest parting, where trips are made up and usually hauled to the shaft or to daylight by some mechanical means. The question as to whether electric power or compressed air is most economical for haulage purposes underground, continues to be debated by the advocates of the two systems. Electric locomotives have come into most prominent favor during recent years in mines where dangerous gases are not present. In gaseous workings, however, it is sometimes necessary to eliminate all fire risks, and in such mines compressed air locomotives are frequently used. In one gaseous colliery of Pennsylvania, where the operating conditions require that heavy loads be hauled up grades as steep as 5 per cent., and around curves of 100-ft. radius, eight compressed air locomotives were recently installed. At this mine the clearance limits were so restricted that it was found impossible to design a single locomotive capable of meeting the conditions. The locomotives, therefore, were arranged to be operated in tandem, and are coupled together back to back, so that one operator can control the two units by a single throttle lever, reverse lever, straight valve, etc.; by means of a flexible connection between the air tanks, the locomotives can be charged simultaneously. When the locomotives are in operation, this flexible connection is closed, but the auxiliary air tanks are then in communication through a rubber hose coupling, so that the cylinder pressures of the two units are equalized. The pressure is maintained by the

reducing valve at 150 lb. per sq.in., the initial pressure in the storage tank being 800 lb. per square inch.

The Lehigh Valley Coal Company, as an experiment, has installed an oil-burning locomotive in one of its mines at Hazelton, Penn., and another at Seneca. The increasing length of hauls has caused the expense of underground transportation to grow rapidly in the last few years, and it is hoped that the oil-burner will bring about an economy in this branch of the industry. Of course, these motors can only be used in mines free from gas. The advantage of oil as a fuel for use in the mine is that its combustion creates no fumes to choke the gangways and interfere with the breathing of the miners. Smoke from these oil-burners is a negligible quantity. The new motor at Hazelton weighs 15,500 lb., is 15 ft. long and 5 ft. high. It has four driving wheels, 24 in. in diameter, and runs on a track of 42-in. gage. It has been built to take up as little space as possible. The top of the smoke stack is even with the top of the boiler, and the cylinders lie close to the side. The oil-burner at Seneca is practically the same, except that it runs on a 3-ft. track. Either of the two machines is capable of hauling from 12 to 15 loaded mine cars up a 3-per cent. grade, and is expected to do the work of 10 mules.

Coal Cutting by Machinery.—It has been found that when the coal seams and other conditions are favorable, the output per man may be doubled when machines are used, while the cost of production is appreciably reduced and an increased value of output per ton is obtained. In specific cases the cost per ton has been 42c., where hand labor had cost 60c.; the output per man being 4½ tons as compared with 2½ tons. The proportion of round coal with machines has also been raised 25 per cent. and greater safety has been insured to the miners. It must be confessed, however, that in some coal seams machines have proved unsatisfactory. Where comparatively little holing is required, and when the roads are of such a nature that close propping is necessary for safety, it is probable that coal cutters will lack favor. It is also true that in some thick seams, where the lines of cleavage occur with regularity, but little labor is required to work the coal, and in such instances mechanical coal cutters will not be adopted. The motive power for driving coal cutting machines continues to be a debatable point. In Scotland, electricity is chiefly in favor, 171 machines being thus driven; 134 are driven by compressed air. In this district electric power has grown in favor most rapidly. On the other hand, in the Newcastle-Durham district, compressed air is used for 140 machines, while only 79 are driven by electricity. In Yorkshire, the proportion is 158 to 94, and in Lancashire 146 are driven by compressed air and only 14 by electricity. Danger is feared from electrically driven machines in gaseous seams.

Some engineers suggest that in fiery mines suitable air compressors should be erected and driven by electric motors, properly encased and placed in the main intake at safe distances from the face, and that the coal cutters should be driven by compressed air from them. This method would be a practical and useful adaptation of electricity to the present method of working by compressed air, thus avoiding the serious risks which are incurred by carrying the currents to the faces.

The advantages of coal cutting have been summed up as follows: An increased percentage of large coal is obtained, and the coal mined is in a firmer and better condition; a more regular line of face is obtained, leading to more regular and systematic timbering; increased safety conditions for the workers; thin seams can be profitably worked; less small is made than in holing by hand; increased output and greater regularity; facility of supervision; a smaller quantity of explosives is required for getting down the coal and in some cases none.

As a general rule, a coal cutting machine requires a space of about 4 ft. clear from the face, so that it may work continuously and without danger of being buried; for this reason, many superintendents advise against the adoption of coal cutters in a mine where the roof has a tendency to fall. It may be stated in this connection, however, that mining machines are now being successfully used under a bad roof, the timbering being done from behind. If the work is properly blocked out, one machine with three attendants will do as much work as 20 men by hand labor. One great objection to the use of mining machines is the scarcity of suitable men to work them. The man who operates a coal cutter should combine the qualifications of a practical miner and a mechanic. It appears, however, that men with mechanical ability prefer to work above ground, so that it is necessary to train miners for the work.

Minor Improvements.—The year 1908 will be known as a period replete with minor improvements rather than a year showing marked advances in actual methods of mining. These small advances have not attracted any great attention; however, the sum total of such improvements is considerable. We have practically witnessed the introduction of rescue apparatus, the installation of telephonic or telegraphic communication underground, the carrying out of the idea of establishing a careful system of shaft signaling, where two separate systems of signaling are used, so that one will be intact in case of accident to the other, the use of electric safety lamps, the more general installation of coke-drawing machinery, the construction of rectangular ovens, and the successful use of reinforced concrete for construction work about mines. Several of our recent mine explosions have also forcibly reminded coal operators

that entries and other narrow work must not be driven ahead of the air, with the expectation that air can be safely carried to the working face by means of temporary brattices, and that the one rational method of underground development is to work the seam in panels or isolated sections. This idea of isolating mine areas has again been brought to our attention by a West Virginia engineer, who has suggested a unique and original plan.

Isolated Panels.—His idea is to confine the deadly gases generated by an explosion, to the panel in which they originate. He asserts that by thus sealing these free men in their panels they will have an additional measure of protection until rescue parties can open the roadways to them. The scheme suggested to accomplish this is to have a series of boiler-plate doors installed in the different headings of each mine, two such doors on the intake and two on the air course of every panel. The frames holding the doors are recessed until they are flush with the ribs and roof. The frames are faced with several thicknesses of canvas to make a close joint when the doors are closed. The doors are hinged on the overhead timbers and make a snug fit; the bottom of each door is swung to within 6 in. of the roof—one pointing to the right and one to the left—and supported in this elevated position by detachable counterweights or by flat triggers that can be displaced by an air pressure slighty in excess of the regular current; a further precaution for closing the doors is to have tripping levers controlled from the partings. In case of an explosion, the unusual air pressure that is suddenly forced through the entries, closes the doors to the various panels.

The idea that a sudden compression of the mine air occurs in case of an explosion, was largely discussed at a recent meeting of mining men at Cardiff, Wales. At this convention, a prominent engineer propounded the theory that a huge force may be suddenly developed by the primary explosion in a coal mine, and that parts distant from the originating center, and free from gas, are devastated as the result of enormous pressure set up by the first explosion. The exponent of this theory stated that many men were instantaneously killed by an explosion before any signs reached them to warn them of danger. One instance was cited where a miner was found dead sitting with his food on his knee. Another instance was that of a man who was found dead actually standing with his drill in his hand after an explosion in the Universal pit, South Wales. Another instance upholding this percussion theory was that of two horses being killed, in whose blood no trace of carbon monoxide could be found.

Earthquakes and Colliery Explosions.—Some students of mining phenomena have lately been calling attention to the possible relation between periods of great seismic disturbance and colliery explosions. The year

1908 was a period of frequent seismic disturbances, culminating with the catastrophe in Italy in December; the many earth disturbances have been accompanied by an unusual number of colliery explosions. The coincidence between these earthquakes and coal-mine explosions has been remarkable. The subject is attracting international attention, and European engineers have gone so far as to suggest that a seismograph be installed at every mine to record the least earth movement and thereby warn the engineers of the possibility of danger. The theory of the influence of earthquakes and the conclusion that atmospheric conditions have a probable effect on the outflow of gases into mines, are questions that can no longer be ignored by the more enlightened mine operators.

Ventilation.—There has been much discussion of late as to the best standard of ventilation. The only practical method is to fix definitely the maximum percentage of marsh gas to be allowed in the return airways. Whenever the air contains a greater percentage of this gas than the amount prescribed, the ventilation should be considered foul. In dealing with ventilation in this way, the reason for fixing the standard by prescribing a certain percentage of marsh gas as the maximum permissible is the fact that when a dangerous amount of carbon dioxide is present the lamps would be extinguished before the gas would become dangerous to life. The greatest difficulty in fixing a certain standard of ventilation is the recognition that 1 per cent. of gas will create a condition of danger in a dusty mine, whereas 2 per cent. would not be any more dangerous in a mine free from dust. Any rules, therefore, will have to be modified in accordance with such conditions.

Mine managers have lately devoted more attention to good air and better sanitary conditions underground. In accomplishing this many of the larger companies have established a special testing department, where the quality of all oils and explosives is determined. The importance of selecting a proper miners' oil is not generally recognized. Aside from the necessity of carrying out the requirements of the law, there are advantages in supplying the miners with the best oil obtainable. As a general rule, an oil of high viscosity will not keep lighted in headings and air courses where there are strong currents; it is also hard to relight, and feeds up the wick very slowly. When buying winter oil, there should be a guarantee that it will not congeal at a temperature above 32 deg. F. In testing miners' oil there is nothing of greater importance than the quantity of smoke it gives off while burning. An inferior oil always makes dense smoke. An oil to which kerosene has been added (which is common practice in most regions) is the greatest offender in this way.

The subject of testing oils, explosives, and other experimentation, cannot be passed without some reference to the matter of tamping shot

holes. This has recently been the subject of considerable discussion by colliery engineers. The experiments of an English investigator show that owing to its inertia and friction on the side of the hole, tamping offers greater resistance to the escape of gases than is generally supposed. This same authority has found that in a 1-in. diameter hole, 2 oz. or 4 oz. of powder would not blow out more than 7 in. of clay tamping, and that when the blast breaks out its burden, such tamping is often found intact in the hole. For high explosives, the 12 or 18 in. of tamping often used is unnecessary, as a few inches would often be sufficient. Many experienced shot-firers claim that in general it is sufficient to use a tamping of water, or that a few inches of clay or paper pushed tightly home is all that is required.

Sanitary Provisions.—At an English colliery, where considerable attention has been directed to the proper sanitation of the underground workings, the manager has erected closets near the pit bottom and near one of the principal roadways of the mine; it is purposed extending such accommodations to other parts of the mine. Each closet is situated in a brick structure, provided with a wooden door, and consists of a galvanized iron pan containing coal dust, placed beneath a wooden seat. Each closet is satisfactorily ventilated with an iron pipe placed over the pan and communiciating with the return airway. The total cost of such a closet, including brick work, is less than $25. One of the stable men empties this pan about twice a week, taking the contents to the surface. It is stated at this mine that such sanitary arrangements are not necessary at the working face.

Education of Miners.—Many operators who have closely studied the coal industry in the United States assert that our greatest hope for the prevention of mine accidents and the conservation of our resources lies in the proper education of our miners. One Pennsylvania manager says, "Our miners today are only so-called miners, and mine superintendents are continually thwarted in their purposes by the enormous proportion of raw labor material that is taken into the mines daily. This raw material is the real cause of waste and is largely responsible for the many disastrous accidents. No further improvement in mining methods can be expected until we have better material from which to build up our miners. No great difficulty should be experienced in doing this, for in many other lines of industry much good has resulted from efforts directed toward educating the workers. When this is done, greater security and greater yield from our coal seams will result."

As an interesting side-light on the cause of mine accidents, it is worth remarking that it has been common practice among many miners to secure certificates of competency by fraudulent methods. The Pennsyl-

vania law requires that all men who propose to become miners in the anthracite mines, shall first serve two years as a miner's laborer, previous to appearance before the examining board to obtain a certificate, which credential is necessary to their employment as miners. The intention of this law is often frustrated by the rankest deception. When a foreign laborer has been employed at some of our mines two or three months, it often occurs to him that if he could secure a certificate, he could mine coal as well as the man he is working for. The general plan followed is to have one of his friends of the same nationality appear before the examining board, and after giving the name of the laborer who desires the certificate, the more experienced miner answers the questions propounded, pays the $1.50 fee, and sells the certificate. When such practices prevail, is it any wonder that accidents are often due to the ignorance of the miner himself?

Relief Fund.—Because coal mine accidents have left so many widows and children without support, it is being widely suggested that the several States levy a tax on coal to create a relief fund. The chief inspector of Pennsylvania has suggested that the State levy a tax of ¾c. per ton on bituminous coal, and 1½c. on anthracite. Anticipating an average increase in the production of coal in Pennsylvania, this tax would produce about $1,500,000 per annum. On the average, colliery accidents in Pennsylvania leave 574 widows and 1315 orphans each year.

Coal Briquetting.—Closely allied with the coal industry is the business of manufacturing briquets, but in the United States briquet making is only in its infancy. Experiments are now being made in the anthracite field, where the fine coal that is not of great commercial value, is being made into briquets. The briquet industry has reached its greatest development in Germany, where a large percentage of the brown-coal production is made into briquets. The principal reason for the success of the industry on the Continent is the fact that the brown coal will not stand transportation in its raw state, and consequently has to be briquetted. One of the most successful briquetting plants in the United States is that installed by the Rock Island Coal Mining Company at Hartshorne, Oklahoma. This company purchased the Norfolk plant, which had been operated for experimental purposes by the Federal Government. The equipment for this plant was built by the Renfrow Briquette Machine Company, of St. Louis, Mo. It has already been made evident to the briquet manufacturers of this country that the demand in the United States is for a briquet weighing from 8 oz. to 2 lb., of uniform quality and of such shape as to be handled by mechanical means. The briquets made at the Rock Island plant are shipped principally to Oklahoma, Arkansas, and Texas points, and find a ready sale in competi-

tion with fancy coal produced by other operators in Oklahoma. The plant has had to run two 10-hour shifts per day since Oct. 1, 1908, in order to meet the demand.

The Peat Industry.—This is a business that is closely related to coal mining and briquet manufacture. It is estimated that the world contains 200,000 sq. miles of peat bogs, of which 36,000 sq. miles are in the United States. Canada is credited with 37,000 sq. miles; Germany, 10,000 sq. miles, and Ireland 1100 sq. miles. If we assume the average depth of all the peat bogs in the world to be 7 ft., then for this depth every square mile would yield 784,000 tons of air-dried peat, and the estimated 200,000 sq. miles of peat bogs represent 156,800,000,000 tons of stored fuel. Considering that the heating value of the peat will average 4000 cal., or 7200 B.t.u., this is equivalent to 94,080,000,000 tons of coal of 12,000 B.t.u., or less than 4 per cent. of the estimated coal resources of the United States. Basing our figures on the present consumption of coal and lignite, without considering a future increase, our peat deposits would last about 75 years. The peat bogs in the United States would be exhausted in about 35 years if utilized as the sole supply of solid fuel. Up to the present time, no mechanical arrangement has been designed that will dry peat cheaply. The only economical drying agents so far discovered are the wind and sun. If some scheme be devised whereby peat could be produced as a fine powder for dust firing, the field of this fuel would rapidly attain importance. One reliable authority states that an economical plant ought to make peat fuel at 60c. per ton. This result is accomplished by first draining the water from the bog, and then using a dredge which can take up an entire section of peat, 12 to 14 ft. in depth, at one time. Recent tests have showed that a good quality of peat has a calorific value of 10,082 B.t.u. per pound. The principal difficulty in utilization of peat under boilers appears to be the frequency with which it is necessary to fire. This advantage is caused by the lightness of the material and also on account of its rapid combustion.

Cost of Drilling for Coal.—During the past few years, coal mine operators have made it a practice to drill carefully all coal areas before locating the shaft or in any way developing the property. This system has made it possible to form preliminary estimates as to the available coal tonnage, character of the seam, quality of the coal, and its physical aspects, such as its dip, thickness, and depth from the surface. Diamond drills have proved most satisfactory for this work. A careful record of the cost of drilling in the coal measures of Pennsylvania showed the following results: The drill used in this special case was of the Sullivan type, having a hydraulic feed and a double-core barrel, taking a 2½-in.

core. The outfit, with one good diamond bit, was furnished by the Birdsboro Steel Foundry and Machine Company, of Birdsboro, Pa., for about $3500. The boiler was of the upright type, behind the machine on a heavy wagon frame. The distance moved between holes averaged about a mile by road. It was open country with good roads, so that moving was not expensive. The core obtained was practically complete, both of rock and coal. The alluvium was from 6 to 19 ft. deep, averaging 9 ft. 9 in. It was clay, with no boulders, and was drilled out with a mud bit. The rate of boring through shale, fireclay and limestone, was about 7.1 ft. per hour of actual cutting; through sandstone, 9.35 ft. per hour; through coal, 15.15 ft. per hour. The total working cost averaged on five holes was 68c. per ft. Counting in the depreciation of the outfit (20 per cent. on $2000 for three months) raised the cost of drilling to 73.5c. per ft. Four holes put down by the same company in Raleigh county, W. Va., are said to have cost $2.90 per ft.

COPPER.

The domestic copper market began 1908 with an unsold stock of 120,000,000 lb. of refined copper. Although the smelters entered upon a severe curtailment of production as early as October, 1907, the refineries were of course obliged to continue at practically unreduced rate in order to work up the supplies of blister copper in transit and in process of treatment coming in for a month or more after the smelters suspended operations. Consequently, the production of refined copper in 1907 was not much less than in 1906. However, by the beginning of 1908 the refineries had also succeeded in restricting production and the latter

COPPER STATISTICS OF THE UNITED STATES.
(In pounds.)

	1902	1903	1904	1905	1906	1907	1908
Alaska	(a)	(a)	2,043,586	4,703,609	8,700,000	6,610,000	4,394,887
Arizona	119,841,285	153,591,417	191,602,958	222,806,020	263,200,000	256,866,761	290,167,795
California	25,038,724	19,113,861	29,974,154	16,697,486	24,421,000	34,398,823	36,890,353
Colorado	8,463,938	7,809,920	9,401,913	9,854,174	9,565,000	13,344,118	13,896,689
Idaho	(a)	(a)	5,422,007	6,500,005	9,493,000	11,471,101	8,749,559
Michigan	170,194,996	192,299,485	208,329,248	218,999,759	224,071,000	220,317,041	222,267,444
Montana	266,500,000	272,555,854	298,314,804	319,179,880	299,850,000	226,290,873	252,558,330
Nevada	(a)	(a)	(a)	(a)	426,000	1,462,450	12,174,269
New Mexico	(a)	(a)	5,368,666	5,638,843	6,262,000	8,652,873	8,523,652
Utah	23,939,901	38,302,602	47,062,889	51,950,782	49,712 000	68,333,115	70,978,952
Wyoming	(a)	(a)	3,565,629	2,393,201	146,000	2,919,137	2,384,356
Southern States	13,599,047	13,855,612	15,211,086	14,907,982	18,821,000	22,408,696	20,822,368
Other States	9,218,490	10,846,477	1,418,065	1,550,000	3,379,000	6,166,098	4,387,836
Total	636,796,381	708,375,228	817,715,005	875,241,741	917,620,000	879,241,766	948,196,490

(a) Included in "Other States."

was at a comparatively low rate in the early part of 1908, some of the chief mines of Butte, including the Anaconda, being idle. Domestic consumption was at a low ebb, but there was a fairly active export business, which had bid up the price about 1½c. above the low point of the last quarter of 1907. The shut-down at Butte was inspired by a variety of reasons. The Amalgamated had a large stock of copper on hand, the suspension of its smelting at Anaconda would be a salutory object lesson in several respects, and finally a readjustment of the labor situation would be facilitated. When the Amalgamated resumed in March, 1908, its cost of producing copper was about 10c. per lb. against 12c., approximately, under the inefficient conditions of 1906-07. John D. Ryan, managing director of the company, stated very plainly in his

announcement of the resumption that the men would have to work more efficiently than they had previously been doing.

At the opening of 1908 the copper market had recovered considerably from the low point of the panic, when certain producers had been obliged to dispose of their accumulation on the best terms offered. Under this condition electrolytic had been sold at 11⅝c. At the beginning of January, 1908, the price was about 13¼c. A fair demand from European buyers advanced the price materially during January, the average for that month being 13.73c., but from the beginning of February the market displayed a sagging tendency and by the middle of the month the price had fallen to 13c. One of the largest producers then began to sell at sharp concessions in price, and others, apparently becoming frightened, adopted the same policy, thus slaughtering the market, sales being made as low as 12c. In these operations an immense quantity of copper was unloaded, a large part of it going into speculative hands. The subsequent course of the market in 1908, which was a year of fluctuation between 12c. and 14⅜c., is related in a following section of this article.

PRODUCTION OF COPPER ACCORDING TO CLASS.
(In pounds.)

Year.	Total Domestic. (d)	Total Foreign. (e)	Grand Total.	Lake.	Electrolytic.
1897	501,370,295	26,938,254	528,308,549	145,839,749	250,000,000
1898	535,900,232	36,055,352	571,955,584	156,669,098	314,107,776
1899	581,319,091	40,659,868	621,978,959	155,845,786	396,410,356
1900	600,832,505	62,484,290	663,316,795	144,227,340	466,092,663
1901	597,443,212	.....	.....	155,570,465	(a)485,016,400
1902	636,796,381	.....	.....	170,194,996	(a)606,270,500
1903	708,375,228	.....	.....	192,299,485	(a)617,293,600
1904	817,715,005	.....	.....	208,329,248	705,478,400
1905	875,241,741	183,252,259	1,058,494,000	219,000,000	(b)760,000,000
1906	917,620,000	(c)247,549,000	(c)1,165,169,000	224,071,000	(b)865,000,000
1907	879,241,766	(c)273,506,124	(c)1,152,747,890	220,317,041	854,441,000

(a) As estimated by the Metallgesellschaft, Frankfurt am Main. (b) Partly estimated. (c) Includes copper from domestic scrap and junk. (d) Entered the same as production of the mines. (e) Difference between the first and third columns.

Year.	Lake.	Electrolytic. (d)	Casting. (d)	Pig Copper. (a)	Total.
1904	208,329,248	705,478,400	(b)45,000,000	44,408,000	1,003,215,648
1905	219,000,000	(c)760,000,000	46,000,000	33,495,000	(c)1,058,494,000
1906	224,071,000	(c)860,000,000	52,000,000	29,098,000	(c)1,165,169,000
1907	220,317,041	854,441,000	47,957,890	30,032,000	1,152,747,890
1908	222,267,444	850,660,325	44,967,250	35,000,000	1,152,895,019

(a) Exported. (b) Estimated. (c) Partly estimated. (d) Includes copper from scrap and junk.

Among the characteristics of 1908 were the prolonged period of dullness and the fitful periods of activity, in certain of which there were immense transactions, when the normal business of a month would be

concentrated in a week or 10 days. The large sales of February have been previously referred to. Another period of large transactions was early in August; another, the last, was just before and just after the election. Unlike the sales of February, those of August and later were made on a rising market, and signaled the first extensive interest of domestic manufacturers in providing themselves with copper. Domestic manufacturers were again large buyers in October and November, also on a rising market. Their willingness to buy under that condition inspired hope among the producers.

Another characteristic of the copper market in 1908 was the extensive speculative transactions. There are always more or less of these, but in 1908 the operations were extraordinary, being undertaken, not only by persons in the business, but also by outsiders. These operations were conducted chiefly in London, originating both there and here, while there were also speculative transactions in refined copper on this side. The latter are generally discouraged by the selling agencies, though there are apt to be overtures to transact business of this nature which cannot be declined, but in 1908 this kind of business was actually solicited in some cases. A good deal of alarm was felt during the last two months of 1908 over the large speculative commitments by outsiders; although the market showed a little weakness under realizations from less important speculators when demands from manufacturers became slack, danger from the larger speculative accounts was avoided up to Dec. 31, 1908; soon after that date their liquidation broke the market.

EXPORTS OF COPPER FROM THE UNITED STATES. (a)
Ore, matte and regulus stated in tons of 2240 lb. Ingots, etc., in pounds.

Country.	1903	1904	1905	1906	1907	1908
Ore, matte and regulus						
Exported to:						
United Kingdom	318	164	50	206	200	168
Germany	.....	102	.....	59	188	2
Brit. North America	.....	3,486	24,690	36,700	82,016	55,367
Mexico	10,667	15,175	12,948	10,600	16,737	7,060
Other countries	1,306	.....	.....	54	.....	552
Total	12,291	18,927	37,688	47,619	99,141	63,149
Ingots and scrap (b)						
Exported to:						
United Kingdom	47,140,717	112,224,371	60,945,794	55,097,070	81,409,441	117,810,314
Belgium	4,209,720	9,365,791	4,997,206	6,475,054	3,822,551	5,560,366
France	53,745,221	99,888,455	74,604,455	80,703,723	93,075,145	115,690,381
Germany	71,130,077	103,825,445	104,575,864	96,629,040	107,607,390	137,453,392
Italy	7,774,016	15,297,091	15,800,967	19,777,296	21,192,908	25,512,267
Netherlands	96,927,346	147,678,581	130,675,386	151,650,293	156,652,270	195,562,619
Russia	10,411,079	22,333,578	18,418,982	9,523,992	4,341,386	4,657,077
Other Europe	16,516,963	29,064,494	25,279,162	25,260,807	26,221,024	30,433,674
Brit. North America	2,644,331	3,472,514	3,019,450	4,176,135	3,747,410	3,977,142
Mexico	165,283	191,429	290,763	263,319	362,411	35,895
China	.....	10,403,034	79,940,250	4,932,128	10,003,592	13,735,899
Other countries	63,971	804,647	16,359,751	262,561	493,873	2,447,101
Total	310,729,524	554,550,030	534,907,619	454,752,018	508,929,401	661,876,127

(a) The exports of ore, matte and regulus are reported as gross weight, the copper contents not being stated.
(b) Includes bars and plates.

IMPORTS OF COPPER INTO THE UNITED STATES.
(In pounds.)

Country.	1903	1904	1905	1906	1907	1908
Ore and matte						
Imported from:						
Brit. North America	(a)243,918	15,046,131	15,403,429	10,329,955	12,803,069	8,326,728
Mexico	(a) 39,261	20,803,961	28,890,239	31,690,058	32,467,418	15,775,626
South America	(a) 77	91,509	1,503,427	4,140,589	8,790,621	9,054,852
Other countries	(a) 1,656	3,006,121	4,308,205	2,874,289	5,657,679	15,652,830
Total	(a)284,912	38,947,722	50,105,300	49,034,891	59,718,787	48,810,236
Pigs and scrap (b)						
Imported from:						
United Kingdom	18,788,558	19,172,854	26,284,302	22,549,321	25,706,852	5,434,435
France	1,926,279	22,075	1,549,138	3,202,168	606,662	168,506
Germany	1,600,766	875,329	2,945,441	5,303,712	6,814,338	1,451,370
Other Europe	240,689	16,943	1,955,358	5,649,689	5,616,261	13,359,117
Brit. North America	15,923,760	17,690,656	23,636,843	30,398,369	30,902,596	30,895,737
Mexico	89,361,100	97,965,593	102,646,343	85,595,359	76,741,532	43,742,993
Cuba	467,832	368,634	433,440	513,240	767,184	349,560
West Indies (c)	317,112	373,743	278,502	399,569	401,585	184,490
Japan	3,604,643	80	.....	6,752,486	9,809,569	8,329,896
Other countries	4,477,256	5,858,535	890,018	16,194,477	35,534,088	58,268,040
Total	136,707,995	142,344,433	160,619,385	176,558,390	192,901,267	162,184,144

(a) Tons of 2240 lb., copper content not given. The imports reported for 1904-1908 are the copper contents of ore, matte and regulus. (b) Includes also bars, ingots and plates. (c) Includes Bermuda.

Fears are frequently expressed that the world's resources of copper are not going to be sufficient for the further expansion of the electrical age that is anticipated, and that for this reason higher prices for copper are to be looked for permanently. Good copper mines are indeed hard to find, but among the many worthless properties developed, some good ones are found nearly every year, and the experience up to date shows that these have been sufficient to maintain the required production at an average price not very different from what copper fetches at present. The resources of the world still being so far from measured why should not this condition be expected to continue? We are now beginning to realize large amounts of copper from the porphyry ore of Bingham and Ely, which only a few years ago were not thought of as possible sources of production. In 1908 an immense new deposit of this kind of ore was blocked out in the Miami mine, near Globe, Ariz., which was the most important development of the year in America. This mine showed nothing but iron stains on the surface. The experience at Mt. Morgan in Queensland, and at Ely, Nev., and Globe, teaches that certain occurrences making no surface showing of attraction to the prospector of old may be underlaid by immense deposits of this kind of copper ore. This is an important geological discovery, leading to the belief that there are important copper resources, hidden from the eye, which may be found by the application of modern science.

Production.—The production of copper in the United States in 1908 is given in the first table. These statistics are compiled in the same way

as those for 1907, which for the first time stated the production upon the uniform basis of output of Michigan mines plus fine copper content of blister copper. The statistics for 1908 are computed from reports of all the producers.

The allocation of the production among the States and Territories can never be made with precision. So far as possible we have made a distribution in the first table of this article.

There are three ways of computing the production, namely, (1) on the basis of the mine output, (2) on the basis of the smelters' output, and (3) on the basis of the refiners' output. For any given year the three figures may be nearly the same, or they may be wide apart. The year 1908 was one in which they were wide apart. Under no circumstances do we place any reliance upon statistics of the mine production, because we know the insuperable difficulties of collecting them accurately. The figures of the smelters and refiners can be collected accurately, but they represent materially different periods of time, because the product of the smelters is upward of two months in transit and process of refining before it appears upon the market. Consequently in a year like 1908 which began with the refiners operating at reduced capacity and several important smelters like Washoe and Cananea closed down, resuming operations later in the year, the respective totals are bound to show a good deal of difference.

Analysis of the Statistics.—Ever since the development of the Bingham and Ely mines was begun we have been talking about the great increase in copper production to come from those places. The débâcle in the winter of 1907-08 diverted attention from what was impending and consequently the great increase in production in 1908, which occurred in the face of what is called a "low price" for copper, burst upon most persons as a surprise. There are indeed surprising features in the statistics, chief of which is the failure of a 13¼c. average to restrict more severely the output of the small mines, whose aggregate made a handsome total in 1907. Alaska, Idaho and New Mexico did, in fact, suffer in 1908, but Wyoming, Missouri and Virginia held their own, while Colorado made a respectable increase. Vermont receded, but only temporarily pending the beginning of smelting by the Vermont Copper Company. Many of the small mines are, therefore, still able to live. This being so, the increases in the big camps were all clear gain.

In Arizona—Bisbee, Globe, Clifton and Jerome made increases, in some cases very large ones. At Butte, in spite of the shut-down in January and February, there was a gain of 29,000,000 lb. The great expansion of operations by the Mammoth swelled the output of California. The enlarged output of Nevada is, of course, explained by Steptoe

Valley. The extension of the Garfield works in Utah made good the deduction of the United States, Bingham and Highland Boy smelters, and the Utah, Boston Consolidated and Utah Consolidated mining companies glutted it with ore. Indeed, the production of the mines in 1908 was doubtless larger than the production of the smelters, many of our reports showing much more copper received in ore than turned out as blister.

During the first part of 1908, Anaconda, Cananea, and three of the works at Salt Lake were idle. At that time production was below the rate of 1907. Anaconda resumed in March, and Cananea in the second half of the year. In the second half the Ely copper began to appear, and the porphyry mines of Bingham put additional capacity into operation. The Arizona districts and Lake Superior produced steadily right along. Lake Superior made about the same output as in 1907.

Stock on Hand.—At the end of 1908 there was 122,357,266 lb. of refined copper in stock at the American refineries. The visible stock in Europe was 124,723,200 lb. Yet the stocks on both sides of the Atlantic were of no such magnitude as to cause concern, if only consumption would regain its normal rate. The world's production of copper is now upward of 750,000 metric tons per annum. Assuming the consumption to be equal to the production, which is the case in the long run, about 60,000 tons per month are required. Consequently, the stocks in Europe and America were only about two months' supply. It is misleading to compare present stocks with those of 10 or 20 years ago when consumption was much smaller. In those former times it was recognized that two months' supply for the world was no more than is normally required as a safety reserve; if stocks were to be absolutely depleted, we should see such runaway markets, with their inevitable mischief, as we had in 1906-07.

CONSUMPTION OF COPPER IN THE UNITED STATES. (a)

Year.	Production.	Stock Jan. 1.	Imports.	Supply.	Exports.	Stock Dec. 31.	Consumption.
1901.....	597,443,212	93,050,230	176,472,369	866,965,811	227,194,184	197,857,698	440,913,929
1902.....	636,796,381	209,587,698	161,551,040	1,007,935,119	376,298,726	162,935,439	468,700,954
1903.....	708,375,228	162,935,439	167,161,720	1,038,472,387	312,822,627	230,111,792	495,537,968
1904.....	817,715,005	230,111,792	182,292,205	1,230,119,002	555,638,552	208,376,672	466,103,778
1905.....	875,241,741	208,376,672	210,724,685	1,294,343,098	548,772,403	132,587,496	612,983,199
1906.....	917,620,000	132,587,496	225,593,281	1,275,800,777	467,839,041	139,385,400	668,576,336
1907.....	1,152,747,890	9,000,000	5,000,000	1,166,747,890	508,929,401	120,000,000	537,818,489
1908.....	1,152,895,019	120,000,000	*Nil*	1,272,747,890	661,876,127	122,357,266	488,514,497

(a) The statistics in the above table up to 1906 inclusive are computed in the old way, namely, on the basis of the production of blister copper and the imports of copper in all forms. The stock on hand at the beginning and end of the year includes not only refined copper, but also the crude copper in transit and in process of refining. The statistics from 1907 and 1908 are computed on the new and more accurate method described in *Eng. and Min. Journ.*, July 25, 1908. Briefly, in this method the basis is production of refined copper, stock of copper in final marketable form and imports of refined copper. This change in method explains the erratic appearance of the figures for 1907 as compared with those for 1906.

AVERAGE PRICE REALIZED FOR LAKE COPPER. (a)

Year.	Number of Companies.	Pounds Reported.	Total Proceeds.	Price per Pound.	Quotational Average.
1905	9	82,372,955	$12,848,152	15.597c.	15.699c.
1906	13	113,411,645	21,714,068	19.146c.	19.616c.
1907	10	66,316,025	11,965,537	18.043c.	20.661c.
1908	15	125,949,248	16,907,498	13.348c.	13.424c.

(a) The statistics in the above table are compiled from the official reports of Michigan copper mining companies, except the column called "quotational average" which gives the figure reported by the *Eng. and Min. Journ.* This figure is the arithmetical mean of its monthly averages, which in turn are the arithmetical mean of the daily quotations. In general, the weighted average computed from the sales actually made by these companies agrees closely with the quotational average, but for 1906 and 1907 it was materially lower, owing to the highly erratic and exceptional character of the business in those years, particularly 1907, when for a long period quotations were made on comparatively small sales, while many of the producers were accumulating unsold stocks of metals, which were finally disposed of at relatively low prices. This was the year when the price of copper fell from upward of 25c. to less than 12c. per pound.

After all, 1908 was not a bad year, except to the wildcatters and the owners of so-called mines that are mines only when copper fetches 18 or 20c. The average price for copper in 1908—about 13¼c.—is nearly 2½c. below the average of 1898-1907, but that decennial average is inflated by the extraordinary prices of 1906-07. If the latter years be excluded, the average for 1908 is only a trifle lower than the previous decennial average. A large proportion of the copper of North America is produced at a cost of 10c. and under, meaning the real cost which includes all development, depreciation and maintenance charges.

Prospects for 1909.—As to the prospects for the near future: Production is now going on at the largest rate in the history of North America and it is still increasing. This is largely a matter of smelting capacity. Nearly all of the new works begun in 1906-07 have now been completed, but some have not yet gone into operation. Steptoe Valley and Cananea are just beginning to get their full gait; also Bully Hill and Balaklala. Teziutlan is ready to start. The Imperial works released smelting capacity at Douglas. Humboldt did not operate in 1908, but is likely to do so in 1909; also the works of the United States Smelting Company near Salt Lake. The installation of converters at Yampa in 1908 added to the capacity of Utah, which will be further increased in 1910 by the Tooele plant. It does not seem as if there would be any scarcity of copper. Given an increasing production the market will be determined by the needs of the consumers. Business is undoubtedly improving, but still very slowly. Some conditions lead to the belief that the revival in 1909 will not be so extensive as has been anticipated by the optimistic. It would not be surprising if the course of the copper market in 1909 should be on somewhat the same lines as in 1908.

Kinds of Copper.—The largest part of the copper produced in the United States is electrolytically refined. Some of the bessemer copper that is very low in gold and silver, like the product of certain mines in

THE WORLD'S COPPER PRODUCTION.
(In metric tons.)

Country.	1900	1901	1902	1903	1904	1905	1906	1907	1908
Argentina (a)	76	793	244	137	157	157	107	224	226
Australasia (a)	23,368	31,371	29,098	29,464	34,706	34,483	36,830	41,910	40,123
Austria-Hungary (a)	1,377	1,356	1,626	1,407	1,473	1,346	1,458	1,062	1,700
Bolivia (a)	2,134	2,032	2,032	2,032	2,032	2,032	2,540	2,540	2,540
Canada (b)	8,595	18,580	17,765	19,637	19,490	21,595	19,110	21,035	24,376
Cape of Good Hope (a) Cape Co.	4,491	4,064	2,794	4,704	5,563	5,105	4,003	4,298	4,559
Cape of Good Hope (a) Namaqua	2,337	2,439	1,727	610	2,337	2,337	2,642	2,540	2,440
Chile (c)	25,715	30,155	27,066	29,923	31,025	29,126	25,829	28,863	38,927
Cuba (d)	.....	.....	.....	.....	.....	.....	1,384	1,388	2,966
Germany—total (a)	20,635	22,099	21,951	32,214	30,252	22,492	20,665	20,818	20,523
(Mansfeld) (a)	(18,684)	(19,032)	(19,050)	(19,810)	(19,578)	(19,878)	(18,085)	(17,343)	(18,000)
Italy (a)	2,797	3,048	3,424	3,150	3,388	2,997	2,911	3,353	3,022
Japan (c)	24,317	27,392	29,034	31,861	33,137	35,944	36,963	39,840	44,700
Mexico—total (b)	22,473	33,943	36,357	46,040	51,759	65,449	61,615	57,491	38,190
(Boleo) (a)	(11,297)	(10,956)	(10,958)	(10,480)	(11,120)	(10,341)	(11,002)	(11,506)	(12,600)
Newfoundland (a)	2,929	2,800	2,906	2,753	2,235	2,316	2,332	1,758	1,453
Norway (a)	3,998	3,439	4,638	6,010	5,502	6,406	6,218	7,122	9,337
Peru (e)	8,353	9,673	7,701	9,497	9,504	12,213	13,474	20,681	22,240
Russia (f)	8,258	8,467	8,817	9,232	9,835	9,515	9,296	14,774	18,174
Spain-Portugal (a)	53,718	54,432	50,587	50,536	47,788	45,527	50,109	50,470	53,425
Rio Tinto	36,304	35,916	35,032	36,382	34,016	32,795	34,642	32,833	35,517
Tharsis (a)	8,092	7,546	6,817	6,421	5,710	4,415	4,816	4,206	4,500
Mason & Barry (a)	3,515	3,709	3,383	2,469	2,997	2,764	2,504	2,662	2,804
Sevilla (a)	1,483	1,313	1,570	1,123	1,351	1,300	2,073	2,337	2,196
Sweden (c)	136	137	178	776	513	1,385	1,209	1,577	2,032
Turkey (a)	2,341	1,665	1,118	1,422	965	711	432	1,270	1,068
United Kingdom (g)	777	541	490	545	501	727	762	711	.....
United States (d)	272,610	271,072	293,053	312,631	400,998	397,069	416,343	398,930	430,099
Total	491,435	529,508	542,606	630,590	693,240	698,931	715,510	722,655	762,111

(a) As reported by Henry R. Merton & Co., Ltd., of London. (b) As reported by Henry R. Merton & Co., previous to 1905, subsequently as reported by the *Eng. and Min. Journ.* (c) As officially reported except for 1908, for which year the figure of Henry R. Merton & Co. is used. (d) As reported by the *Eng. and Min. Journ.* (e) As reported by Henry R. Merton & Co. for 1900–1902, as officially reported 1903–1907, as per Henry R. Merton & Co. for 1908. (f) As officially reported (g) As officially reported, 1900–1905, subsequently as per Henry R. Merton & Co.

WORLD'S PRODUCTION OF COPPER. (a)

Year.	Metric Tons.	Short Tons.	Year.	Metric Tons.	Short Tons.	Year.	Metric Tons.	Short Tons.
1879	154,471	170,310	1889	265,516	292,741	1899	476,194	525,021
1880	156,500	172,547	1890	274,065	302,166	1900	491,435	541,561
1881	166,065	183,093	1891	280,138	308,862	1901	529,508	583,517
1882	184,620	203,550	1892	309,113	340,808	1902	542,606	597,951
1883	202,697	223,481	1893	310,704	342,562	1903	630,590	694,910
1884	223,884	246,840	1894	330,075	363,920	1904	693,240	764,758
1885	229,315	252,828	1895	339,994	374,856	1905	698,931	770,221
1886	220,669	243,295	1896	384,493	423,917	1906	715,510	788,492
1887	226,492	249,716	1897	412,818	455,147	1907	722,655	796,366
1888	262,285	281,179	1898	441,282	486,529	1908	762,111	839,846

(a) The statistics for 1879–91 are as reported by Henry R. Merton & Co.; 1892–1908 as per *The Mineral Industry*.

Arizona and Tennessee is sold directly for casting purposes. Most of the copper remelted from junk and scrap is also sold for casting. Lake copper constitutes a class by itself, but of it there are several grades, viz. (1) Prime Lake; (2) electrolytic Lake (refined electrolytically in order to extract the silver content); (3) Lake mixed with electrolytic copper (as done at the Lake Superior smeltery); (4) Arsenical Lake. Among 1, 2 and 3 there is no great distinction in the market, although certain fancy brands are apt to command a small premium from cus-

tomers who prefer them. Arsenical Lake used to sell at a small discount, but in 1908 a preferential demand for it developed, certain manufacturers of heavy trolley wire specifying an arsenic content of 0.5 per cent., the superior durability of such copper being considered to outweigh its inferior conductivity.

In the testimony in the recent suit of A. S. Bigelow against the Calumet & Hecla company, T. L. Livermore, of the latter company, made the following interesting statements:

"Of Calumet copper I can say that sometimes it has been of the same price as electrolytic and sometimes higher. When electrolytic copper did not have the good reputation for excellency and uniformity that much of it has today, it was more common to get a better price for Calumet than it is now, but today while on a rising market for copper we ordinarily command a little better price, ranging from ¼ to ½c. per lb., than is commanded by electrolytic copper, yet on a falling market we are obliged to take the same price in order to dispose of our product, and do take it.

"I think that the reputation of Calumet copper established in the early days of which I have spoken, when electrolytic copper, as a rule, was not as good as it is today, has descended to the recent period with some users and they have preferred Calumet copper on account of that actual difference which once did exist; but for the most part today, I believe from all I have learned from our customers that it is the uniformity of quality of the Calumet copper which gives it the preference over electrolytic copper where it does do so, and that as a matter of fact there is no absolute necessity of Calumet copper for the uses of consumers which gives it a preference over electrolytic. The reputation of superiority I think has been for the benefit of my company and I should be very glad to have it maintained."

Mr. Livermore was asked what degree of conductivity the Calumet & Hecla copper possessed and he replied that that portion of it which was put into wire bars had an electric conductivity of 99.5 per cent., or more; frequently more.

"I think at one time Tamarack and probably Osceola sold in competition with us and brought as high a price, but in recent years I think they have brought lower prices than ours." Mr. Livermore was questioned as to whether the product of the Copper Range mines was known to the trade as being copper carrying a high percentage of arsenic. Mr. Livermore replied that it was clearly distinguished from Calumet & Hecla copper. He said he thought it was regarded as a different and lower grade than the Calumet and the Quincy product, but as compared with Tamarack and Osceola he was not so confident. He said he was under

the impression that sometimes the product of the Copper Range mines sold as high as that of the Tamarack and Osceola and he thought that it had done so within the last year in large quantities.

ELECTROLYTIC COPPER REFINERIES OF THE UNITED STATES.
(Approximate annual capacity at ends of each year.)

Works.	Location.	1906 Capacity, Pounds.	1907 Capacity, Pounds.	1908 Capacity, Pounds.
Nichols Copper Company	Laurel Hill, N. Y.	288,000,000	300,000,000	300,000,000
Raritan Works	Perth Amboy, N. J.	288,000,000	300,000,000	300,000,000
American Smg. and Ref. Company	Perth Amboy, N. J.	132,000,000	132,000,000	144,000,000
U. S. Metals Refining Company	Chrome, N. J.	144,000,000	144,000,000	144,000,000
Baltimore Cop. Roll. and Mfg. Co.	Baltimore, Md.	130,000,000	130,000,000	130,000,000
Balbach Smelting and Refining Co.	Newark, N. J.	48,000,000	48,000,000	48,000,000
Boston and Montana Copper Co.	Great Falls, Mont.	48,000,000	48,000,000	48,000,000
Tacoma Smelting Co.	Tacoma, Wash.	18,000,000	18,000,000	18,000,000
Mountain Copper Co.	Oakland, Cal.	3,000,000	3,000,000	3,000,000
Chicago Copper Refining Co.	Blue Island, Ill.	2,000,000	2,000,000	2,000,000
Calumet & Hecla Mining Co. (a)	Buffalo, N. Y.	25,000,000	25,000,000	25,000,000
North American Lead Co.	Fredericktown, Mo.	.....	2,000,000	2,000,000

(a) Refines Lake copper.

REVIEW BY STATES.

Alaska.—The production of this Territory decreased in 1908 because it was impossible to operate profitably many of the mines on a 13c. basis. About 20 per cent. of the output in 1908 came from the district of Prince William sound; the remainder from Kasaan peninsula and Copper Mountain. The Copper River railway, building from Cordova, was extended to a point a little beyond Childs Glacier (47 miles). This railway, with steamers now in operation or under construction, will give easy access to the Chitina district in 1909. The building of a railway from Valdez into the Copper River region is said to be under consideration by a new company.

Prospecting in Chitina valley was checked in 1908 by the lower price for copper. At the Bonanza mine the season was spent in erecting a plant for developing the mine. Development work was prosecuted on what is known as the extension of the Bonanza, in the upper part of McCarthy Creek valley, by the owners of the Mother Lode claim and by the Houghton Alaska Exploration Company. The largest operations in the remainder of the field were conducted by the Alaska Consolidated Copper Company, on Nuggett creek; the Hubbard Elliott Copper Mines Development Company, on Elliott creek; and the Great Northern Development and the Alaska Kotsina Copper companies, on Kotsina river.

There was considerable prospecting during 1908 in the Nabesna river region, on Copper creek, in the vicinity of Skolai glacier, and on White river near the international boundary line. The best-known copper prospects of Nabesna river are situated near the foot of Nabesna glacier, in the

vicinity of Orange Hill. The ore deposits consist chiefly of chalcopyrite and a little bornite, with oxidation products. They are of the contact-deposit type. Development work is not yet sufficiently advanced to give an idea of their extent. There are a number of other copper prospects in the vicinity of Nabesna glacier, the copper minerals being those just mentioned or, at a few places, native copper. The copper minerals on Copper creek consist chiefly of chalcopyrite and chalcocite. Cuprite and native copper are also present in many places on the surface. The copper deposits of White river consist of copper sulphides and native copper. Native copper is found in greatest amount in the vicinity of the boundary line, where large masses resulting from oxidation of copper sulphides have been known for a long time.

Of the copper mines on Prince William sound the Bonanza (Latouche island) was in continuous operation during 1908, and the Gladhaugh mine for a part of the year. Small shipments of copper ore were made from about half a dozen other properties. The ore averaged between 7 and 8 per cent. copper. There was much activity in prospecting at Latouche and Knight islands and at Galena, Boulder, Landlocked, and Fidalgo bays. The U. S. Geological Survey expresses the opinion that when the Bering and Matanuska coking coals are made accessible by railways, many copper ores of too low grade to ship to Tacoma can be profitably reduced in a local smeltery.

Owing to the lower price for copper, operations were suspended during the winter of 1908-09 at all of the mines of the Ketchikan district except the Jumbo. The Mount Andrew and Rush & Brown mines were idle throughout 1908. Mining was resumed in August at the Mamie and Stevenstown mines, and a relatively large tonnage of ore from them was treated at the Hadley smelter. The largest production and most extensive developments during 1908 were made at the Jumbo mine, near Sulzer. The Niblack mine was operated from April 15 to Oct. 30, during which period a shaft was sunk to a depth of 350 ft. and developments were extended on the 300-ft. level, where two orebodies were opened. At the It mine, on Kasaan peninsula, mining was begun early in the spring, and some high-grade ore was developed. A tramway one mile long was built to tide water, a wharf was erected, and ore shipments were begun early in October.

Arizona.—This Territory made a largely increased production in 1908, due to gains by nearly all of the districts. The Humboldt smelter was idle throughout the year, but the company having been reorganized, operation may be resumed in 1909. A new works was put in operation by the Southern Arizona Smelting Company at Sasco, to smelt the ore

of the Imperial mine, previously treated at the Copper Queen works. Smelting capacity at the latter was thereby released.

The mining enterprises controlled by Phelps, Dodge & Co. were, on Jan. 1, 1909, consolidated into a single corporation under the name, Phelps, Dodge & Company, Incorporated. The new company has a capitalization of $50,000,000, which was used for the purchase of the following shares at the valuations stated: Copper Queen Consolidated Mining Company at $135 for each $10 share, $27,000,000; Moctezuma Copper Company at $307 9-13 for each $100 share, $8,000,000; Detroit Copper Mining Company of Arizona at $150 for each $25 share, $6,000,000; Stag Cañon Fuel Company at $800 for each $100 share, $4,000,000; to be held for future issue, $5,000,000. On Nov. 1, 1908, the investment in the plants taken over by the corporation, exclusive of the mines themselves, was $28,949,739.53.

Bisbee.—The production and net earnings of the Copper Queen Consolidated Mining Company and the Calumet & Arizona Mining Company for the last six years have been as stated in the accompanying tables.

COPPER QUEEN STATISTICS

Year.	Production, Pounds.	Net Earnings.	Dividends.
1903	37,257,470	$2,201,640.40	$ 800,000
1904	50,151,552	2,960,659.70	800,000
1905	64,625,955	5,609,486.30	2,300,000
1906	79,219,655	7,625,854.76	6,500,000
1907	63,341,055	4,471,137.08	3,800,000
1908	76,125,162	.....	3,000,000

The difference between earnings and dividends is represented by expenditure on increased plant facilities, and undistributed. Since 1887 the company has paid a total of $30,060,000 in dividends.

CALUMET & ARIZONA MINING COMPANY.

Year.	Net Earnings.	Copper Produced, lb.	Price Received.	Gold and Silver Value.	Gross Product.	Cost Per lb.
1903	$1,341,474	25,535,857	11.558c.	$144,862	$3,096,807	6.89c.
1904	1,683,518	31,638,660	12.562c.	195,926	4,170,374	7.86c.
1905	2,314,263	31,772,896	14.932c.	178,843	4,923,172	8.21c.
1906	4,827,872	37,470,284	17.96c.	238,464	6,968,127	5.71c.
1907	2,114,047	30,689,448	18.102c.	210,846	5,765,636	11.22c.
1908	857,700	28,048,329	12.948c.	234,358	3,859,854	11.00c.

Clifton.—The chief producers of this district are the Arizona Copper Company, Detroit Copper Mining Company, and Shannon Copper Company. The mines of the Detroit Company are situated in the same beds of felspathic rock which are yielding the product of the Arizona Copper Company and the Shannon Copper Company. The property of the company consists of 145 mining claims, and the ore now extracted

amounts to about 36,000 tons per month, yielding about 3 per cent. of copper. The bulk of the ore is concentrated mechanically to a grade of about 15 per cent., and smelted at Morenci in the company's own smelting works.

DETROIT COPPER MINING COMPANY.

Year.	Production.	Net Earnings.	Dividends.(a)
1903	16,869,300	$ 543,456.00	.....
1904	16,424,394	603,340.00	.....
1905	14,632,117	532,684.28	.....
1906	20,347,497	973,456.42	.....
1907	17,974,581	814,874.11	$80,000
1908	24,223,172	1,072,016.00	480,000

(a) The company has paid a total of $4,057,590 in dividends up to the end of 1908.

The increased production in 1908 was owing in great measure to improved facilities for treatment. The future production with the present equipment can be maintained at approximately two million pounds of copper per month.

At the annual meeting of the Arizona Copper Company, Feb. 24, 1909, the chairman stated that the production of copper in the last fiscal year was 16,973 tons, against 13,704 in the previous year. The cost of production in the first semester of 1907-08 was £42 8s. per ton; in the second, £36 10s.; average for year, £39 9s. The average yield of all the ore was 2.3 per cent. The second-class ore yielded 1.995 per cent. in the first half and 2.143 per cent. in the second. The first class ore yielded 6.64 per cent. as average for the whole year. Total ore mined was 751,220 tons. Profit for year was £288,138. An immense amount of low-grade ore has been developed. New flue-dust chambers were put in use in August, 1908, and these are recovering a greatly increased quantity of flue dust, which now totals 90 tons per day. This is being briquetted for resmelting, wherefrom important economy has resulted.

Courtland. (By H. W. Chittenden.)—This new camp is situated in Cochise county, 35 miles north of Bisbee, and nine miles south of Pearce. The rock series consists of quartzite, overlaid by limestone. The total thickness of the limestone does not appear to be more than 2000 ft. The whole series has been tilted to a fairly uniform, steep, northeast dip, and many large faults have broken the formation into rather small fault-blocks, duplicating the sedimentary series and greatly increasing its apparent thickness. Into these sedimentary rocks have been intruded a rather coarse-grained diorite and a quartz porphyry.

The ore occurs along zones of fracture in the limestone and quartzite, replacing the rock between the fractures. In all places so far discovered

the ore has been closely associated with the igneous rocks. The orebodies are usually marked on the surface by large outcrops of gossan, often copper stained. The primary ore, as shown in the deeper workings in the western part of the camp, is a mixture of iron and copper sulphides and a silicious gangue. Except for small rich streaks, the primary ore has been of very low grade. Secondary alteration, however, has resulted in great enrichment and in the eastern part of the camp the bottom of the enriched ore has not yet been reached. The secondary ore consists of malachite, azurite, cuprite, and other oxidized copper minerals, in a gangue of iron, manganese, and aluminum oxides, often very silicious. The oxidized ore is often of very high grade. Surrounding the ore are large masses of nearly barren silicious iron and aluminum oxides, passing into the unaltered country rocks.

The Copper Queen, Calumet & Arizona, and Great Western companies are each operating two shafts, and a few smaller prospecting shafts are also in operation. The Copper Queen has considerable ore developed about the Casey shaft, and is sinking the Silverton shaft to open the eastern part of the camp. The Calumet & Arizona has large orebodies blocked out at the Germania shaft, and is sinking the April Fool shaft to develop the northern part of the camp. The Great Western company has fine orebodies well opened at the Mary shaft, and also has much ore near the surface at the Humboldt shaft. At the Leadville shaft, belonging to an independent operator, some sulphide ore of good grade has been encountered.

A company has been incorporated to build a railway connecting Courtland with Douglas. This road, which is to be called the Colorado & Mexico railroad, will be controlled by the Phelps-Dodge interests, and operated in connection with the El Paso & Southwestern. The length of the line with its spurs will be about 45 miles. Surveys are already well under way, and construction will probably be started very soon. Ore can be shipped from several of the shafts as soon as the railroad is finished.

Globe.—The chief producers of this district in 1908 were the Old Dominion, United Globe, and Arizona Commercial. The last is planning to build a smeltery. The most prominent feature of the year in this district was the remarkable development of the mine of the Miami Copper Company. At the end of the year this company had developed about 12,000,000 tons of ore that is reckoned as averaging 2.8 per cent. copper and is expected to yield 40 lb. per ton. On the 470-ft. level the orebody is opened for 1300 ft. in length and 800 ft. in width. The dimensions of the orebody on the 570-ft. level (at present the lowest) are expected to be as large. The full magnitude of this great orebody

has not yet by any means been disclosed. The ore is of the disseminated character. The construction of a dressing works will be begun in 1909. This will be of 2000 tons daily capacity. It is expected that the ore will concentrate from 12:1 to 16:1, with extraction of upward of 70 per cent., and that the annual production will be about 25,000,000 lb., at cost of 9c. per lb., New York.

Jerome.—In this district the United Verde Copper Company is the only producer of consequence. According to the *Boston News Bureau* the results of this company (which is a close corporation) have been as given in the accompanying table:

STATISTICS OF UNITED VERDE COPPER COMPANY.

	1906	1907	1908
Copper production, lb	.....	33,012,339	36,183,089
Silver production, oz	.....	.....	494,574
Gold production, oz	.....	11,730	20,334
Cost of mining	.....	$1,068,293	$ 796,529
Cost of smelting	.....	1,281,674	1,215,993
Charged to depreciation	.....	298,243	320,407
Price received for copper, per lb	20.22c.	18.167c.	12.343c.
Price received for silver, per oz	67.435c.	64.158c.	52.453c.
Price received for gold, per oz	$20.50	$20.50	$20.50
Cost of copper, per lb (a)	8.69c.	10.54c.	9.046c.

(a) Value of the precious metals is not credited against cost of producing the copper.

At the end of 1908 the company had 9,543,237 lb. of copper on hand.

Kelvin.—Extensive developments were made in this district in 1908 by the Ray Consolidated and Kelvin-Calumet companies. The latter is being taken over by a new concern called the Ray Central. In this district there are very large deposits of disseminated ore, lying more or less flat, that are thought to assay about 2 per cent. copper. The Ray Consolidated plans to build a 3000-ton mill in 1909. This company was promoted by interests identified with the Utah Copper Company. The Ray Central is a Lewisohn company; its plans are not yet far advanced.

(By James Douglas.)—The financial gloom with which 1908 opened did not acutely affect the large copper mines of Arizona, if we may judge by their activity. The United Verde ran continuously and produced its normal output. In the south the Warren district increased rather than diminished its contribution, and the Clifton witnessed no falling off.

Arizona's neighbor, Sonora, suffered severely. The Greene-Cananea company, which shut down in October of 1907, resumed work in September of 1908, under the management of Dr. L. D. Ricketts. The concentrating plant, which had been built from designs by Doctor Ricketts during the previous administration, underwent no change. But the furnace and converting plants were completely rebuilt on the old site;

the reverberatory was remodeled advantageously, in both its smelting and steam-generating qualities; the bedding and conveying devices, which had been planned by Arthur S. Dwight and the Robins company, were put into working order, and justified the economical results which were claimed for them. Notable saving in cost of smelting has resulted, and all and more than was expected has been realized from the introduction of the caving system in the mines. This great property has resumed work with every probability of attaining success, despite the enormous capital with which it is overburdened. As C. F. Shelby has, in *Eng. and Min. Journ.*, described with much candor and fullness of detail most of the improvements introduced in his smelting department, outside testimony would be superfluous. Such success has attended reverberatory operation in a furnace 100 ft. long by 19 ft. wide, that a second furnace will probably be built and the present furnace enlarged. To the reverberatory is passed the flue dust and fine concentrates, partly calcined and partly raw. The cost of reverberatory smelting with oil as fuel, giving the furnace credit for the waste heat used in generating steam, is said not to exceed $1.30. In the construction and operation of the reverberatory furnace some novel features have been adopted, such as protecting the walls by introducing highly silicious ore through a number of openings in the roof.

At Nacozari the old mill has been put out of commission, and the new mill of 2000-tons capacity is treating 1500 tons of Pilares ore a day. The concentrates are still shipped for metallurgical treatment to Douglas, and yield in copper about 2,000,000 lb. a month. The economy obtained at Cananea in the reverberatory smelting, with oil as fuel, of concentrates and flue dust, has revived the discussion of the wisdom of resuming smelting at Nacozari. Besides discarding the old concentrator, its same fate has attended the gas plant and gas engines, which have been supplanted by a large power plant in which Curtis engines are used to operate the generators. The wood gas and gas engines were not rejected on account of dissatisfaction with cost, but the size of the new transmission plant required for moving the new mill, and supplying the necessary power to the mines would have demanded a cumbersome equipment of gas engines, and exhausted rapidly the available wood supply. Accurate estimates of cost of a horsepower under the two systems have not yet been made.

At Bisbee the ore from every section of the Copper Queen mine except the Lowell is now being hoisted by a skip through the new Sacramento shaft. By a travelling belt and tripper it is discharged into a train of railroad cars, the object being to secure a uniform mixture. In the central power house, situated near the Sacramento shaft, power is

derived from Curtis turbines, and is distributed for underground haulage by electric current and by compressed air to the engines of the old shafts, which will be used only for the hoisting of timber and of men.

At Douglas the Calumet & Arizona works were not only enlarged, but also rearranged to meet the increased demands made on them by the production of the Superior & Pittsburg mines, as the Calumet & Arizona and Superior & Pittsburg companies hoist as much ore as the Copper Queen. At the Copper Queen reduction works at Douglas the shipment of about 300 tons of concentrates a day from Nacozari, necessitates a modification of both method and plant. Experiments are under way to determine whether sintering or reverberatory smelting will be used as a supplement to the blast-furnace practice.

Nothing of note occurred in the Clifton district, but the output of all the three smelting companies showed an increase. In the Warren district the Copper Queen produced less than in 1907, and the joint mines of the Calumet & Arizona and Superior & Pittsburg much more than in 1907. The result of the mining in the Warren and Clifton districts showed an increase for 1908 over 1907 of approximately two and a half million pounds. The Imperial company erected a second furnace and contemplates the building of a third. Saddle Mountain, Twin Butte, Helvetia and most of the smaller mines, did not resume operations, or worked in a very languid way. The Old Dominion furnaces exceeded their 1907 output by about 940,729 lb., and of the total output about 30,560,832 lb. came from the company's own mines. The development of the Miami mine and the success in floating that company gave a great impulse to both mining and smelting in that territory; but last year none of the companies except the Old Dominion was a large producer. The closure of the smelter on the Agua Fria continues to impede general copper mining in Yavapai county.

During 1908 a new district loomed up as a possible large producer. As far back as the prosperous Tombstone days the Turquoise district, situated in the foothills of the Dragoons, where they sink into the Sulphur Spring valley, attracted much attention on account of its silver mines. They were said to have been abandoned on account of the refractory character of the ore and the presence of copper. Nearby there has, of late, been opened in what is known as the Courtland district, by the Calumet & Arizona company and by the Great Western Mining Company, a most promising bed of copper carbonate in a manganese gangue. The deposit is in limestone cut by intrusive rocks, and the geological conditions, as well as the actual developments, promise well. A line of railroad—the Arizona & Colorado—is being built from Douglas to Courtland, which will probably be completed by June, 1909, when some of the ores

from that district will reach the Douglas smelters. The year, 1909, will determine whether another notable group of mines will add to the already large production of the Territory. If so, Arizona will maintain its supremacy over Montana and the Lake, for all of the running mines unless overtaken by accidents or unfavorable general conditions, will maintain to the full their present output. The low-grade mines of Gila and Pinal counties, notably the Miami and the Ray group, cannot contribute much till the concentrators now designed or under construction, are in operation.

Under the tax law passed by the legislature of Arizona in 1907, the gross product of the mines and mining claims is taxed, and, therefore, the tax statement gives besides the quantity, the actual gross value of the copper, gold and silver extracted from the minerals mined in the Territory. The law was passed in 1907, but was effective for taxes due for 1906. The returns are, therefore, available for 1906 and 1907. The number of companies taxed in 1906 was 62, which was increased to 80 in 1907. The total quantity and value assessed in 1906 was: Copper, 255,012,155 lb.; gold, 125,016 oz.; silver, 2,704,044 oz., with a total gross value of $53,801,781. In 1907 the figures were: Copper, 253,784,698 lb.; gold, 118,374 oz.; silver, 2,423,723 oz., with a total gross value of $54,788,674. In both years the Copper Queen heads the list of gross value of product with $14,236,428 in 1906, and $13,699,107 in 1907. In 1906 and 1907 the United Verde comes next with $8,038,126 and $7,060,126. The Calumet & Arizona follows with $7,337,748 in 1906, and $6,226,664 in 1907. The Arizona Copper Company occupied the same position in both years with $4,973,400 in 1906 and $6,160,050 in 1907. In 1906 there are only three other companies, all copper companies, which produced more than $1,000,000, whereas in 1907 there were five.

The most equitable system of taxing mines is a question upon which there is great difference of opinion. As the rate is assumed to be adjusted to the total value of the property taxed, the total value of the product in metal or mineral of mines, above a given minimum, is perhaps open to less ambiguity than a valuable product based on net returns, as is the system applied in Montana. The Butte mines in 1907 were taxed on the following net returns: Anaconda (1-1[1]), $5,456,395; Anaconda (1-B), $362,790; Butte & Boston (1-1), $1,239,055; Butte & Boston (1-B), $10,412; Boston & Montana (1-1), $7,049,988; North Butte (1-1), $3,271,214; Parrot Silver (1-1), $14,300; Parrot Silver (1-B), $127,431; Red Metal (1-1), $1,432,761; Trenton (1-1), $481,624; Original Consolidated (1-B), $663,260; Monidah Trust (1-B), $3,053; La France (1-W), $151,013; Clark-Montana Realty (1-B), $188; total net proceeds,

[1] 1-1 means outside city limits, levy 18¼ mills; 1-B means inside city limits, levy 29⅞ mills; 1-W means inside Walkerville city limits, levy 28¼ mills.

$20,263,482. Deductions may be drawn from the tables as to cost of production, but they will be of doubtful accuracy.

California.—The production of this State increased materially in 1908 owing to the more extensive operations in Shasta county. The Mammoth Copper Mining Company was the largest producer. This company having installed a converter plant is no longer obliged to ship its matte to Salt Lake, but on the contrary is now in a position to buy matte from other smelters. In 1908, the product of several concerns in California was converted at the Mammoth works. The Bully Hill company, having completed its railway, put its smeltery in operation in 1908, but shipped its whole product as matte. The Balaklala company also put its smeltery in operation, but up to the end of 1908 had not converted any of the matte, wherefore its output does not figure in the statistics. With this new plant in operation, the outlook is of course for a largely increased production from California in 1909. More or less work was done in copper mining in other countries, but there were no features of particular interest, unless it be the resumption of mining in the Low Divide district in Del Norte county after more than 40 years of idleness.

The Balaklala Consolidated Copper Company in 1908 completed its smeltery of 1250 tons daily capacity. Up to date mine construction has amounted to $292,899; smelter construction, $873,682; converter plant, $102,512; railway, $83,279. The mine was operated for 60 days in 1908 and the smeltery for 52 days. Indications are that the ore developed will average about 2.7 per cent. copper, 0.025 oz. gold and 0.75 oz. silver per ton. The ore mined was 18,751 tons at an average cost of $2.42 per ton. When operations are in full swing it is believed that the cost of mining will be about $2 per ton. The smeltery comprises four 18-ft. McDougal furnaces, three 55x240 in. blast furnaces, one 17x92 ft. reverberatory furnace, and two converter stands. The cost of smelting 25,120 tons of ore was $2.67 per ton. It is expected that a cost of about $2.50 per ton of ore reduced to matte will be attained and about $15 per ton of copper converted.

Colorado.—In spite of the lower price for copper, this State made an increased production in 1908, which was due partly to the Leadville district and partly to some new discoveries in the San Juan, especially at the San Antonio mine in the Red Mountain district and at the Frank Hough mine near the dividing line between Hinsdale and Ouray counties.

Idaho.—As in previous years, the bulk of the copper produced in this State in 1908 came from the Snowstorm mine, in the Coeur d'Alene, which is the only important copper mine that has yet been opened in Idaho. It was worked continuously in 1908, the ore, which is highly silicious, being shipped to smelters in other States, and in British Colum-

bia, chiefly for use as converter lining. About 10,000 tons of ore, averaging about 4½ per cent. copper and 6 oz. silver per ton were shipped, and the company paid dividends at the rate of $45,000 per month. The White Knob mine, in Custer county, was idle during the greater part of 1908. At Loon Creek, the Lost Packer company successfully operated its 100-ton smelter for 90 days during the summer, producing some high-grade matte. Developments in the Seven Devils district were of no particular importance.

Michigan.—Some of the mines at Lake Superior made slightly diminished outputs in 1908, but others increased. The net result was a total but little different from that of 1907. The data of the production are given in the accompanying tables.

COPPER PRODUCTION IN MICHIGAN.
(Pounds of fine copper.)

Mines.	1901	1902	1903	1904	1905	1906	1907	1908
Adventure	.....	606,211	2,182,608	1,380,480	1,606,208	1,552,628	1,244,872	90,870
Ahmeek	.....	.....	.....	350,000	1,552,957	3,077,507	5,527,672	6,280,241
Alloues	.....	.....	.....	.....	1,167,957	3,486,900	2,934,116	3,047,051
Atlantic	4,666,889	4,949,368	5,505,598	5,321,859	4,049,731	1,439,082	*Nil.*	*Nil.*
Baltic	2,641,432	6,284,819	10,580,997	12,177,729	14,384,684	14,397,557	16,704,868	17,724,854
Cal. & Hecla	82,519,676	81,248,739	76,490,869	80,341,019	83,812,370	94,529,821	88,055,723	81,660,723
Centennial	806,400	.....	.....	641,294	1,446,584	2,253,015	2,373,572	2,196,377
Champion	.....	4,165,784	10,564,147	12,212,954	15,707,427	16,954,986	16,489,436	17,786,763
Franklin	3,757,419	5,259,140	5,309,030	4,771,050	4,206,085	4,571,570	4,401,248	3,703,421
Isle Royale	2,171,955	3,569,748	3,134,601	2,442,905	2,973,761	2,937,098	2,667,608	3,011,664
Mass	837,277	2,345,805	2,576,447	2,182,931	2,007,950	2,106,739	2,078,677	1,766,930
Michigan	.....	166,898	275,708	2,746,127	2,891,796	2,875,341	2,665,404	3,000,206
Mohawk	160,897	226,824	6,284,327	8,149,515	9,387,614	9,352,252	10,107,266	10,295,881
Osceola	13,723,571	13,416,398	16,059,636	20,472,439	18,938,965	18,588,451	14,134,753	21,250,794
Phoenix	93,643	.....	202,823	1,162,201	273,219	*Nil.*	*Nil.*	*Nil.*
Quincy	20,540,740	18,988,491	18,408,288	18,343,160	18,827,557	16,194,940	19,796,058	20,600,361
Tamarack	18,000,852	15,961,528	15,286,093	14,961,885	15,824,008	9,832,644	11,078,604	12,806,127
Trimountain	.....	5,730,807	9,237,051	10,211,230	10,476,462	9,507,933	8,190,711	6,034,908
Winona	.....	101,188	1,036,944	646,025	*Nil.*	278,182	1,285,863	*Nil.*
Wolverine	4,946,126	6,473,181	8,999,318	9,764,455	9,464,418	9,548,123	9,273,351	9,555,233
Victoria	.....	.....	.....	.....	.....	.....	1,207,237	1,290,040
Others	640,591	700,067	75,000	50,000	.....	.....	100,000	.....
Totals	155,507,465	170,194,996	192,299,485	208,392,485	218,999,753	224,071,103	220,317,041	222,267,444

DIVIDENDS PAID BY MICHIGAN MINES.

Mine.	1906	1907	1908	Mine.	1906	1907	1908
Calumet & Hecla	$6,500,000	$6,500,000	$1,950,000	Quincy	$1,250,000	$1,485,000	$504,554
Copper Range Con.	2,600,000	2,000,000	1,205,495	Tamarack	480,000	420,000	.....
Mohawk	700,000	900,000	250,000	Wolverine	660,000	1,050,000	600,000
Osceola	1,541,400	1,249,950	192,300	Totals	$13,231,000	$13,604,950	$4,702,349

Year.	Dividends.	Year.	Dividends.	Year.	Dividends.	Year.	Dividends.	Year.	Dividends.
1850	$84,000	1870	$700,000	1890	$3,415,000	1902	$3,440,000	1906	$13,231,000
1855	168,000	1875	1,920,000	1895	3,280,000	1903	4,980,000	1907	13,604,950
1860	120,000	1880	3,080,000	1900	9,811,200	1904	5,432,300	1908	4,702,349
1865	510,000	1885	1,970,000	1901	7,496,900	1905	9,224,600		

(By C. L. C. Fichtel.)—The Keweenaw Copper Company became a producer in 1908; at the end of the year its mill was handling about 120 tons of ore daily. The Keweenaw Central Railroad was completed and maintained regular service between Calumet and Mandan. The Tamarack company shipped about 80 tons per week from the old Cliff property. The Ojibway company made good progress in development. Exploration was begun on the Seneca tract. Developments were also made upon the Gratiot property, owned by the Calumet & Hecla. The five shafts of the Mohawk company were sunk deeper. Ahmeek began the sinking of two new shafts to work the northern and central portion of its tract. Allouez deepened its shafts. At the stamp mill owned jointly by the Centennial and Allouez two new heads were added. Calumet & Hecla made many improvements. All the shafts on the conglomerate lode were operated, but the copper content of the ore gradually decreased and the ground available from some of the shafts is becoming exhausted. The Osceola amygdaloid formation was opened extensively, all of the six shafts being sunk deeper. One shaft on the Kearsarge lode was operated. Tamarack has discontinued sinking its shafts vertically; henceforth all additional depth will be gained by inclines on the lode. The north and south Kearsarge branches of the Osceola company developed in good shape. La Salle, a subsidiary of the Calumet & Hecla, was developed during the year. The Tecumseh shaft showed some good copper rock. Quincy started a new shaft, known as the Pontiac, or No. 9. The Hancock company sank its shaft below the 15th level, and at the 13th level started a crosscut to connect with No. 1 shaft. At the latter, the crosscut opened the west lode and found it more uniform and of better grade than the Hancock lode; this will be the main working lode hereafter. Isle Royale developed the extension of the Baltic lode. The Superior property is on the verge of becoming a producer and is considered to have as good prospects as any mine at Lake Superior. Atlantic continued developments at its section 16 tract, from which some rock sorted out at the end of the year was put through the mill. The Champion and Baltic mines of the Copper Range company developed in good shape, and the Trimountain is gradually approaching the other two in production. In Ontonagon county developments were made by the Elm River, Wyandot, Winona, King Philip, Adventure, Mass Consolidated, Michigan, and Victoria companies. The remarkable showing, opened close to the surface, at the Lake property has continued with depth. The Adventure in diamond drilling cut an encouraging amygdaloid formation. Explorations by the Victoria company were rather disappointing. The Michigan resumed the

construction of its mills and encountered a much better showing on the Calico lode.

Missouri.—The North American Lead Company, of Fredericktown, continued to be the only producer of consequence in this State. Its production was about the same as in 1907.

Montana.—During January and February the Anaconda smeltery was idle, but on March 1 operations were resumed. In announcing the resumption, John D. Ryan, managing director of the Amalgamated, stated plainly that the miners at Butte would have to be more efficient. The necessity for adjusting this matter was of course the prime reason for the closure, the cost of producing copper at Butte having risen to 12c. per lb., or more. After the resumption of work, the cost of production was reduced to the neighborhood of 10c. per lb., or practically the figure of 1905 and earlier. During the 10 months of actual operation in 1908 the Anaconda works produced copper at a great rate, wherefore the large increase in the output of Montana.

North Butte in 1908 produced 40,276,430 lb. of copper at a cost of about 9c. per lb. The cost was increased by the practical shut-down in January and February. If it had not been for that, the output for the year would doubtless have been 48,000,000 lb., the shipments during many months of the remainder of the year having been in excess of 4,000,000 lb.

Butte Coalition produced nothing in January and February and but little in March. During the remaining nine months it operated at the regular rate, producing about 2,300,000 lb. of copper per month, and making an output of 21,400,580 lb. for the year, at a cost of about 10½c. per pound.

REPORT OF BUTTE MINES FOR 1907 TO COUNTY ASSESSOR.

	Anaconda.	Boston & Montana.	Butte & Boston.	North Butte.
Tons of ore extracted	976,685	824,420	193,830	284,003
Gross proceeds	$9,775,118.65	$7,511,479.99	$1,609,763.66	$3,224,763.01
Cost of mining	3,905,222.28	3,262,266.55	639,379.71	990,405.55
Freight on ore	733,352.81	101,455.75	39,472.92	55,305.36
Cost of reduction	2,957,965.15	2,225,884.17	444,020.91	1,065,404.11
Cost of marketing	617,513.18	788,988.27	191,285.30	(a)
Total cost	8,214,053.42	6,378,604.74	1,314,158.84	2,111,115.11
Net earnings	1,561,065.23	1,132,875.25	295,604.84	1,113,647.99

	Washoe.	Red Metal.	Trenton.	Parrot.
Tons of ore extracted	115,620	234,960	85,260	57,964
Gross proceeds	$1,052,956.73	$2,080,249.34	$646,109.31	$457,479.86
Cost of mining	487,268.56	1,070,272.97	308,987.68	230,593.97
Freight on ore	14,602.81	53,628.53	10,318.42	8,125.93
Cost of reduction	221,881.11	884,537.21	203,843.39	156,416.42
Cost of marketing	83,028.19	(a)	60,700.48	51,714.88
Total cost	806,860.67	1,908,438.71	574,939.97	596,851.20
Net earnings	246,096.06	71,810.63	67,259.34	.....

(a) These companies sell their ore to the Washoe smelter.

(By H. K. Welch.)—The Anaconda and St. Lawrence mines were hampered to a certain extent by the fires smouldering in the old stopes. The High Ore, 2800 ft. deep, is now the deepest mine of Butte. The deepening of the Belmont shaft so as to connect it with the lower levels of the Anaconda will save a haul of nearly 400 ft. in hoisting the ore of the latter mine. At the Tramway mine of the Butte Coalition Company, a new headframe and hoisting engine were installed, and new machine shops and other surface equipment were erected. A great deal of development work was done in the Minnie Healy ground on the 1400-, 1500- and 1600-ft. levels. Several connections with the Rarus mine were made.

The Boston & Montana operated the Mountain View, Leonard, Pennsylvania, and East and West Colusa mines. At the West Colusa work was somewhat retarded by the fire that has been burning for more than a year. At the Leonard a new shaft was sunk, and the daily output of this mine during the latter part of 1908 was greater than ever before.

The Butte & Boston operated the Silver Bow, Berkeley, and East and West Greyrock mines. The Parrot company worked the Parrot and Little Mina mines. The showing in the Parrot mine was rather poor, but in the Little Mina several bodies of good ore were opened. The Trenton Mining Company, operating the Gagnon mine, obtained better ore than for several years previous.

The Original and West Stewart mines, belonging to W. A. Clark, were operated throughout 1908. The ore in these mines has improved with increased depth. The North Butte company sank its shaft from the 1800- to the 2200-ft. level. On the 2000-ft. and 2200-ft. levels crosscuts were driven to the veins which were found to be fully as good as higher. The mine made an average output of 1400 to 1500 tons of ore per day. The Pittsburg & Montana Company abandoned the smelting experiments that had previously engaged its attention, and under the management of Oscar Rohn, built a new 250-ton concentrating mill, remodeled the smeltery to conform to standard lines, and went upon what is believed to be a paying basis. The La France Copper Company operated the Lexington mines and opened some good bodies of copper ore. The Davis-Daly company underwent a complete reorganization, but continued its development work practically without interruption. The Tuolumne company developed a body of commercial ore and promises to become a regular producer. The other companies at Butte carried on prospecting and development work, which did not materialize in any result of particular importance.

At Butte about 10,000 men were employed steadily, and upward of 16,000 tons of ore were produced daily, of which the Amalgamated mines contributed about 12,500 tons, the North Butte, 1450; W. A. Clark, 1100;

Butte Coalition, 1100; Pittsburg & Montana, 350; and others, 300 tons.

In the Corbin district, in Jefferson county, the Boston & Corbin company attained a depth of 500 ft. and opened several bodies of commercial ore. Development work was also done upon the Black Jack, Corbin-Wicks, Corbin Consolidated and Alta. The property of the Montana Consolidated Mining Company, near Basin, was operated steadily. The formation in the Corbin district is somewhat similar to that of Butte, the granite being coarse-grained and micaceous, while the veins resemble those of Butte, but sulphide ore is found nearer the surface. The Monitor Mining Company, in Missoula county, opened some good ore, of chalcopyrite character, of which several shipments were made. The Copperopolis mine, in Meagher county, was worked by leasers who made several shipments of high-grade ore.

Nevada.—The large increase in the copper production of this State in 1908 was, of course, but a forerunner of the further large increase that is to be expected in 1909, being due to the beginning of operations at the Steptoe Valley smeltery. This smeltery is owned jointly by the Nevada Consolidated Copper Company and the Cumberland-Ely Copper Company. Developments in the mines of both these companies have been highly satisfactory. The Giroux company, owning the other important mines in the Ely district, has not yet become a producer to any considerable extent, owing to difficulties of various kinds. Late in 1908 the controlling interest in this company was purchased by Cole, Ryan & Co., and plans were formulated for more active operations, adequate financing of the same being arranged.

The report of the Nevada Consolidated Copper Company for the year ended Sept. 30, 1908, covers a period largely devoted to development and preparation for future production. James Phillips, Jr., president, said:

"For the final four months of this fiscal year the company has been, upon a constantly increasing scale, an actual producer of copper. It may be accepted with confidence that in January, 1909, our monthly production will be at a rate equivalent to an annual production of at least 35,000,000 lb., and that by April, 1909, when the additions to the concentrating and smelting plant are completed, the increased facilities will result in raising the annual output to 50,000,000 or 55,000,000 lb. of copper. Despite the financial panic and the delays incidental to prosecuting construction during two winters, just 18 months elapsed between the turning of the first spadeful of earth on the smelter site and the beginning of the concentration and the reduction of ores in the first section of the plant, and at this time the treatment of ore and the production of copper are being carried on in five thoroughly equipped sections of the works, three of which are treating Nevada Consolidated ore. It is confi-

dently expected, since we possess the ore to warrant it, that the construction of two additional units will raise the annual production of blister copper to approximately 70,000,000 lb. yearly. The works have now been in operation a sufficient time to assure us that when the full measure of mining and smelting capacity is reached, the Nevada Consolidated will produce copper as cheaply as any copper company. The steam shovels at Copper Flat have demonstrated the great economy of this method of mining."

Following is an abstract of the report of Pope Yeatman, consulting engineer of Nevada Consolidated, presented at the recent annual meeting:

"Prospecting has been done on the Eureka section and over 5,500,000 tons of an average grade have been developed. This increases the amount of ore blocked out from 14,432,962 tons to fully 20,000,000 tons. The possibilities for a large increase of tonnage are very great; not only through a considerable extension of the present developed orebodies and partially prospected areas, but also in large areas not yet touched, which give similar indications of promise to those shown in the developed area. The prospecting carried on has been of such a nature also as to greatly increase what might be called "possible ore reserves." Drilling is now being continued and further ore is being rapidly developed. In the Ruth section, one drill hole, in ore, is so placed as to prove a continuation of the rich Ruth orebody for a considerable distance. Drilling is to be continued on this ground also.

"Work with the steam shovels was resumed on the stripping of the overburden the latter part of March and has been continued since. In the latter part of May, regular operations were commenced in mining the Eureka deposit with the steam shovel, and enough work has since been done to show the value of this method of working the Eureka deposit. Up to Nov. 1, 185,277 tons of sulphide ore have been mined. The total yardage up to Nov. 1 for stripping is 427,244.

"During the latter part of 1907 work on erection was considerably curtailed, but the force was increased in January, 1908, and very materially so in April of the same year, since which time construction has been carried on vigorously. The concentrator, designed for the treatment of 4000 tons of ore per day, is now completed with the exception of the sixth section, which should be placed in commission in December. The capacity of the plant has been proved to be equal to or greater than originally estimated, and the results obtained, though capable of improvement, may be considered satisfactory. The plant is now being operated with two sections on Cumberland-Ely ore and three sections on Nevada

Consolidated ore. The sixth section will be used for Nevada Consolidated ore.

"Ten McDougall roasters have been erected and four are now under construction. Two reverberatories have been built and both have been proved to be satisfactory. The output has been large, and it has been found that no special coals are required, but successful operations have been carried on with the ordinary coals of Wyoming and Utah. Natural draft is sufficient. The first furnace was charged on June 22. The stacks and flues have been tried out and have proved eminently successful. The blast furnace is now under construction and should be completed by the end of the year. The converter plant of three stands was blown in July 9, and since then all three stands have been in operation at different times.

"Owing to the increased development of ore the board has authorized the increase of the plant for the treatment of Nevada Consolidated ore. Additions are now under way and should be completed by April next. The additions consist of increasing two units of the concentrator by 15 to 20 per cent. and adding a fourth unit with a capacity of 15 to 20 per cent. more than the old single units; two McDougall roasters; one reverberatory furnace; one stand of converters. It is now recommended that still another unit be added to the concentrator for the handling of Nevada Consolidated ores."

The Cumberland-Ely Copper Company, according to its official report, up to Feb. 1, 1909, produced 198,611 tons of ore, of which 177,589 tons were milled, showing an average copper content of 3.1 per cent. The average extraction by the mill was 71.19 per cent.; in December, 1908, it was 73.33 per cent. and in January, 1909, it was 76.12 per cent.

New Hampshire.—A small amount of copper was produced by the Milan Mining and Milling Company, of West Milan, which mines a deposit of pyrites averaging about 40 per cent. sulphur that is sold as a sulphur ore. Some of this pyrites is copper-bearing, several carloads having shown the following analysis: Copper, 8 per cent.; silica, 7.8; sulphur, 38.18; iron, 31.83; gold, 0.08 oz. and silver, 4 oz. per ton. The company expects in a short time to have a new ore-dressing plant in operation.

New Jersey. (By Henry B. Kümmel.)—Explorations have been conducted at the mines along the base of First Mountain, three miles north of Somerville, where the ore occurs in a sheet of altered shale (1 to 3 ft. thick), which contains about 2 per cent. of native copper. A concentrating mill of 350 tons daily capacity has been erected, together with a small reverberatory furnace. For the first time conditions seem to be favorable for a thorough test of these New Jersey copper ores.

New Mexico. (By Reinold V. Smith.)—Although the production of copper during 1908 fell short of that for 1907, due entirely to the drop in the price of metal, nevertheless copper has become a decidedly important factor in the mineral industry of this Territory. Many millions of dollars' worth of development has been done in the last two years, and the total reserves of copper thus opened have become very large. The most extensive development was in the southwestern and central western parts of the Territory, especially in the Burro mountains, the Organ mountains, the Mogollons and the Magdalenas. But in many other localities also development was carried on in a systematic manner. Rotary and churn drills were used in several localities, especially in the Grant county properties, and have been instrumental in developing the large deposits of the Burro mountains and elsewhere.

Geographically the copper ores are widely distributed over the Territory, commercial bodies of ore being found in 16 counties out of the total of 24. Some of these deposits have long been known, as those at Santa Rita, which are among the oldest worked copper deposits in the United States.

The production of the Territory in 1908 came principally from the camps of Grant and Socorro counties with smaller amounts from Santa Fé, Dona Ana, Luna and Sierra counties. Among the largest producers may be mentioned the Savannah Copper Company, which has taken over and consolidated the properties of the Comanche Copper Company with its smelter at Silver City and its railroad to the mines, and the Copper Gulf properties; the Chemung Copper Company operating with a large force of men at Tyron; the Burro Mountain Copper Company at Leopold where the big concentrator is kept busy on 3 to 4 per cent. ores; and the Azure Mining Company, all of Grant county.

The copper ores in the Tertiary, known as the "Red Beds," occur in many places, and were again during 1908 the subject of investigation on the part of several companies. Their form is usually carbonate and oxide, but frequently metallic, and their tenor in copper is nearly 2 per cent., and in places approaches 3 per cent. The problem of their reduction seems near solution at the present time.

North Carolina. (By Richard Eames, Jr.)—Copper ore has been found and worked at more than 250 places in this State. The important deposits are in Davidson, Ashe, Rowan, Cabarrus, Granville, Guilford and Mecklenburg counties. Considered collectively, the ore is a sulphide carrying gold and silver. Generally considered it is of low grade, but copper mining would assume important proportions by the erection of a plant to make acid, using this acid in copper extraction and for the manufacture of fertilizers.

The copper product for 1908 was the smallest in years; the Gold Hill district, one of the principal shippers, being comparatively shut down. Several car-loads were shipped from the Virgilina district, and trial car-loads from scattered localities. The copper ore mined in 1908 was not over 500 tons. Some of this was concentrated for shipment and some shipped in lump form. Several new enterprises are starting up in 1909, and one new locality reports a fair quantity of average ore for shipment.

Pennsylvania.—A small amount of copper was produced in the form of concentrates at Lebanon in this State.

Tennessee.—The producers in this State were the Tennessee Copper Company and the Ducktown Copper, Iron and Sulphur Company. The former produces blister copper. The latter ships its matte to Chrome, N. J., for converting. The Tennessee Copper Company perfected its

STATISTICS OF TENNESSEE COPPER COMPANY.

Items.	1906		1907	
	Per Ton Ore.	Per lb. Copper.	Per Ton Ore.	Per lb. Copper
Mines development	$0.1067	0.343c.	$0.1318	0.407c.
Mining, hoisting, etc.	0.7817	2.512c.	0.9389	2.904c.
Crushing and sorting	0.0693	0.227c.	0.0804	0.249c.
Railway	0.1389	0.438c.	0.1329	0.411c.
Blast furnace	1.4864	4.765c.	1.6219	5.016c.
Engineering and laboratory	0.0370	0.118c.	0.0628	0.194c.
General	0.1387	0.445c.	0.1703	0.526c.
Converting	0.2733	0.876c.	0.2402	0.743c.
	$3.0320	9.724c.	$3.3792	10.450c.
Adjustment of ore account	0.0013	0.004c.	0.0045	0.014c.
Cost of fine copper in pig	$3.0307	9.720c	$3.3747	10.463c.
Freight, insurance and selling		0.68c.		0.68c.
Taxes and all other expenses		0.51c.		0.67c.
Total cost per lb. copper		10.91c.		11.79c.

process of making sulphuric acid from the gases delivered by its smelting furnaces and contemplates the production of 300,000 tons of tower acid per annum. The Ducktown company is installing a similar plant on a smaller scale. The Tennessee Copper Company made a contract with a newly organized concern, called the Independent Fertilizer Company, whereby the latter is to take the sulphuric acid product of the copper company for a long term of years, but after the agreement as to terms and the payment of a large cash consideration, some hitch developed on the part of the fertilizer company.

(By B. Britton Gottsberger.)—The year 1908 was for the Ducktown copper district a busy one. The labor situation which has, on account of the isolated location of the mines, always been a serious one, was much relieved by the general business depression, and while the actual

number of men employed did not much exceed that of 1907, a great increase in efficiency was obtained, resulting in increased output and a material reduction in working costs.

The decrease in the price of copper prevented the Tennessee Copper Company from making any but the most necessary additions to the plant, but the development of the mines was actively pushed. The shaft at the Burra Burra mine was sunk to the seventh level and the drifts on the third level, which had been in progress for some time, were completed; so that the orebody is now opened on that level for its entire length, a distance of more than 2000 ft. Diamond drilling in the new ground developed a large orebody in the north end of the mine. At the Polk county mine an additional level was opened and a considerable body of ore of good grade was exposed and will soon be ready for stoping.

A feature of interest was the trial of the method of loading ore in the stopes by means of a special shovel furnished by the Bucyrus company. The shovel operated by compressed air worked smoothly, no difficulty being experienced in handling ore which had been broken to a reasonable size. Its use was, however, not found to be an advantage, since it was difficult to obtain a sufficient tonnage in one stope to meet the capacity of the shovel. In attempting to do this large blasts were necessary, and great difficulty was experienced in breaking large pieces of ore, after the blast, to a proper size for handling by the shovel and later by the crusher. The shovel itself would easily handle any pieces small enough to pass through the dipper, which was of 1¼-cu.yd. capacity; but after hoisting to the surface many of the lumps proved too large for the 18x36-in. Blake crusher, and hoisting was often interrupted on this account. These difficulties were mainly due to the small space available for working the shovel, the face of the stope being about 80 ft. wide. The shovel has been transferred to the company's quartz quarry where flux for the smelter is obtained, and at this point it is doing excellent work.

At the smelter no special change in methods of operation occurred, the most important fact developed during the year being that while the production in one operation of a matte of converter grade (30 per cent. or more) is quite feasible and in fact was made for one month on practically all the furnaces, the copper losses by this method were larger than the loss by smelting in two operations. It is hardly likely that any further attempts along this line will be made.

The adoption on the blast furnaces of a slotted tuyere 12 in. long by 4 in. wide, in place of two tuyeres each 3¼ in. in diameter, proved a decided success, remedying the defect of too small tuyere area in the original design. The furnace equipped with these new jackets handled

a much larger tonnage than the others, and the new tuyere will be adopted on all the furnaces.

When the smelter was built the method of conveying air to the blast furnaces was by independent blast pipes from each engine to the various furnaces. On account of changes to the furnaces this system was abandoned; all engines now deliver air to one large air main from which connections are made to the different furnaces. No change in the operation of the furnaces was noted with the new arrangement, and its flexibility will be a decided advantage in operating the furnaces for the production of gas for the acid plant, removing all danger of the blast's being cut on the furnace from failure of the engine.

The development of greatest interest was that of the sulphuric-acid plant. The year was largely spent in gaining experience as to the proper method of correlating the work of the smelter and the acid plant and solving the many problems which naturally arise in so novel an undertaking. It was found that a radical change in the construction of the furnace tops was necessary, and that special care must be taken at all points to prevent the dilution of the gases by means of false air. After some preliminary attempts a comprehensive plan was devised, and is now being executed, which will completely solve the question of converting the sulphur fumes into commercial sulphuric acid on a profitable basis, rendering available the sulphur hitherto wasted, but now considered a constituent of the ore no less valuable than the copper. The development of the plant will undoubtedly control the fumes to such an extent that the troublesome question of damage to vegetation, which has so long been a matter of dispute, will be eliminated.

Utah.—The increase in the copper production of this State occurred in spite of the idleness of three of the smelteries, the United States having been in operation for only a few weeks at the beginning of the year, while the Bingham and Highland Boy were closed throughout the year. However, the capacity of the Garfield works was doubled, fully making good the deficiency of the other works, but nevertheless there was a glut of ore and a large accumulation of unsmelted products at the end of the year. The Yampa company added a converting plant and thus became able to handle its own matte which previously had been shipped to the United States smelter.

The increase in the copper production of Utah was, of course, due primarily to the operations of the porphyry mines of Bingham. By the end of the year, the Utah Copper Company had got practically into full swing with its whole milling capacity of 7000 tons of ore per day; its copper production for the year was 43,873,918 lb. The Boston Consolidated was less fortunate, it having been found that the ore stripped by

the steam shovels in preparation for extracting for milling was not only of rather low grade, but also was higher than usual in iron, wherefore the concentrate was inevitably of low grade. By a curious smelting contract, the iron in this concentrate was penalized. The company secured a modification by agreeing to hold back its concentrate to a certain amount (the smelter being glutted with ore) and abandoned mining by steam shovels, temporarily at least, concentrating its attention upon underground development in a better grade of ore. For these reasons, the company delayed the completion of its 3000-ton stamp mill and at the end of the year but little more than half of the capacity was in operation. The transportation of the large tonnage of ore from Bingham cañon to the mills and smelters is still a matter of great difficulty. The Utah Copper Company has considered seriously the construction of a new railway line of its own, but according to late advices, such arrangements have been made with the Rio Grande Western Railway Company as will secure a better service.

The Utah Consolidated Mining Company shipped its output in 1908 to the Garfield smelter. In the latter part of the year, the renewal of this contract came up for consideration. After considerable negotiation, it was decided to disagree and the mining company made a contract with the new smelting company under conditions outlined by U. H. Broughton as follows:

"In January, 1909, the company's smelting plant in Salt Lake valley was closed by order of the court, after the case had been carried

STATISTICS OF UTAH CONSOLIDATED.

	1904	1905	1906	1907	1908
Ore treated tons	233,700	286,200	296,989	279,642	248,215
Copper, lb	13,553,483	17,264,474	18,533,974	13,987,551	10,648,248
per ton of ore	57.9	60.3	62.4	50.0	43.0
Silver, oz	268,880	374,685	457,812	390,296	265,283
per ton of ore	1.1	1.3	1.6	1.4	1.07
Gold, oz	23,374	28,290	42,601	34,554	23,440
Net profit	$1,179,412	$2,835,008	$1,887,385	$1,164,348	$326,312
Mining cost	.....	.....	473,760	582,866	461,711
Exploration and development	.....	.....	84,864	107,155	73,441
Smelting cost	.....	.....	747,717	867,087	902,266
Custom ore purchases	.....	.....	274,032	131,796	*Nil.*
Refining charges, freight, marketing, etc.	.....	.....	267,921	227,152	141,401
Miscellaneous	.....	.....	70,773	70,754	150,410
Total costs	(a) 1,295,626	(a) 1,590,881	1,919,067	1,986,810	1,729,229

(a) Costs not segregated in report.

to the court of last resort. The company was then confronted with the question of building a new smelting plant in another place, which meant the cessation of production for about a year and a half, or the alternative of entering into a contract for the smelting of its ores with the Garfield Smelting Company. In February a contract was made with the

Garfield Smelting Company for one year. This contract was the best that could be made under the circumstances; by its provisions the company had the option, in six months' time, of extending the contract for a second year, and the right of discussing the questions of a certain modification and the making of a contract for a period of five years. In July the company exercised its option to extend the contract for a second year, and informed the Garfield Smelting Company that it was prepared to take up the other questions."

After protracted negotiations the two companies failed to come to an agreement, and the Utah Consolidated then entered into a contract with W. D. Thornton, of Butte, for the smelting of its ores for a period of 10 years, commencing April, 1910. Under the provisions of this contract Mr. Thornton agreed to erect a smelting plant in Tooele county, Utah, to be in operation by April, 1910, and treat the output of the Utah Consolidated up to 1200 tons per day, at a rate which is 67.60c. per ton more favorable than the contract existing with the Garfield Smelting Company. The cost of transporting ore to the Garfield smelter is 48c. per ton, while the cost of transporting ore to the Tooele smelter will not exceed 15c. per ton. Subsequently the Thornton contract was turned over to the newly organized International Smelting and Refining Company.

The Boston Consolidated Mining Company in its fiscal year ending Sept. 30, 1908, produced from its Sulphide mine 79,300 tons of ore containing 7447 oz. gold, 55,705 oz. silver and 3,459,911 lb. of copper. The cost of mining, including development, was $183,992; transportation to smelter, $43,613; smelting, $217,049; freight and refining charge on bullion, $60,188; total, $504,841. There was deducted for losses in smelting and refining 249,880 lb. of copper and 6573 oz. of silver. After crediting value of gold and silver, the cost of the copper delivered at New York was 10.22c. per lb. At the Porphyry mine, the first shipment of ore mined by steam shovel to the Garfield mill was made on Jan. 13, 1908. As the shovels removed the capping they exposed an unlooked for grade of ore, satisfactory as to copper content but high in iron pyrites, which greatly reduced the ratio of concentration. Underground development subsequently proved that not over 16 per cent. of the payable orebody is included in this class of iron ore. Steam-shovel work was gradually suspended as the ore became available from underground working. During the year 411,175 tons of stripping were removed at a cost of 17.72c. per ton. The amount of ore mined by steam shovels and shipped to the Garfield mill was 26,647 tons. Meanwhile, underground developments were made with a view to the inauguration of mining by the caving system. In the course of this development work 128,391 tons of porphyry ore were obtained. The first section of the Garfield mill was put in oper-

ation on Jan. 27, 1908. Only four sections were operated until Sept. 25, when four more were put to work. The mill recoveries ranged from 70 to 72 per cent., and milling cost was reduced from a maximum of $1 per ton experienced in the first few months of operation to 47.2c. per ton in September. The total cost of the mill to date is $1,568,604. The mill treated 143,284 tons of ore, producing 9935 tons of concentrate, containing 397 oz. of gold, 4967 oz. of silver and 2,937,599 lb. of copper. Sidney J. Jennings, consulting engineer to the company, made a report under date of Dec. 1, 1908, in which he states that the Porphyry mine has 27,889,000 tons developed of ore averaging 1.52 per cent. copper, besides 9,550,000 tons averaging 1.23 per cent. He estimates that the ore will be mined by steam shovels for 31c. per ton and by the caving system for 70c. Both methods will have to be employed. The capacity of the mill is estimated at about 1,000,000 tons per annum. Ratio of concentration will be 13:1 and the concentrates will assay approximately 14.40 per cent. copper. On the supposition that one-half of the ore will be mined underground and one-half by steam shovels, the cost of production will be as follows: Mining, 50.5c. per ton; transportation to mill, 28.7c.; milling, 40c.; smelting and refining charges, 70c.; general expense, 10.2c.; selling commission, 3c.; interest on bonds, 10.3c.; total, $2.127; this cost will decrease as the proportion mined by steam shovels increases. The recovery of copper from the average ore should be 19.5 lb. per ton, and the cost delivered at New York with allowance for gold and silver, 10.5c. per lb. Up to Sept. 30, 1908, the company had expended a total of $3,078,499.83, of which $135,197 was for porphyry development; $488,484 for steam shovel development; $1,600,483 for Garfield mill and shops; $91,668 for steam shovel equipment; $136,028 for steam shovel railway; $86,800 for gravity tramway; $137,758 for railway extension.

The Ohio company resumed construction of its concentrating mill in the early fall of 1908 and it is expected that the first unit will be ready for operation early in 1909. The full capacity of the mill will be 2000 tons per day. The properties in the Bingham district of the U. S. Mining Company and the Bingham Consolidated (which is in the hands of a receiver) were practically idle in 1908.

Outside of Bingham, mines in the Tintic district produced in 1908 about as usual. The plant of the Tintic Smelting Company was finished in August and was blown in, but did not operate continuously. At the Cactus mine of the Newhouse Mines and Smelters in Beaver county, operations were conducted with full force after the first quarter of the year. Sinking is being continued from the tunnel level, it being the intention to carry the shaft 600 ft. below this level and drive a tunnel direct to the mill, thus cutting out 2½ miles of railway haul. The smelt-

ery at Ogden was taken over by a new company, the Independent Smelting Company.

Vermont. (By George A. Packard.)—Very little copper ore was mined in Vermont in 1908. The Eureka mine at Corinth was closed during the entire year, and but little work was done on the nearby property which has been partially developed by the Corinth Copper Company. At South Strafford the Vermont Copper Company continued equipping its mine. Power development was seriously delayed by winter and spring freshets, which twice carried away the dam on the White river when in a partially completed stage. This power plant, which is situated at Sharon, is now ready for operation. The station is built for 750 h.p., but the equipment so far installed includes water-wheels and generators for only 500 h.p. This is transmitted at 6600 volts, 3-phase, and the motors, locomotives, hoists and pumps have been tested and found satisfactory. The flues and stack are completed, and the furnace will be blown in early in 1909. It is expected that between 300 and 400 tons of ore per day will be treated at the start. Plans for a sulphuric-acid plant are being prepared, and surveys for a railroad to connect with the Boston & Maine system are completed.

Virginia.—In spite of the lower range of prices for the metal, the output of this State which attained considerable proportion in 1907 was well maintained in 1908. The major portion was derived from the Virgilina district. At the Cabin Branch pyrites mine, the copper content of the ore has been found to increase with depth, and a matte smelting furnace was erected and operated in 1908.

Wyoming. (By Henry C. Beeler.)—The copper production of this State continued small in 1908, owing to difficult conditions of transportation, which are gradually being overcome as railways are further extended. The only smeltery in the State is at Encampment, where the Penn-Wyoming Copper Company operates the Ferris-Haggarty and the Doane-Rambler mines. This company built 44 miles of railway from Walcott on the Union Pacific to Encampment in 1908, and ran the greater part of the year. It is now doing development work in its two mines in order to make a larger production in 1909. Several new discoveries were made in the Encampment district, that of the West Virginia-Wyoming company, in Battle Basin, being the most important, inasmuch as it proves a greater extent of the contact ore of the district.

After about six years of experimenting, the Rambler Mining Company, operating in the Medicine range in Albany county, succeeded in separating the platinum and palladium which occur in commercial quantity in the copper ore, and during the fall of 1908 the company had a force of men reopening the mine. The building of the Laramie, Hahns Peak

& Pacific Railroad from Laramie to the North Park coal fields brings transportation within a few miles of this and adjacent mines on Lake and Douglas creeks, as well as the Shawnee and other properties on French creek and the Snowy range, where some good copper prospects are being quietly developed. The Williams-Luman mine at Depass on Copper mountain, Fremont county, is opening the largest deposit of copper ore yet discovered in Wyoming. About 2000 ft. of development work has disclosed an orebody 80 ft. wide, the length not yet having been proved. The Boysen power dam in Big Horn cañon at the west end of Copper mountain is relied upon for power for the adjacent mining territory. The plant will probably be in operation by the spring of 1909.

Copper Mining in Foreign Countries.

Algeria.—The Kef Oum Theboul mine, 11 km. southeast of La Calle, has been extensively developed, but is not now in operation. It has a wide vein of chalcopyrite mixed with blende and argentiferous galena in schists and sandstones, and is now considered as a zinc mine rather than a copper mine. The Ain-Barbar mine, 22 km. from Bône, yields a complex sulphide ore. The Achaiches mine, 36 km. from Collo, is operating in a deposit of cupriferous pyrite in schist. The Cavallo mine, 18 km. west of Djidjelli, is now idle; the ore is found in a lenticular mass, consisting of complex sulphides in igneous rocks. The Tadergount mine, 34 km. southeast of Bougie, has opened a group of veins containing gray copper and oxides, in schist and Liassic shales. The Djebel Ouenza mines belong to the Compagnie des Mines d'Ouenza, and are situated near the Tunisian frontier, 25 km. from Clairfontaine, on the Tébessa railroad. It is an oxidized deposit, covered by an immense capping of cupriferous hematite. Attempts to work it for copper have not yet been attended with success. The Bou-Kadra deposit of the Compagnie de Mokta-el-Hadid is a little north of Ouenza. It consists of veins of copper-bearing pyrites running through a mass of hematite which also contains copper and manganese. The Mouzaïa deposit consists of a capping of iron ore traversed by veins of argentiferous gray copper, with masses of copper oxides and pyrite veins; but little work has been done on this since 1876. The Hendjir deposits are 14 km. west of Aïn Sefra (Oran), and belong to the Compagnie Industrielle et Minière des Cuivres d'Algérie.

Argentina.—The mines of Rioja province are situated along the Famatina range of mountains, which parallels the Andes, the highest point of the chain being 5950 m. In 1907 the Government, as a cost of approximately $1,000,000 gold, constructed a wire ropeway to assist in transportation. The main ropeway is 35 km. in length, and the capacity of the

line is 40 tons per hour. One mining company is developing copper-silver lodes, another is smelting lead ores, and two concerns are producing a high grade of copper matte. One company started producing in May, 1908, since which time it has shipped 2700 tons of the matte to the United States. The climate is dry and healthful. The Famatina Development Corporation, Limited, stated officially, on June 26, 1908, that the mines are producing more ore than was originally estimated, and it may safely be taken that there are more than 200,000 tons of ore available for stoping (or over four years reserves on the basis of 150 tons of ore per day). Development work will be continued as soon as the smelter and converter plant are in running order, which was expected to be before the end of the summer. Latest advices from this mine are that pyritic smelting has proved unsatisfactory.

Australia.—Owing to the lower prices for the metal, the copper production of this Commonwealth displayed a falling off in 1908. The works of the Electrolytic Smelting and Refining Company at Port Kembla were completed and made ready for operation in December. This concern has contracts with the Mt. Morgan and other companies for raw material, and for the first time puts Australia in position as a producer of electrolytic copper.

Queensland.—This State experienced a great reaction after the boom in copper, yet its production in 1908 was somewhat larger than in 1907. The Mt. Morgan company of course holds the premier place. In its mine the oxidized ore in the surface levels is rapidly becoming exhausted, but the supply of pyritous ore from the intermediate levels is well maintained, while development work underground has revealed the existence of larger quantities of auriferous copper ore than was previously estimated from the diamond drilling. The average grade of the ore so far mined is 3.3 per cent. copper and 8.62 dwt. of gold per ton. The present reduction plant is capable of treating 216,000 tons of ore per annum. At the O. K. mine developments were satisfactory, ore reserves being considerably augmented. Toward the end of 1908, this mine made an output of about 140 tons of blister copper per month. The Chillagoe company maintained a satisfactory output in 1908.

New South Wales.—Several of the important producers suspended operations in 1908. The Great Cobar, Ltd. maintained its position as the largest producer and completed extensive additions to its plant. At the 1000-ft. level, the lode has been found to improve in width with no diminution in value.

Tasmania.—The Mt. Lyell company experienced a satisfactory year, and added materially to its ore reserves.

South Australia.—The Wallaroo & Moonta company found it necessary to restrict its production in 1908, while making certain necessary readjustments in operating conditions.

WALLAROO & MOONTA STATISTICS.

	1904	1905	1906	1907	1908
Ore from Wallaroo mines, tons	21,706	35,189	43,241	53,571	40,383
Ore from Moonta mines, tons	10,686	8,161	10,360	16,397	10,913
Precipitate, tons	494	930	957	833	813
Outside ores and matte, tons	5,381	5,151	3,520	4,865	1,439
Ore smelted, tons	38,995	49,961	58,068	75,666	58,009
Copper placed in store, tons	5,835	6,501	7,561	8,627	6,448
Silver to mint, oz.	7,147	4,781	3,614	5,845	7,416
Gold to mint, oz.	1,260	1,646	1,643	2,009	1,950
Sulphuric acid delivered, tons	3,433	5,312	5,112	5,379	4,953
Bluestone made, tons	181	340	328	224	201

A large deposit of sandstone, mineralized with copper carbonate, is reported to exist in South Australia about 92 miles northwest of Port Augusta and 1½ miles north of Mount Gunson. The Government Inspector says that this remarkable copper deposit is extensive and exceedingly promising, being horizontal and near the surface, so that it can be easily and cheaply worked. When the mine is equipped with the necessary treatment plant, it should become an important producer of copper.

British East Africa.—According to a report of the commissioner of mines, copper has been found in British East Africa in the valley of the Tsavo river, which flows from Kilimandjaro eastward. The principal deposit lies about 70 km. eastward from the station of Tsavo on the Uganda railway, and consists of a quartz lode carrying copper. The lode extends from the foot of the Kiulu mountain through the Tsavo valley in a southeasterly direction.

Canada.—The copper production in the Dominion, computed from direct reports from all the producers, was 53,725,213 lb. in 1908, compared to 46,356,382 lb. in 1907. These statistics are computed on the basis of blister copper reported. Several smelters producing matte in Canada reported matte shipments in excess of the blister reported by the concerns converting their product, the difference being explained, of course, by material in transit and accumulated in stock.

The Granby company, in the year ended June 30, 1908, smelted 858,432 tons of its own ore and 24,179 tons of purchased ore; a total of 882,611 tons. The product was 21,126,926 lb. copper, 300,593 oz. silver and 40,139 oz. gold. The average price realized for copper was 13.333c. per lb.; for silver, 56.625c. per ounce. The average cost per ton of ore, including all expenses, was $3.486. The average cost of copper, after deducting value of gold and silver, was 10.31c. per pound. Expenses

include all development work, renewals and repairs. There was expended for additional mining property, $23,743; new construction and equipment at mines and smelter, $242,574; total, $266,317 for additions.

The president's report says: "The new Victoria shaft and surface equipment, including crusher, electric hoist, belt conveyer, etc., have now been in operation 12 months, and have given satisfaction, both in quantity of ore handled and economy of cost. The mechanical devices now in operation are able to handle about 5000 tons of ore daily. The mines shipped to the smelter a total of 858,432 tons during the past year against 644,549 tons during the previous year. The grade of ore mined was not as good as in the previous year, carrying about 1 lb. less of copper and 10 to 12c. less per ton of ore in gold and silver. The extraction per ton of ore on the average was 23.42 lb. copper, 0.2865 oz. silver and 0.0454 oz. gold; as compared with 24.43 lb. copper, 0.3088 oz. silver and 0.0503 oz. gold in 1906-07, and 24.30 lb. copper, 0.3107 oz. silver and 0.0513 oz. gold in 1905-06. During the summer months a lower-grade ore was shipped, due to mining the glory holes on the surface, which are low grade and which cannot be mined during the winter. Consequently, the ores will average richer in winter and spring and poorer in summer and fall. The quantity of ore developed by diamond drilling has been larger than the ore extracted."

The report of the British Columbia Copper Company for the fiscal year ended Nov. 30, 1908, shows that during seven months the mines shipped 319,527 tons of ore. At the smeltery three blast furnaces in six months handled 321,427 tons of ore, which yielded 5,767,355 lb. of copper, 13,597 oz. of gold and 58,204 oz. of silver. The cost per ton of ore handled, including all charges from ore in place to sale of the metal products was $2.63 per ton; the cost of producing, refining and marketing per pound of fine copper, after crediting expenditure with value of gold and silver realized, was 9.996 cents.

(By Ernest Jacobs.)—In the Boundary district in 1908, the Granby company smelted 1,023,000 tons of ore; the British Columbia Copper Company, 358,000 tons; the Consolidated Mining and Smelting Company, 41,000 tons; the Dominion Copper Company, 22,000 tons (this company operating its smelter for only a few weeks previous to going into liquidation last summer). Numerous improvements were made at the Granby smelting works, including plans for the installation of three new converters, which will have a capacity for producing 36,000,000 lb. of copper per annum, and will replace the plant at present in use. The lengthening of the eight blast furnaces, at present 44 in.x18 ft., to 22 ft. each has been commenced; when this is completed, probably by June, 1909, the works will have capacity for 4000 tons of ore per day instead of 3000

tons, as at present. The Granby mines are already equipped to deliver 5000 tons of ore per day. On an adjoining group of claims the Consolidated Mining and Smelting Company opened what promises to be a large body of pay ore. The British Columbia Copper Company made some important improvements at its mines and smeltery.

The mines of the Rossland district are estimated to have produced 5,300,000 lb. of copper in 1908, as follows: Center Star, 2,606,000 lb.; Le Roi, 1,705,000; Le Roi No. 2, 928,000. The most important result of the year was the discovery of orebodies of good grade in the deeper levels of all three mines; in the Le Roi on the 1650-ft. level; in the Center Star down to about 2000 ft. on the incline; and in the Le Roi No. 2 on the 700-ft. level, and deeper in a diamond drill bore.

In the Coast district the Marble Bay mine, Texada island, shipped upward of 10,000 tons of ore containing 5 per cent. copper. The Britannia company resumed operations on a small scale at Howe Sound in June, and shipped about 10,800 tons of ore, containing approximately 646,000 lb. of copper. The Tyee Copper Company shipped several thousand tons of ore from the Indian Chief group. The Tyee smelter, at Ladysmith, received ore also from various small mines. The Britannia smelter was not operated in 1908. The Hidden Creek Mining Company developed a large body of ore on Observatory inlet, Portland Canal, along a distance of about 2000 ft., ranging in copper from 2 to 10 per cent., but it did not ship any in 1908.

Chile.—The copper production in 1908 was 35,000 metric tons (compared to 28,854 tons in 1907), which it is believed will increase further with the opening of the deposits of Collahuasi, Calama, Chiquicamata, and several others, as also with the starting of establishments of greater capacity for lower grade ores. The average grade of the ore worked in Chile in 1907 was 9½ per cent. a portion of which goes abroad, with a mean percentage of 20 per cent., to be treated there. The mines lack systematic development, and properties worked more or less constantly for a long time show but little depth. The greatest depth is about 1000 m., at the Dulcinea mine in Puquois, Copiapo, but this is an exception. The mines that go beyond 500 m. are not more than eight in all the Republic. Developments by the Braden Copper Company of Rancagua are said to have been satisfactory in 1908. A new smeltery is in course of construction under the direction of Count Bernard de Saint Seine in the zone of Naltagua, in the department of Melipilla. A hydrometallurgical plant of 100 tons daily capacity, to leach ore (4-6 per cent. copper) with dilute sulphuric acid and precipitate with scrap iron, has been built by the Sociedad Chilena de Fundiciones at Guayacan.

China. (By T. T. Read.)—Copper ores occur at numerous places in the north and southwestern parts of China. In the southwest (Ssu-chuan Yunnan, Kueichou and Kuangtung) copper ores are widely distributed, according to the reports of the various explorers. Yunnan is the source of all the present production, but it is impossible to learn the amount, as no records of any sort are kept, and as it is a government monopoly it does not pass through the hands of the customs. It must be quite large, however. The mining district is now reached by a railroad extending up from French Indo-China. Only fragmentary accounts of the deposits and the methods of work exist.

Colombia.—In the environs of Natagaima (department of Tolima), on the right bank of the Magdalena river, there is a mining region which, according to F. Pereira Gamba, has a remarkable resemblance to Lake Superior. Numerous copper ore veins are visible, and native copper in small grains or lumps weighing several pounds are found. The Magdalena as a transport way and the River Achique as motive power present advantages as regards the opening of mines.

Cuba.—This country is now a considerable producer of copper. The importation of ore and matte into the United States in 1908, which represents practically the production, was 40,877 long tons, containing 6,538,047 lb. of copper.

French Congo.—A copper mine is to be worked at Mnidouli, whither a railway is now being built. This line, commenced last November, is now 12 km. long, and will, it is estimated, reach Mnidouli in less than two years, 121 km. from the starting point at Brazzaville. What is believed to be an extensive copper deposit has been discovered at Bamanga, 7 km. from the railway station of Ponthierville. The first veins located were 20 to 90 cm. thick, and about 80 tons of crude ore shipped to Antwerp was found to contain 35 per cent. copper. Prospecting is being continued. Outcrops have been found 4 km. north and 5 km. south of Bamanga.

German Southwest Africa.—Prospecting operations in this colony yielded poor results in 1908. The Otjisongati mine produced 525 tons of ore, averaging 18 per cent. copper. Development of the Otavi mines and railway suffered from a scarcity of labor. The smeltery began regular work early in the year, 30 tons of ore per day being treated, with an output of eight tons of lead and 10 tons of copper matte, assaying 40 per cent. copper. Ore from the Guchab mine, containing 30 to 40 per cent. copper, was shipped directly. The total output of the company was 25,700 tons of ore, of which 14,000 tons were shipped and 3200 tons smelted, the remainder being put into stock.

Germany.—It was reported that the Mansfeld company would suspend work at the Martin shaft, Kreisfeld, at the close of 1908, the vein of ore that has been worked for the last 70 years having become exhausted.

Italy.—According to Major A. Greco, the Italian copper mines contain poor ore, with the exception of Montecatini. The chief ones are in Liguria and Tuscany. In the first district the most important is the Libiola mine. However, the main part of the ore is from the deposits of Tuscany—Montecatini, Boccheggiano, Fenice Masselana, Capanne Vecchie, etc. Except for the Montecatini ore, which contains 20 per cent. copper, the percentage varies from 5 per cent. to 12 per cent.; in some cases, even less than 5 per cent. Pyrites is abundant near the Alps—at Alagna (near Novara), Agordo (near Belluno), and other places in the province of Turin. The ore, found in numerous veins, was for some time smelted by an electro-metallurgical company with works at Agordo, but subsequently was exclusively utilized for manufacture of sulphuric acid. On the north slope of the Appenines, near Parma and Piacenza, there is an abundance of chalcopyrite, though very poor in copper.

Katanga.—The directors of Tanganyika Concessions, Ltd., have issued an interim report showing progress made since the last meeting of shareholders, from which the following is extracted:

The smelting operations commenced on the Kansanshi mine in March, 1908, proceeding satisfactorily. It is expected that at least 100 tons will be the normal monthly production from the present furnace. Mr. Allan Gibb's general conclusions as to the Star of the Congo mine, on which developments have been commenced with the object of preparing it to supply an output of at least 1000 tons of copper monthly as soon as the railway arrives, are summarized as follows: The copper-bearing beds are about 3500 ft. long and 500 ft. wide. At places full 250 ft. of the width will not be of immediate value on account of mining or metallurgical difficulties. At places a considerable portion of the remaining 250 ft. is so low in copper as not to be of immediate value. The general cost of mining and delivering the two classes of ore to bins will be 4s. per ton of ore. The mine can be quickly put into condition to produce 12,000 tons of ore monthly, which will insure a production of 1000 tons of copper monthly. The bulk of the ore to water level can conveniently be removed by haulage arrangements that will not require shafts. There are 320,000 tons of ore with 15 per cent. copper available that can be directly smelted to bar copper at a cost of about 35s. per ton of ore, assuming coke can be delivered at £5 per ton. There are 900,000 tons of ore with 6.5 per cent. copper available that can be treated by the reduction method at a cost of 12s. per ton of ore. Large metallurgical works cannot be erected at any point nearer than nine miles

from the mine. The general investigation of the Kambove mine shows there are about 3,000,000 tons of ore in sight.

At the annual meeting of Tanganyika Concessions, Ltd., in December, 1908, Robert Williams, the managing director, made the following statements to the stockholders:

"I have the assurance of the railway contractor that he will have the rails through to the Star of the Congo mine in 18 months, and Mr. Gibb has promised to have plant ready at the Star mine in 18 months to produce from 1000 to 3000 tons of copper per month. I am confident that within three years from date the Katanga mines will be producing 5000 tons of copper per month from the eastern section of the copper belt owned by the Union Minière Company. The cost of the copper from the Star in Katanga will not exceed £15 per ton on the spot, although we believe a considerably lower figure will be nearer the mark, when the Kambove and other mines are producing, but even at £15 per ton the cost would not exceed £25 per ton delivered in Europe via Beira.

"The Kansanshi mine has been producing from 80 to 90 tons of copper per month at a cost of £15 per ton, at the works, and we are sending up a larger furnace, which will produce from 200 to 250 tons per month at a cost of about £12 10s. per ton. Within a year I expect we shall have the railway so close to Kansanshi that we will bring down the whole of the present stock by traction engines to the rail-head, as well as the regular monthly output, and land it in England at £30 per ton. Certainly we should have landed in England within two years from this date 3000 tons of copper from Kansanshi.

"Last July I carried through railway agreements as follows: A.—The Benguella railway will continue the construction of its line from Lobito bay to the frontier separating Angola from the Congo state between the 11 deg. and 12 deg. parallels south. B.—The Lower Congo railway will construct the section from Ruwe mine to meet the Benguella railway as soon as the lines of approach enable it to do so. C.—The Katanga Railway Company will construct a line from the frontier of Rhodesia to the Star of Congo mine as soon as the Southern railway reaches that frontier. The Katanga Railway Company will extend its line from the Star of Congo mine to Ruwe, and eventually to a navigable point on the Lualaba, as and when circumstances justify that extension. D.—The Chartered Company agrees to form a company with sufficient funds to construct a line from the present terminus of its railway at Broken Hill to the Congo state frontier.

However, the Chartered Company was unable to finance its section, and other arrangements had to be made. The following rates originally agreed upon for the carriage of ores, metals, concentrates, etc., from the

northern boundary of northwestern Rhodesia to Beira or any intermediate point are illustrative of the conditions that are expected: 1d. per ton per mile in respect of an annual tonnage of less than 250,000 tons. ¾d. per ton per mile in respect of an annual tonnage equal to or above 250,000 tons. ½d. per ton per mile in respect of an annual tonnage equal to 1,000,000 tons or more. The rates for the carriage of coal or coke imported via Beira or from any intermediate point in Rhodesia will be: 1d. per ton per mile for full trainloads of 300 tons. 1½d. per ton per mile for loads of 100 tons or upward, but below 300 tons.

Korea.—The Djin Dong mine in Kapsan has recently been acquired by Americans. This mine has been worked by the natives for upward of 1000 years, copper having been produced by crude methods of smelting. The mine is said to possess a large deposit of pyrites, free from quartz, averaging 10 per cent. in copper. Conditions are favorable for mining and smelting, and it is believed that this mine will become a profitable producer.

Mexico.—On the basis of the net imports into the United States plus the Boleo production, the copper production of Mexico in 1907 was 126,710,000 lb. Our reports from all of the smelting works, producing blister from Mexican material in 1908, give 89,576,464. Boleo and Moctezuma increased their production, but not enough to offset the decrease from Cananea, which was idle during the major part of the year, and Teziutlan idle throughout the year. Operations are now in full swing again at Cananea, and the new concentrator at Moctezuma is working. Boleo was the largest producer in Mexico in 1908, Aguascalientes next, and Cananea third.

The most noteworthy event was the resumption of work at Cananea on July 11. It required a good deal of nerve on the part of the management to suspend operations and practically disband the organization of a great mining camp, but, as it is always easy to say after such a step has been taken, it was the correct and obvious thing to do. The future of the Greene-Cananea company depends upon its ability to produce copper at a much lower cost than heretofore. By the introduction of improved methods of mining and the modernization of the smelting works, the management now expects to be able to compete successfully with the other big copper-producing companies.

The property of the Moctezuma Copper Company consists of about 2500 acres of mineral ground near Nacozari, on which has been operated the Pilares mine. This is in a mineralized body, oval in shape, and which retains approximately its dimensions between surface and the 700-ft. level. It is about 1700 ft. in diameter from north to south, and 1200 ft. in diameter from east to west, and contains a large area of profitable ore.

A narrow-gage steam railroad five miles in length connects the mine through the Pilares tunnel with the concentrating mill at the town of Nacozari. The tunnel itself, with its branches, is over a mile in length, and is large enough to allow of the railroad cars reaching through it the different sections of the mines, and receiving their charge from large bins excavated out of the ore. These bins are of a capacity of several thousand tons each, and are fed through chutes extending to the surface levels, the chutes also being excavated from the mineralized ground.

The quantity of what may be considered ore depends entirely upon the grade which it is profitable at a given price for copper to work, but the mine is at present opened up for an extraction of 1500 to 2000 tons a day of ore of an average grade of 3 per cent. The daily capacity of the concentrating mill just completed at Nacozari is 2000 tons. At Nacozari is a well-designed power plant, equipped with Curtis turbines of over 4000 h.p., for transmitting high-voltage current to both the mill and the mine. For the time being it is found to be more profitable to convey the concentrates and rich ores by the Nacozari railroad to Douglas, Arizona, 70 miles distant, than to smelt them on the spot, the ores being treated at Douglas at the same profit to the Copper Queen company as though they were custom ores supplied by an unallied customer.

The old concentrating mill, still intact, with its very efficient gas engine and gas generating plant, which up to within a few months treated 600 tons of ore a day, is now out of commission, though it can be stated at short notice as a supplemental plant to the new mill, should this course be desirable.

MOCTEZUMA COPPER COMPANY.

Year.	Pounds Copper.	Net Earnings.
1903	10,281,970	$456,524.55
1904	11,061,649	598,992.36
1905	10,160,016	533,117.66
1906	12,714,726	1,195,424.18
1907	9,640,390	833,236.25
1908	15,522,580	524,902.00

The increased capacity of both the mine and the concentrating mill, owing to recent improvements, is indicated by the fact that in May, 1908, the production was 784,892 lb. of metallic copper, whereas by the month of October it had reached 2,300,000 lb. of copper, contained in 9500 tons of 12½-per cent. concentrates. A production of approximately two million pounds of copper a month can now be maintained; and, if the market demanded it, this production could be increased to three million pounds per month. Up to the end of 1908 the company had paid a total of $2,262,000 in dividends. None was paid in 1908.

The Teziutlan Copper Company in 1908 completed its new smeltery, comprising two blast furnaces, capable of treating 1000 tons of charge, or about 750 tons of ore, per day. The company is estimated to be able to produce 24,000,000 lb. of copper per annum. However, the works have not yet been put in operation, the management awaiting a higher price for copper, although it is estimated that the company can deliver copper in New York at a cost of only 8c. per pound.

Newfoundland.—The Tilt Cove mines made a profit for the year ending Dec. 31, 1907, of £51,034, one-half of which was paid to the Tilt Cove Copper Company and the other half was credited to the Cape Copper Company. The output from the East mine was 27,062 tons of 3.3-per cent. copper. The South Lode returned 12,961 tons of 3.53-per cent. copper. The ore reserves at the East mine and the South Lode are given as 45,756 tons.

Norway.—In 1908 about 250,000 metric tons of pyrites were exported. At Sulitjelma, the production in 1908 was about the same as in 1907, i.e., about 110,000 tons of ore, of which nearly 100,000 tons was pyrites. From its own ore, the company produced 750 tons of copper, and 175 tons from ore from the Alten Kobberverk. Foldalen produced 53,000 tons of pyrites, averaging 47 per cent. sulphur and 2 per cent. copper. The Röros Kobberverk produced about 700 tons of refined copper and about 10,000 tons of pyrites for export. Rustvangen in Tönset produced 18,728 tons of pyrites, assaying 44 per cent. sulphur and 3½ to 4 per cent. copper. Bossmo Kisgruber, at Mo in Ranen, produced about 22,000 tons of concentrated pyrites, assaying 49 per cent. sulphur.

Persia.—According to a British consular report extensive deposits of copper ore occur within 20 miles of Kerman. Coal has been found in the same district.

Peru.—The Cerro de Pasco Company was the most important producer in 1908, its output having been 25,000,000 lb. as compared to 20,000,000 in 1907. The Rio Blanco works of the Peruvian Mining, Smelting and Refining Company is a well-designed plant of 300-tons daily capacity, comprising sintering pots and a modern blast furnace.

Portugal.—The best copper mines are in Algarve, while in the district of Beja are the Cova Redonda and Barrancos mines, important because of the extensive network of veins. The principal mines of cupreous pyrites are in the municipality of Mertola (district of Beja). The best known is that at St. Domingos, served by the harbor of Pomarao, on the River Guadiana, its production in 1907 having been 361,270 tons of pyrite and 1058 of copper precipitate of 60 per cent. In the same district is the mine of Aljustrel, served by the Government railway, its production in 1907 being 13,087 tons of pyrite and 1475 tons of copper precipitate of

70 per cent. In the municipality of Grandol (district of Lisbon) are the mines of Lougal and Serra Carvoeira. Besides these mines, there are others in the districts of Castello-Branco, Avein, Guarda, Villa Real, and Braganza.

Rhodesia.—In the eastern part of Rhodesia the copper mines of Umkondo and Edmundian are being developed. The Edmundian started smelting in October, 1908. The concentrating plant consists of pneumatic stamps and Elmore vacuum oil concentrator, and the smelting plant of calciners and reverberatories. The matte is shipped. Power is obtained from a hydro-electrical power plant four miles from the mine.

Rumania.—This Kingdom has three copper districts, viz., Mehedinti, Dobroudja and Suceava. At Baia-de-Arama in the first mentioned district, and throughout almost the entire zone of serpentines and andesites are found veins of chalcopyrite. The mineralized zone is extensive and continuous, and the geological occurrence is said to be identical with that at the great Servian mine at Bor. In Dobroudja, numerous discoveries of copper ore have been made near the center of the Tulcea district, notably at Akpunar and Altântepe, at which places considerable development has been done.

Russia.—The production of this Empire showed a large increase in 1908, notwithstanding the fact that one of the largest smelters in the Urals was shut down for remodeling. At the beginning of 1908 an association of the Russian copper producers was formed, which controls practically 90 per cent. of the production. Through its selling agency this association has made contracts with all the copper rolling works in Russia at prices relative to the London market.

Smelting at the Spassky works in the Akmolinsk district of Siberia, was described by William A. Heywood in *Min. and Sci. Press* of July 11, 1908. The ore is smelted in brick blast furnaces 4x20 ft. at the tuyeres and in smaller ones 4x10 ft., the hight from tuyeres to charge

PRODUCTION OF COPPER IN RUSSIA.
(In poods. 1 pood =16.381kg. =36.114lb.)

Years.	Ural.	Caucasus.	Siberia.	Kirghis Steppe	Altai.	Finland.	Various.	Total.
1895	151,511	166,728	.....	1,394	15,888	21,858	.....	357,379
1896	167,574	149,698	.....	1,868	13,239	23,640	.....	356,019
1897	220,783	162,534	.....	3,586	15,427	21,360	.....	423,690
1898	236,863	173,993	.....	2,440	16,341	15,445	.....	445,082
1899	253,610	171,568	.....	5,754	15,292	13,664	.....	459,888
1900	241,148	227,079	.....	11,273	11,322	13,354	.....	504,176
1901	217,063	247,348	.....	21,993	13,193	17,311	.....	516,908
1902	279,135	213,273	.....	25,238	7,431	13,231	.....	538,308
1903	265,116	262,919	.....	17,902	7,546	10,126	.....	563,609
1904	265,915	296,666	.....	30,513	7,344	.....	.....	600,438
1905	225,800	223,800	67,000	.....	.....	.....	53,900	570,500
1906	288,600	232,300	40,500	.....	.....	.....	74,700	636,100
1907	457,904	310,244	68,957	.....	.....	.....	65,253	902,358
1908	522,584	296,379	155,116	.....	.....	.....	54,616	1,028,695

floor being 5 ft. About 35 per cent. of bituminous coal is used. The furnaces run very slowly, producing matte containing 55 per cent. copper. The slag is remarkably high in silica and lime, and low in iron. The matte is refined to blister copper in reverberatory furnaces. The ore assays about 16 per cent. copper. The smeltery is 560 miles from the nearest station on the Siberian railway and the mines are 72 miles from the smeltery, yet ore is carried by camels or oxen to the smeltery for about $3 per ton, while machinery is hauled the 560 miles to railway for about $15 per ton.

(By I. I. Rogovin.)—The large increase in copper production in Russia in 1908 was due to the Ural and to Siberia. In the latter region the increase was due to the extension of work by the Spassky Copper Company. The importation of copper into Russia decreased somewhat. The total consumption of copper in Russia in 1908 was approximately 1,200,000 to 1,300,000 poods. Notwithstanding the fall in prices, the market was very lively. A number of works which suspended operations resumed later on, while the Spassky Copper Company not only increased its output, but also took over the Kishtym works in the Ural, where an electrolytic refinery is to be established. It is the hope of those engaged in the copper industry in Russia that the Empire will soon be relieved of the necessity of importing foreign refined metals. Russian electrolytic copper has already appeared in the market in a small quantity.

Santo Domingo.—According to a British consular report, copper ranks next in importance to gold in the Dominican Republic. Several lodes are believed to be profitably worked in Mount Mateo, Nigua river, in the province of Santo Domingo; the ore of these mines is said to yield from 30 to 33 per cent. copper. In Recodo, in the province of Santo Domingo, there are several copper lodes, as well as in Pedro Brand. In the province of La Vega, at Cotuy and Bongo, there is also copper-bearing quartz. These deposits have not been developed, but they seem to be rich. There is also a copper mine in Asiento Frio, district of Monte Cristi, and one in the Rio Arriba section of the province of Azua.

Servia.—The mines of the Compagnie Française des Mines de Bor produced 1900 metric tons of copper in 1908, an increase of 680 tons compared to 1907. The production would have been greater had it not been for labor difficulties.

South Africa.—The O'okiep and Nababeep mines of the Cape Copper Company in the year ended Aug. 31, 1908, made an output practically the same as in the preceding year. From the O'okiep mine 14,812 tons of 15.02 per cent. copper were mined, while from the Nababeep mine the output was 54,974 tons of 4.78-per cent. copper. The reserves of ore at the O'okiep mine are estimated at 10,000 tons of 20-per cent. copper ore

and at Nababeep 150,000 tons of 5-per cent. ore. From the Nababeep North mine 5069 tons of 5.83-per cent. copper were mined. At the O'okiep smelting works 25,586 tons of ore were treated, the matte produced averaging 51.2 per cent. copper. At the Nababeep works 51,872 tons were smelted, the matte obtained averaging 49.7 per cent. copper.

Spain.—The Rio Tinto in 1908 extracted 604,275 tons of pyrites for shipment and 1,115,610 for local treatment, a total of 1,719,885 tons. The quantity of cupreous ore invoiced to consumers in 1908 was 589,815

RIO TINTO STATISTICS.

	Pyrites Extracted.				Pyrites Consumed.		Copper Produced at Mines.
	For Shipment.	For local treatment.	Total.	Average Copper contents.	Tons 2240 lb.	Average Copper contents.	Tons 2240 lb.
				p.c.			
1876	180,962	159,196	340,158	1.5	158,597	1.5	946
1877	251,360	520,391	771,751	2.375	211,487	2.	2,495
1878	218,818	652,289	871,107	2.78	211,403	2.18	4,184
1879	243,241	663,359	906,600	2.78	236,849	2.45	7,179
1880	277,590	637,567	915,157	2.965	274,210	2.481	8,559
1881	249,098	743,949	993,047	2.75	256,827	2.347	9,466
1882	259,924	688,307	948,231	2.805	272,826	2.401	9,740
1883	313,291	786,682	1,099,973	2.956	288,104	2.387	12,295
1884	312,028	1,057,890	1,369,918	3.234	314,751	2.341	12,668
1885	406,772	944,694	1,351,466	3.102	354,501	2.270	14,593
1886	336,548	1,041,833	1,378,381	3.046	347,024	2.306	15,863
1887	362,796	819,642	1,182,438	3.047	385,842	2.283	17,813
1888	434,316	969,317	1,403,633	2.949	333,149	2.208	18,522
1889	389,943	824,380	1,214,323	2.554	395,081	2.595	18,708
1890	396,349	865,405	1,261,754	2.883	397,875	2.595	19,183
1891	464,027	972,060	1,436,087	2.649	432,532	2.651 1.309	21,227
1892	406,912	995,151	1,402,063	2.819	435,758	2.569 1.465	20,017
1893	477,656	854,346	1,332,002	2.996	469,339	2.659 1.544	20,887
1894	498,540	888,555	1,387,095	3.027	485,441	2.594 .968	20,606
1895	525,195	847,181	1,372,376	2.821	518,560	2.595 .986	20,762
1896	591,752	845,580	1,437,332	2.931	549,585	2.529 1.068	20,817
1897	575,733	812,293	1,388,026	2.810	582,540	2.595 .967	20,826
1898	644,518	820,862	1,465,380	2.852	618,110	2.600 1.023	20,426
1899	644,271	1,005,573	1,649,844	2.719	636,323	2.511 1.120	20,230
1900	704,803	1,189,701	1,894,504	2.744	665,967	2.553 1.187	21,120
1901	633,949	1,294,827	1,928,776	2.627	641,935	2.680 1.025	21,100
1902	627,967	1,237,322	1,865,289	2.517	595,092	2.342 1.495	21,659
1903	688,919	1,229,619	1,918,538	2.390	667,748	2.320 1.241	21,565
1904	672,344	1,276,475	1,948,819	2.340	663,744	2.105 .978	21,218
1905	627,336	1,202,768	1,830,104	2.363	660,724	2.182 1.124	19,530
1906	655,328	1,268,388	1,923,716	2.411	632,307	2.302 1.198	21,287
1907	641,858	1,265,090	1,906,948	2.417	607,944	2.112 1.048	21,251
1908	604,275	1,115,610	1,719,885	2.265	589,815	2.087 .952	24,256

tons against 607,944 tons in 1907, and 632,307 tons in 1906. Of the deliveries in 1908 Germany took 300,361 tons, the United Kingdom 118,156 tons, and the United States 171,298 tons. The deliveries of sulphur ore, which were 477,843 tons in 1906, and 619,814 tons in 1907, were 688,477 tons in 1908. The copper produced in 1908 by treatment at the mines amounted to 24,256 tons, and the fine copper in pyrites shipped was 9958 tons, a total of 34,214 tons. The following quantities were marketed: Refined and blister copper, 25,093 tons; copper in pyrites, 9821 tons; total, 34,914 tons. The profit on the sale of products in 1908 was £1,446,504. Satisfactory progress has been made with the opencasting of the San Dionisio lode. Two of the tunnels, through which the overburden will be removed, are now complete, and progress has been made with the third or lowest outlet. The extension of the South Lode opencast is proceeding well, and will effect a more economical extraction of the mineral in the near future. The output of mineral for 1908 was considerably reduced through the subsidences mentioned last year, but no further slips have occurred.

The Tharsis Sulphur and Copper Company in 1908 produced 440,886 tons of ore as compared with 457,692 tons in 1907. Shipment of copper precipitate amounted to 3292 tons in 1908 as compared with 2753 tons in 1907. Production of refined copper was 4427 tons, compared with 4410 tons in 1907.

Transvaal.—The most important occurrence of copper in the Northern Transvaal is at the Messina mine, where there are orebodies of considerable size, which contain a large proportion of high-grade copper glance. The orebodies occur as lenses in fissures in a gneissoid granite. Promising indications of copper have been discovered on other properties in the same district.

The Copper Markets in 1908.

New York.—The year opened with prices ruling at about 13¾ for Lake copper and 13½ for electrolytic. During January, there was an entire absence of demand on the part of domestic consumers. Europe, however, continued a large buyer and thus sustained prices, which at the end of the month had advanced to 13⅞c. for Lake and 13¾ for electrolytic. In February it developed that for the time being European buyers were satisfied, and an effort was made by American holders to induce fresh purchases through a reduction in prices, which gradually established quotations at the end of the month on a basis of 12⅞c. for Lake and 12⅝c. for electrolytic. These low prices had the desired effect, and the Chinese, who are always alert to any undue depression in the price of copper, also became buyers to a moderate extent. The demand thus created,

however, was not of long duration, and by the first part of March quotations stood at 12½c. for Lake and 12¼c. for electrolytic. Domestic consumers at that time took hold of the market, so that a steady improvement set in and prices advanced at the end of the month to 13⅛c. for Lake and 13c. for electrolytic.

Prices remained on about this level during April, when fairly large quantities were purchased by home consumers for nearby delivery to take care of orders booked by them. Interest was also evinced in purchases for more distant delivery. In May, fairly large quantities of copper were pressed on the market by Japanese holders, owing to a severe financial crisis then existing in the Far East, and tended to unsettle prices, which declined toward the middle of the month to 12¾c. for Lake and 12½c. for electrolytic. At the end of the month, some improvement was again noticeable, brought about by the more confident feeling shown in the stock exchanges of the world, and quotations were established at 12⅞c. for Lake and 12⅝c. for electrolytic. In June, activity continued to a moderate degree, and prices practically remained unchanged. By this time a slow but gradual improvement had manifested itself in the more important copper-consuming industries. For instance, the wire industry, which during January had stood at 25 per cent. of the normal capacity, showed an increase in February to 33 per cent., March 40 per cent., April 50 per cent. and May-June 60 per cent.

This improvement made still further progress in July and both domestic and foreign consumers were buyers of large quantities of copper. In consequence of these transactions, prices advanced during August to 13⅞c. for Lake and 13⅝c. for electrolytic. A disturbing influence toward the second half of that month was offerings of copper held by speculators, which lots, however, were readily picked up by consumers at a discount from prices quoted by first hands. After the great activity during the preceding two months, a reaction was to be expected, and occurred during September. First hands were, thanks to the large sales made, in a comfortable position, and did not press on the market. Sundry small lots of second-hand copper, however, were offered and made themselves disturbingly felt. Prices declined steadily, and at the end of September closed at 13½c. for Lake and 13¼c. for electrolytic.

The approaching elections had a restricting influence on buyers during the first half of October. The feeling as to the outcome of the elections became more hopeful during the second half of October and, influenced by a cheerful tone on the stock exchange, which inspired confidence, buyers took hold of the market in earnest. Enormous quantities of copper were purchased during the last 10 days of October and

immediately after the election of Mr. Taft, so that prices advanced by leaps and bounds and on November 10 reached a basis of 14¾c. for Lake and 14⅝c. for electrolytic. The activity of the market, however, received a check during the second half of November, when the demand subsided and when copper bought by speculators during the advance was realized upon. While necessarily under these conditions prices had to decline somewhat from the high level, many of the first hands showed no disposition to enter the market, having practically disposed of all the copper that they expected to have available up to the end of the year. The undertone, therefore, remained sound and prices steady at a level of 14¼@14½c. for Lake and 14@14¼c. for electrolytic. These conditions lasted nearly to the end of December, the holiday season always exercising a check upon business, but at the end the market started upward, the year closing with 14½c. for Lake and 14¼c for electrolytic.

AVERAGE PRICE OF LAKE COPPER PER POUND IN NEW YORK.

Year.	Jan.	Feb.	Mar.	April.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.
1900	16.33	16.08	16.55	16.94	16.55	16.00	16.16	16.58	16.69	16.64	16.80	16.88	16.52
1901	16.77	16.90	16.94	16.94	16.94	16.90	16.51	16.50	16.52	16.60	16.60	14.39	16.55
1902	11.322	12.378	12.188	11.986	12.226	12.360	11.923	11.649	11.760	11.722	11.533	11.599	11.887
1903	12.361	12.901	14.752	14.642	14.618	14.212	13.341	13.159	13.345	12.954	12.813	12.084	13.417
1904	12.533	12.245	12.551	13.120	13.000	12.399	01.505	12.468	12.620	13.118	14.456	14.849	12.990
1905	15.128	15.136	15.250	15.045	14.820	14.813	15.005	15.725	15.978	16.332	16.758	18.398	15.699
1906	18.419	18.116	18.641	18.688	18.724	18.719	18.585	18.706	19.328	21.722	22.398	23.350	19.616
1907	24.825	25.236	25.560	25.260	25.072	24.140	21.923	19.255	16.047	13.551	13.870	13.393	20.661
1908	13.901	13.098	12.875	12.928	12.788	12.877	12.933	13.639	13.600	13.646	14.386	14.411	13.424

AVERAGE PRICE OF ELECTROLYTIC COPPER PER POUND IN NEW YORK.

Year.	Jan.	Feb	Mar.	April.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year
	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.
1900	15.58	15.78	16.29	16.76	16.34	15.75	15.97	16.35	16.44	16.37	16.40	16.31	16.19
1901	16.25	16.38	16.42	16.43	16.41	16.38	16.31	16.25	16.25	16.25	16.22	13.82	16.11
1902	11.053	12.173	11.882	11.618	11.851	12.110	11.771	11.404	11.480	11.449	11.288	11.430	11.626
1903	12.159	12.778	14.416	14.454	14.435	13.942	13.094	12.962	13.205	12.801	12.617	11.952	13.235
1904	12.410	12.063	12.299	12.923	12.758	12.269	12.380	12.343	12.495	12.993	14.284	14.661	12.823
1905	15.008	15.008	15.125	14.920	14.627	14.673	14.888	15.664	15.965	16.279	16.599	18.328	15.590
1906	18.310	17.869	18.391	18.375	18.457	18.442	18.190	18.380	19.033	21.203	21.833	22.885	19.278
1907	24.404	24.869	25.065	24.224	24.048	22.665	21.130	18.356	15.565	13.169	13.391	13.163	20.004
1908	13.726	12.905	12.704	12.743	12.598	12.675	12.702	13.462	13.388	13.354	14.130	14.111	13.208

London.—January disclosed an increase of 1922 tons in the visible supply, bringing the total up to 19,710 tons. Standard copper, which had closed in December at £60¾ for cash warrants, opened with a rise of 5s., advancing to £62½, with three months' warrants touching £63¼. Sales were largely covered with American supplies, whereby the American producers were relieved of substantial quantities. The closing days of the month witnessed restricted business and a gradual decline to £61⅛ for cash warrants, and £61¾ for three months.

In February the market became weak in response to unpromising advices from the United States, and values declined to £60⅜ cash, and £61 forward. Heavy sales followed, but an almost uninterrupted decline carried values down until on Feb. 19 three months' warrants touched £56⅝. Much of the selling on the London market was understood to be for American account, and was further prompted by grave apprehensions of trouble in the British shipbuilding and engineering industries. The closing prices of Standard were £58 for cash warrants, and £58⅛ for three months. In March fluctuations were within narrow limits. On March 12 some revival was manifest, and the price of three months' warrants advanced from £58⅛ on March 11 to £60 on March 16. The fourth week of the month witnessed a large speculative business, but the consumptive trade was not so encouraging. The closing prices were £59⅜ for cash warrants, and £60¼@£60⅜ for three months.

April opened with a flat market, and by the 10th prices had fallen to £57¾ cash, and £58¼ three months. There were rumors of wholesale reselling for Chinese account, some unshipped orders being certainly countermanded. The next feature was a recovery induced by covering of short sales, and a concerted buying movement which coincided with a bull campaign in copper mining shares, whereby values were carried up to £58⅝ and £59 respectively on April 16. The month closed at £57½ for cash warrants, and £58⅛ for three months. May started with a fall of 10s. per ton, caused mainly by unfavorable statistics; but at the end of two days the selling pressure had subsided and a recovery of 17s. 6d. was recorded, due to speculative orders. Toward the middle of the month a well-sustained bull movement was initiated in Wall Street, which broadened so as to attract speculators toward the metal. Brokers —undeterred by actual conditions of trade—encouraged their clients to invest for the rise, and a large volume of business resulted. The expanded demand was met by heavy selling, supposed to be for American account. Cash warrants advanced from £56¾ on May 12 to £59 on May 19. Meanwhile producers were alert to take advantage of the improvement, and accepted in many cases prices which were well below the Standard parity, Japanese brands being prominent after a long period of inactivity. The consumptive inquiry meanwhile was far from satisfactory, manufacturers being unanimous in bewailing the scarcity of orders. Prices gave way at the first sign of reaction in Wall Street, and gradually fell to £56⅞ at which cash warrants changed hands on May 27. The closing figures were £57⅝ for cash, and £58¼ for three months. June was remarkable for the strength displayed by the market throughout the month in face of unfavorable statistics, poor trade, heavy offerings from Japan, unsettled and generally languid stock and share

markets. There was sustained buying by speculative investors. Leading producers were thus able to market their copper without conceding any substantial reductions in price. The result was that fluctuations were very narrow; the three months' price ranging from £58¼ at the opening, up to £59¼ on June 11, and thereafter by easy stages down to £57 on June 29. Closing prices were £56¾ cash, and £57⅜ for three months.

In July, prospects at the opening were not encouraging, trade being restricted and demand readily met by dealers. Statistics disclosed an increase of 6553 tons during the previous month, and stocks in private hands were heavy. Further shipments of the metal from the United States and from Japan promised to be heavy, and consumers were naturally reluctant to buy more than they actually required. The leading American producers had meanwhile stood firm and now began to find a better home demand, and the London market improved in consequence. Three months' warrants, which had opened at £57⅛, changed hands in considerable volume at £58⅝ on July 9. On July 20 consumers began to buy freely after a long period of abstention, and an industrial revival was manifest, particularly in electrical branches. Three months' warrants changed hands on July 24 at £60, a figure which had long been regarded as a remote possibility. American manufacturers bought largely at 13⅝@13¾c., apparently without satisfying their requirements even at the higher figure. The most noticeable feature in the Standard market was the absorbing power of the speculative demand. Bull accounts were freely liquidated, and bear selling assumed large proportions, while large quantities were marketed from Japan, Chile, China and European producers; but the speculative force absorbed all that was offered, and on the last day of the month £61 was paid for three months' warrants, final quotations being £60 for cash warrants, and £60¾ for three months. In August the market displayed considerable buoyancy in spite of an increase of 4423 tons in the visible supply during the previous month, but fluctuations were within narrow limits, the highest being touched on Aug. 10, when three months' warrants commanded £63, and the lowest on Aug. 25, when £60⅜ was accepted; the month closed with cash warrants at £61⅛@61¼, and three months at £61⅜. In September improved conditions in the iron and steel trades inspired confidence, and prices rose on Sept. 8 to £61¼ for cash warrants, and £62¼ for three months. At this point the improvement was checked by the leading American producers unexpectedly reducing their price for electrolytic quality by £1 per ton. By Sept. 22 prices of Standard had fallen to £59⅛ and £60 respectively. The close was quiet, cash warrants being held for £59⅞, and three months' for £60¾.

October found the market sensitive at the outset, the continued increase in the visible supply inspiring caution. American consumers bought but sparingly, and the American producers found Europe their best outlet, particularly for electrolytic quality. In two days the London price dropped 12s. 6d. per ton. The lowest was touched on Oct. 9, when cash warrants stood at £59⅛ and three months' at £60. On Oct. 21 a recovery set in, and the price advanced £2 in two days. Toward the end of the month there were persistent reports of a powerful syndicate operating in support of the market. The rumor was not visibly substantiated, but it coincided with successive advances in the American price of the metal and a general improvement in the European demand. The month closed with £61¾ for cash warrants, and £62⅝ for three months. November opened with a moderate improvement in the London market where, by Nov. 9, cash warrants commanded £65⅛, and three months' £66¼. The speculative element was responsible for some reaction, particularly toward the middle of the month. The lowest was touched on Nov. 23, when cash warrants changed hands at £62⅝, and three months at £63⅝. Closing values were £63⅞ for cash warrants, and £64¼ for three months. December was clouded at the outset by sinister political developments, as also by instability in the stock markets. There was considerable liquidation of holdings—chiefly by Continental sellers—early in the month, and prices fell back in consequence. The lowest was touched on Dec. 8, when £62⅛ was accepted for cash warrants, and £63⅛ for three months. Speculative enterprise raised prices to £63⅝ cash, and £64⅜ for three months', on Dec. 10; but during the next few days they fell £1. The closing quotation was £63¾ for cash Standard.

AVERAGE PRICE OF STANDARD COPPER (G. M. B.'s) IN LONDON.
(In pounds sterling per ton of 2240 lb.)

Year.	Jan.	Feb.	Mar.	April.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
1901.....	71.78	71.17	69.54	69.61	69.60	68.83	67.60	66.34	65.97	64.11	64.51	52.34	66.79
1902.....	48.43	55.16	53.39	52.79	54.03	53.93	52.89	51.96	52.68	52.18	51.03	50.95	52.46
1903.....	53.52	57.34	63.85	61.72	61.73	57.30	56.64	58.44	56.82	55.60	56.30	56.36	57.97
1904.....	57.500	56.500	57.321	58.247	57.321	56.398	57.256	59.952	57.645	60.012	65.085	60.375	58.884
1905.....	68.262	67.963	68.174	67.017	64.875	65.881	66.867	69.830	69.667	71.406	74.727	78.993	69.465
1906.....	78.869	78.147	81.111	84.793	84.867	83.994	81.167	83.864	87.831	97.269	100.270	105.226	87.282
1907.....	106.739	107.356	106.594	98.625	102.375	97.272	95.016	79.679	68.375	60.717	61.226	60 113	87.007
1908.....	62.386	58.786	58.761	58.331	57.387	57.842	57.989	60.500	60.338	60.139	63.417	62.943	59.902

SUMMARY OF THE LITERATURE OF 1908 ON THE METALLURGY OF COPPER.

BY L. S. AUSTIN.

Reverberatory Smelting.

Reverberatory Smelting at Anaconda.[1]—These articles give detailed plan, elevation and sections of one of the large reverberatories at Ana-

[1] *Eng. and Min. Journ.*, LXXXV, 1189, 1234.

conda (described in THE MINERAL INDUSTRY, XV, 275) and should be studied for the actual construction of such a furnace. The solid bottom of the furnace is composed of 4 ft. in depth of slag poured in molten condition. All parts of the furnace except the 26-in. sand bottom exposed to the molten charge and to the intense heat are of silica brick, while the backing or filling is of common brick, both laid in a mortar of ground silica. It will be noted that below the door sill the walls are stepped, becoming thicker downward as particularly shown by the sections.

The coal is charged at four openings through the roof of the fire-box, which openings are ordinarily kept covered by brick lids. There are two circular stoke holes, 12 in. in diameter at the fire-box wall, by which it is possible to note that the coal is kept at the level of the top of the fire-bridge, 3000 lb. of coal being dropped on an average every 40 minutes. In the roof, above the fire-bridge, a row of checker-holes or ports 3x3 in. is left for the admission of secondary air, these holes being opened or closed to give just the required proportion. The fires are grated on an average every four hours. The clinkers are broken up by long 2-in. bars, and are worked downwards through the grate, leaving only enough for the necessary depth of clinker bed. This is hot heavy work for the men. The time for grating is generally 35 to 45 minutes. A good deal of the finer, partially burned coal falls into the ash-pit, whose bottom is a gutter carrying a constant stream of water. This sweeps away the ashes and clinkers, the larger piles being separated on a grizzley. These large pieces are removed as they accumulate. The finer material is borne away by the water to the washing plant where as much as 10 per cent. of unburned coal is saved by concentrating, and this material is later briquetted with fine ore and with pond-slimes.

So-called "floaters," silicious aggregates or lumps of unfused or baked charge, float upon the bath when the rest of the charge has melted down, and, when near the front of the furnace, may be pulled out with hooks. They appear especially after freshly claying or patching the furnace, and also if the charge is too high in sulphur.

Skimming, in which the slag readily flows from the furnace, is controlled by the use of a rabble with a blade of 5x9 in. and a handle 12 ft. long; as they get too hot before the "skimming" is completed, two or three have to be used. When the matte begins to show, the slag having run low, the flow is shut off by putting a rabble (with a blade 6x16 in.) across the channel and building sand as a dam against it. The rabble is then removed, the door placed on the sand dam and clayed up.

Matte is tapped as needed by the converters. In case it accumulates too much it is tapped off to ladles, or, if these are not available, to a hole in the ground, to be subsequently broken up and sent to the

blast-furnace as "matte chips." At one side of the furnace are two matte tap-holes, one 21 ft. 3 in. from the bridge, the other 18 ft. 9 in. beyond. The flow from these unites into a common slag-launder of cast-iron which leads the flow outside the building above the ladle-tracks to the ladles. The matte tap-jackets are of copper, 24 in. square by 4½ in. thick, and will last five months. It takes 15 minutes to tap two ladles each of 7 to 10 tons. Every ladle is sampled, the sample being taken from the flow in the runner.

The ladles are of wrought steel, and have a thin coating or lining of silicious ore mixed with clayey slimes from the settling ponds. A freshly lined ladle before use is dried out with a wood fire, and may receive either reverberatory or blast-furnace matte. Once used, such a ladle is said to be "lined out." It is impenetrable to water. When one ladle is filled the next one is moved into place. The small amount of spilling is but little regarded, and is later gathered up to go to the blast-furnace.

To stop the matte tap-hole the dolly with its conical clay plug is pressed into the hole, this pressure being supplemented by striking the end of the dolly with a hammer. A pointed tapping bar is then driven through the fresh fire clay, its point reaching into the matte. The rod becomes somewhat eaten or corroded by the matte and is driven in a little way from time to time. If it does not seem, upon driving, to be free, the attempt is made to drive it until it does feel free. If too far gone for this, the tapping bar is pulled out with a claw, and a chisel-pointed bar is used to drill out the hole. If this does not serve an air drill is used, and, when it gets into soft material, a tapping bar is again inserted, this being driven until the matte is reached.

An autographic register is kept of the draft both at the bridge and at the outlet-flue. Thus at the bridge it will amount to 0.85 in. of water, at the flue 1.7 to as high (in winter) as 2 in. The autographic record shows that the dropping of the charge and of coal lessens the draft.

A table is given of the typical performances of the seven furnaces for 24 hours whose average per furnace is as follows: Coal used, 59.6 tons. Total charge smelted, 256.4 tons. Coal per ton of charge, 23.2 per cent. Weight of a charge, 14.6 tons. The average content at the calcines was Cu, 8 per cent.; SiO_3, 29.5; S, 7.3; CaO, 2.6; and FeO, 36.9. The slag produced contained Cu, 0.35 per cent.; SiO_2, 38.6; CaO, 3.5; and FeO, 41.9. The matte contained 46 per cent. of copper.

Starting up a reverberatory furnace: Scrap-wood is at first burned on the grates, and the fire is gradually increased by additions of coal. As the roof rises, as determined by careful measurement, the tie-rods are slacked, so that it rises no more than 2 in. as it heats up. If the roof

becomes distorted the high points are kept down by using struts set between the roof and the heavy structure above. After two days' firing the first charge is dropped a little at a time and is followed by others until the furnace is full and in working condition, as shown in the graphic curves, Fig. 1. It usually takes a week to get a furnace in full working condition, and the first matte is not tapped until about 250 tons of it have accumulated in the furnace.

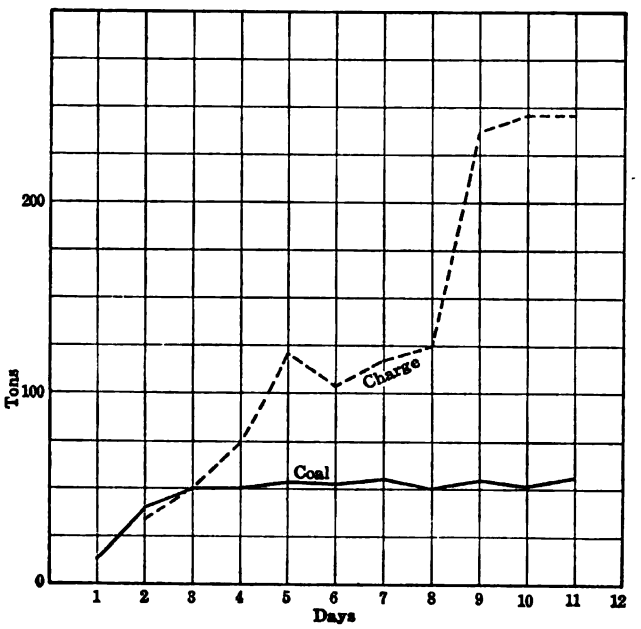


FIG. 1. COAL AND CHARGE IN STARTING UP A REVERBERATORY FURNACE.

The bottom of one of these reverberatories is practically indestructible; the roof and bridge need rebuilding every eight or nine months, and every two years the walls and roof are reconstructed throughout.

Claying, repairing or fettling: This may be done every four to six weeks and the time to do so may be known by the conker plates at the fire-bridge getting red hot. To do it the matte is allowed to accumulate and the slag is closely skimmed. The matte is mostly tapped off (about 20 ladlesful of 7 to 10 tons each) as quickly as possible, this taking several hours to complete. The furnace being now tapped dry, the two or three doors on each side nearest the fire-bridge are removed, and about 20 tons of sand containing 95 per cent. of silica are added, this being thrown against the fire-bridge and against the side-walls so as to bank them up. The furnace cools down a good deal during this performance.

A screen analysis of the McDougal calcines smelted show 10 per cent. on 10-mesh; between 10 and 30-mesh, 25 per cent.; between 30 and 80-mesh, 30 per cent.; finer than 80-mesh, 35 per cent. From a monthly report of the reverberatories we gather: (1) The slag produced equals 72 per cent. of the charge. (2) Sulphur volatilized 40 per cent. (3) Copper lost (principally in the slag) 6.8 per cent. On a six months' basis this has varied from 3 to 7 per cent., and 5 per cent. may be considered a fair average figure. (4) The ratio of concentration of ore into matte was 18.6 per cent., of 5.4 tons of charge into one of matte. It will be seen that the slag is nearly four times this quantity, and will, with the SO_2, oxygen and CO_2, make up the required 100 per cent.

The Garfield Works[1].—A large portion of the ore smelted at this plant comes from the Bingham district, Utah, and may be divided into two classes: (1) Crude pyrite carrying 2 to 4 per cent. copper, 20 to 40 sulphur, 20 to 50 iron, 10 to 30 silica, together with 1 to 4 oz. per ton in silver and $0.50 to $4 per ton in gold. (2) Concentrates mostly finer than 20-mesh size and carrying 20 to 25 per cent. copper, 20 to 30 sulphur, 25 silica, together with $20 per ton in gold and silver. The ores from other camps comprise gold-copper pyrite concentrates from Newhouse, Utah, and silicious gold ores from Southern Utah and Nevada.

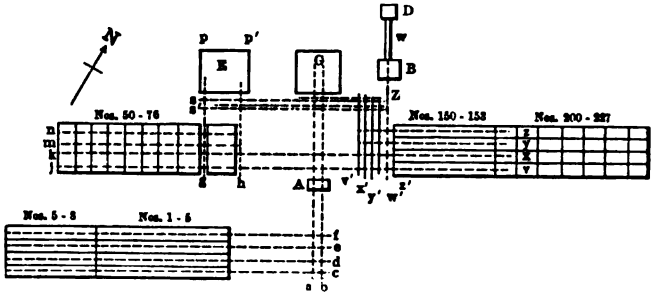


FIG. 2. CONVEYER AND BIN SYSTEM AT THE GARFIELD WORKS.

Fig. 2 is a plan of conveyer, bin-system and sampling-mills of the plant. The numbered rectangles are the ore-bins, *E* is the sampling mill for oxidized lump ore, *F* and *G* are the sampling-mills for sulphide ore, and the lettered broken lines represent belt-conveyers 16 to 20 in. wide by which the ore is sent to and from the sampling-mills and to the various storage-bins.

The arriving ore may be divided into three classes, oxidized ore, sulphide ore, and concentrates. The oxidized ore, which is more or less lumpy, is shoveled or dropped from the railroad cars into the receiving

[1] *Mines and Minerals*, XXVIII, 305. *The Mineral Industry*, XVI, 352.

bins Nos. 50-76 over the belts *j* and *k*. These bins have movable plank bottoms by which the ore can be drawn off upon the belts *j* and *k*, which deliver it to the belts *g* and *h*, these taking it to the sampling mill *E*. When sampled it is spouted upon the conveying belts *s* and *l'*, which take it to *y'* and *w'*, and these by belts *x* and *v* to the storage bins, Nos. 150-153.

The sulphide ore consists of lumps and fines. This is dropped from the car standing above bins Nos. 50-76, which are over the belts *m* and *n*. From the bins the ore is drawn off to *m* and *n*, which deliver to *a* and *b* to be sent to the sulphide sampling mill *G*. The ore is sampled, and at the same time, the lumps are screened out. The lump-ore is shot from the mill upon the conveying belts *s*, which take it, via *y* and *w*, and *z* and *y*, to the bedding bins Nos. 150-153. The fine sulphide ore goes by belts *a* and *b* to the bins Nos. 5-8, or to bins Nos. 1-5.

Concentrates are valued by receiving from the car each tenth shovelful for a sample, while the principal is dropped through the hopper *A* upon the conveying belts *a* and *b*, thence going to the bins Nos. 5-8 and Nos. 1-5.

Finally roasted ore from the Huntington-Heberlein plant is crushed in the 24x36 in. rock breaker *D*, raised by elevator to the bin *B*, and from this drawn off upon the conveyer *z* in order to go to storage at bins Nos. 150-153.

Fig. 3 is a cross-section of one of the four bins Nos. 150-153 showing the method of bedding and reclaiming for use.

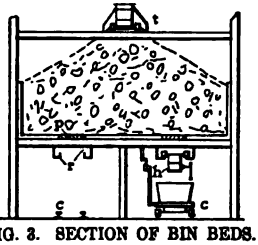


FIG. 3. SECTION OF BIN BEDS.

Bins Nos. 200-227 receive limestone, iron ore and coke for use at the blast-furnaces. Both these and Nos. 150-153 are of the cross-section shown at Fig. 3, so that the charge-car can be run under any one of them to get its load for the blast-furnaces.

At *t* is shown any one of the troughed conveying belts *z*, *y*, *x*, or *v*, of Fig. 2, and the traveling-tripper by which ore is distributed from end to end of the bin in the same way as is done at Cananea (THE MINERAL INDUSTRY, XVI, 356). This insures an even quality of ore. To reclaim

or withdraw it for consumption, a charge-car *c* is brought beneath the bins and receives the ore from a hopper *h* which is capable of being moved to the drawing-off point. Ore is allowed to drop into this hopper by removing loose planks which form part of the bottom of the bin. Some shoveling has to be done, but most of the ore can be drawn off freely. The removable hopper is suspended by the rails *r* as shown in the engraving.

The fine sulphide-ore is withdrawn from the bins Nos. 5-8 and Nos. 1-5, and is taken by tracks to the upper part of the McDougal roaster-house, a cross-section of which is shown at Fig. 4. It is discharged into

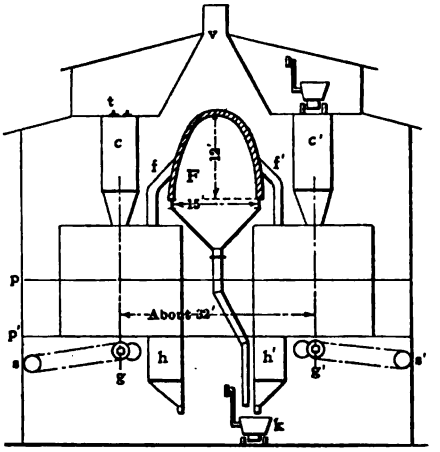


FIG. 4. SECTION MC DOUGAL ROASTER HOUSE

the hoppers *C* for feeding to McDougal roasters, 18 ft. in diameter, of which there is a double row, eight in a row, or 16 in all. The smoke from the roasters is taken by the branch flues *f* and *f'* to the catenary brick-lined flue *F*, 15 ft. wide by 12 ft. high, furnished with a hopper-bottom by which the accumulating dust is periodically withdrawn. The discharge from the roasters enters the hoppers *h* and *h'*, one to each roaster, and the ore is taken away to the reverberatory-furnace by charge-cars *k*.

Fig. 5 is a transverse-section of the blast-furnace building, showing the arrangement of a furnace. The building has two floors, one 20 ft. 6 in. above the ground-floor and one 32 ft. above it. The upper, *f*, is the charge-floor where the charge-cars *c* enter and discharge into troughs *h* at the feed-doors of the furnaces. When it is wished to charge the furnace the air-cylinders *a* are operated to tip the contents of the trough *e* into the furnace. The ore falls a distance of at least 12 ft., and the

surface of it is maintained lower than the working floor *f'*. This fall tends to pack unduly the charge and to increase the quantity of flue-dust made. The middle floor is intended to be used to bar off accretions which form in the furnace-shaft, but, as a matter of fact, the sulphur fumes at this point are so excessive that but little advantage can be obtained from its use. The draft from the furnaces is received in the balloon-flue *F* and the accumulating flue-dust is withdrawn into cars *k*. The ground-floor of the furnace-building is higher than that of the converter-building adjoining, so that matte can easily be drawn from the forehearth into

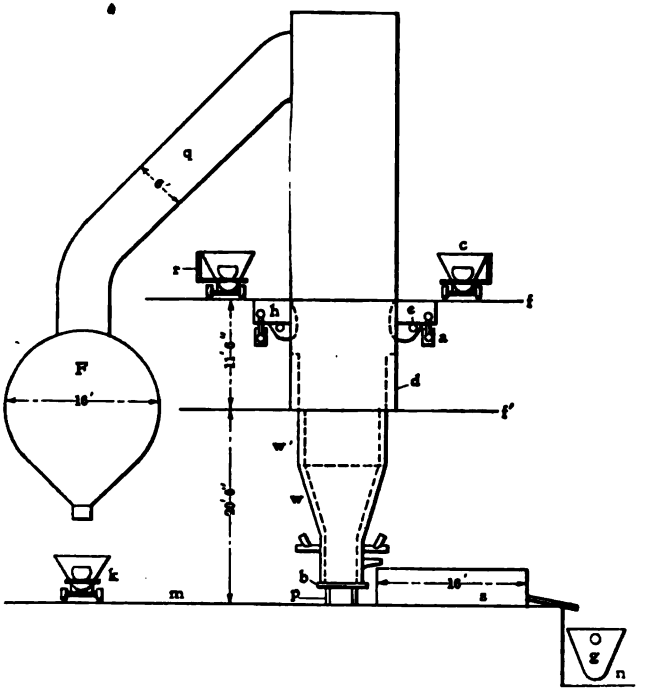


FIG. 5. SECTION OF MATHEWSON BLAST FURNACE.

ladles *g*. This matte contains 20 per cent. copper, and is of too low grade for converting; it is accordingly transferred by the 10-ton ladle *g* to a casting-machine, and there cast into blocks, which are resmelted from time to time in a separate furnace to produce a 30-per cent. copper matte.

For each furnace there is a charging-train of three cars, hauled by an electric locomotive. Two of the cars hold each a 6000-lb. charge of ore, flux and slag; the third car holds 1000 lb. of coke constituting 8.3 per

cent. of the charge, though as low as 5 per cent, has been used. The charge-car is fitted with a scale so that it is weighed as it is drawn off from the bins.

Two blast-furnaces, 4x20 ft. at the tuyeres, put through 325 tons each of charge daily. Each furnace takes per shift on the charge-floor, two feeders, two helpers, two train men, and on the tapping floor a tapper and helper, or in all eight men per shift.

Pot-roasting of part of the copper ores is here performed. There are 26 blowing-stands each holding a pot 8 ft. 8 in. in diameter by 4 ft. 2 in. high, the pot being in two segments, a bottom and a top ring, the two being bolted together. The pots stand in one line and are commanded by a gantry-crane by which they are handled and charged. A rotary blower, driven by an 85-h.p. motor, furnishes blast for the pots. When in operation each one is covered with a sheet-steel hood, and the smoke is led away by a horizontal branch-pipe to the sheet-steel main.

To charge an empty pot, a layer of ashes 1 in. thick is scattered over the false-bottom, 2000 to 3000 lb. of hot roasted ore from the McDougal roasters is next added, and the blast is started. Next enough wet roasted ore is added to fill the pot, making the complete charge five to six tons. The hood is put on by the crane, and the ore blown for eight or nine hours. On completion of the sintering and roasting, the hood is removed, the pot and its charge lifted by the crane, carried to the end of the yard, and dumped on a concrete platform. Men, with the occasional aid of the crane, break up and feed to a 24x36 in. crusher the larger lumps of sintered ore, crushing it to 6 in. size. The product of the pots yields from 5 to 10 per cent. of flue-dust.

The reverberatory furnaces are of the large Washoe type, described in THE MINERAL INDUSTRY, XV, 275, and are 19x112 ft. hearth-dimensions. The fire-box is partitioned into two so that either end may be grated without disturbing the other. The grates have forced blast furnished by a No. 10 Sturtevant fan. As at Anaconda, each furnace has two Stirling waste-heat boilers of 350 h.p. each, which are by-passed for repairs. A furnace smelts 180 to 225 tons of charge with a consumption of 40 to 50 tons of Wyoming coal.

Much trouble has come from the poor draft, and from the fact, that in consequence, not enough fuel can be burned to do rapid melting. It sometimes takes two shifts to remelt slag which has set near the front or tapping end. An examination of the elaborate flue-system, as given in the article, seems to indicate that the trouble is due to the quick turns in the flues, to the great cooling of the gases which come away from the blast furnaces, and to the obstruction due to the gases passing through the waste-heat boilers.

The slag carries 40 per cent. silica; the matte contains 40 per cent. of copper, and is tapped into the 10-ton ladles from the fullest furnace. The furnace is repaired once a month with crushed quartz, after tapping the furnace dry. For each shift are employed one skimmer, one helper, one fireman, two grate-cleaners, and one ash-man, or a total of six per shift.

Converter slag is thus treated: The slag is poured from the converters into 8-ton wheeled pots. These are taken outside the building and poured into molds set on cradles. When the slag has cooled in the molds it is tipped by the cradles into bins set below the molds, and from the bins the slag is drawn off into railroad cars, and charged to the blast furnaces.

The flue-system: The blast-furnace gases pass through 450 ft. of balloon-shaped flue *F*, Fig. 5. This joins a 10-ft. balloon-flue from the converter, the two delivering to an east-west catenary flue 24 ft. wide by 20 ft. high, having a steel hoppered-bottom. The gases from the McDougal roasters, delivered to the catenary flue *F*, Fig. 4, enter another catenary flue. These two flues unite in the opposite compartments, continuing as a flat-roofed flue, 350 ft. long and having each of the compartments 21 ft. wide. Parallel and near the main flue is a catenary flue of equal length, 28 ft. wide by 18 ft. high, which receives all the reverberatory gases. These two deliver into a damper-tower from which leads three flues 50 ft. long. Each of these is 24 ft. wide and 20 ft. high. By means of three sheet-steel dampers in the tower and one in each flue, the three kinds of gases (blast furnaces, roaster and reverberatory) can be either kept separate until they reach the stack, or two or more can be made to enter one or more of the three final flues. The gases now enter another brick tower, then (under 240 ft. of track) through three double flat-roofed flues to a third tower. From this tower the gases pass by three parallel brick catenary flues, which extend 1700 ft. to the main stack, where there are more steel dampers. The roofs of the flat-roofed flues are made by springing flat arches, I-beam to I-beam, set transversely to the flue.

Catenary flues of the large size mentioned, if they become distorted for any reason, are liable to fall. The first of them were made 9 in. thick, but were blown over by a high wind. They were rebuilt 13 in. thick and are now comparatively stable. The main stack, with its base 200 ft. above the furnace-floor, is 300 ft. high and 30 ft. in diameter.

Brackish water for the condensers and for jacket-water is pumped from wells near the lake. The fresh-water supply is brought from Riter, 12 miles away, in a 40-in. red-wood pipe laid on the surface.

For the blast-furnaces there are three Connersville blowers, 300 cu.ft. displacement per revolution, run at 70 r.p.m., and driven by Allis-Chalmers tandem-compound Corliss engines. There is a separate 42-in. main for each furnace.

This side-hill plant was expected to save in its first cost, but its expense has been enormous. It has cost up to the present, $5,000,000 on an estimated capacity of 3500 tons daily, or more than $1400 per ton daily or nearly $5 per ton per year, and yet the plant is not entirely economical or satisfactory in operation.

Regenerative Reverberatory Furnace.—The Peyton Chemical Company, Oakland, Cal., has constructed an oil-burning furnace, making use of the regenerative principle for heating the primary air.[1] It consists of a rectangular hearth, 17x60 ft., and of 1020 sq.ft. area, fired by six ¼-in. oil-jets at either end, the oil being injected and atomized by air from a Connersville positive-blast blower at a pressure of 7 to 10 lb. per sq.in., each burner using 50 cu.ft. of free air per min.

From each end of the rectangular hearth there leads away a flue 6 ft. wide by 7 ft. high by 150 ft. long and having six right-angle turns on the way to the stack, the flues uniting into one and then branching into the stack. At the branch was set a reversing valve, Fig. 6.

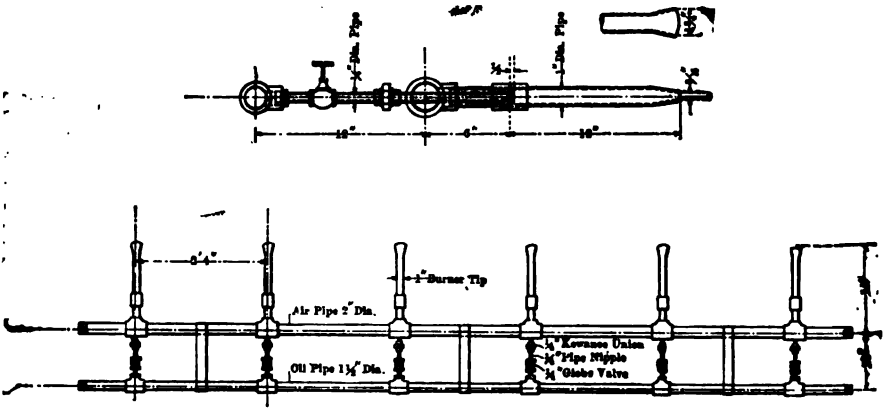


FIG. 6. DETAILS OF CRUDE OIL BURNER AT WORKS OF THE PEYTON CHEMICAL COMPANY.

The valve consists of a cast hub with three wings of ⅜-in. steel-plate bound with 6-in. angle-iron. The wings are so spaced that in reversing the furnace one flue is opened to the air, and the other closed by one movement of the valve. The air is consequently drawn through the hot flue, which has just been in use, through the furnace, and by the other flue to the stack.

[1] *Eng. and Min. Journ.*, LXXXVI, 898.

Under the old conditions, when using a furnace 20x40 ft., the furnace handled 60 to 65 tons per day; after the adoption of the regenerative system the tonnage increased to an average of 90 tons daily, and it has smelted as much as 112 tons in that time. By regeneration two advantages are obtained. First, the entire furnace area is in use, and second, the temperature of the entering air being raised 400 to 1000 deg. C., the temperature of combustion is much higher, and consequently melting proceeds more rapidly. When using the more recent furnace, 17x60 ft., above described, 176 tons, consisting of 56 per cent. of hot calcines, 25 per cent. of cold calcines, and 15 per cent. of silicious ore, could be put through in 24 hours.

It would appear that this furnace could be arranged to do equally effective work, using powdered coal instead of oil. The chief drawback would be the accumulation of flue-dust and ashes from the coal. However, a flue could be cleaned out by running the other one sufficiently long, until the former was cooled and cleaned out.

The escaping gases have a temperature of 500 deg. C., so that waste-heat boilers could be installed to advantage. At the time of reversal the entering air has a temperature of 1000 deg. C., which drops during 1½ hours to 400 deg. C. It was computed that the temperature of combustion was, with heated air, 1733 deg. C. and with preheated air 1893 deg. C. The measured temperature agrees closely with these figures.

Blast-furnace Smelting.

A Down-draft Blast-furnace.—R. L. Lloyd describes a successful down-draft blast-furnace called the "Volcan furnace," which is operated near Santiago, Chile.[1] It consists of a blast-furnace having a closed-top forehearth with an escape hole or throat in the top for the escape of the blast. The breast, 14x16 in., between the blast-furnace and the forehearth is open, so that part of the blast escapes with a brilliant flame by the throat, part, in the usual manner, up the furnace-shaft.

The ore is chalcopyrite with a hematite gangue, low in silica and yielding a difficultly fusible slag. The blast, traversing the forehearth, keeps it so hot that the slag remains liquid, and is skimmed off by a skimming door at the front.

But little gas escapes by the shaft, and but little smoke is seen at the charge-floor. The gases formed are withdrawn around the furnace just below the feed floor. No flue dust is made. The furnace runs with a neutral atmosphere, and uses 14 per cent. of coke. The matte is tapped as usual at the bottom level of the forehearth. It carries 55 per cent. of copper.

[1] *Eng. and Min. Journ.*, LXXXV, 763.

Pyrite Smelting at Tilt Cove, Newfoundland.—F. S. Nichols has given particulars of the pyrite smelting at Tilt Cove in 1896.[1] Using four furnaces six days weekly, there were put through during 294 days, 40,133 tons of ore with cold blast and without the use of coke or other fuel. Coke was, however, used to line the forehearth, to smelt slag, and to start the furnaces. The coke thus used amounted to 806.5 tons for the year. The Tilt Cove ore is a low-grade pyrite of the following composition: Cu, 3.69 per cent.; S, 34.98; Fe, 37.06; SiO_2, 13.23; Al_2O_3, 2.47; CaO, 0.55; MgO, 0.0465; As, 0.0107; Sb, 0.116; Ni, 0.3927; Co, trace; O and loss, 7.218 per cent. Round furnaces having eight tuyeres were used, and treated coarsely crushed mine-ore. The forehearth had cast-iron plates and was lined with a mixture of clay and coke-dust. Five men were needed per furnace.

To start the furnace a cord-wood fire was made on the hearth and on it was dumped 500 lb. of coke. When this was burning brightly about a ton of ore was dumped in, and when hot, a blast of 4 oz. per sq.in. was started. The furnace was filled with charges to the feed-floor, the blast pressure being meanwhile increased to 12 oz. per sq. in. As long as the furnace kept up a continuous cracking and gave off dense volumes of smoke it worked well. The slag was basic, carrying 0.25 per cent. of copper; the matte contained 7.06 per cent. of copper, indicating a concentration of 2:1. Matte was occasionally made containing 18 per cent. of copper, but this was with higher-grade ore containing some quartz mixed with it. The cost for crushing, transporting the ore to the smelter for smelting, and for piling the matte for shipment, including labor and material, was $1.37 per ton.

Matte Smelting at Ingot, Cal.—A furnace 42x150 in. hearth dimensions was used, having 12 tuyeres to the side. There are three wrought-steel jackets to the side having a slight bosh only, and that above the tuyeres.[2] The average composition of the ore-charge during five months, including silicious ore added, was Cu, 2.81 per cent.; Fe, 11.6; CaO, 5.4; Zn, 14.3; S, 20.2; SiO_2, 18; Al_2O_3, 5.3; and BaO, 7.4 per cent. Both zinc and sulphur were burned off, so that the resulting slag contained FeO, 18.4 per cent.; CaO, 19.5; ZnO, 12; SiO_2, 28.6; Al_2O_3, 10; BaO, 9.8; and Cu, 0.2 per cent. The matte averaged 28.3 per cent. copper and 16 per cent. zinc, and there was a concentration of 10:1. There was gold and silver in the ore-charge but their quantity was not given; however, the silver loss was given at 5.7 per cent., and there was an actual gain of 39 per cent. of the gold, due to the presence of uncredited traces of it in some of the ores. The results show that 41 per cent. of the zinc entered the slag,

[1] *Eng. and Min. Journ.*, LXXXVI, 462.
[2] *Eng. and Min. Journ.*, LXXXV, 443.

11 per cent. the matte, and the residue, 48 per cent., was volatilized. Commenting on the above figures we note the large quantity of zinc both in matte and slag. In the latter especially the silica seems low for so large a proportion of difficultly fusible bases consisting of zinc and baryta.

To enable the furnace to run it was found necessary to use, not only a hot blast, but also a large proportion (7 to 7½ per cent.) of coke, and to run with a hot top. When the hot-blast pipe-stove was being repaired cold blast was tried for a while, but the coke had to be increased to 13½-14½ per cent. Naturally such increased coke resulted in decreased oxidation, lowering of the grade of the matte, the volatilizing of less of the sulphur and zinc, and the slagging of less iron.

Corrosion of Copper Blast-furnace Jackets.—Geo. B. Lee relates his experience in regard to the corrosive action of jacket-water upon steel blast-furnace water-jackets.[1] In 10 to 12 months the inner plates were reduced by corrosion to ⅛ in. from the original thickness of 0.5 in., while the boilers of the works were unaffected. The conclusion was that this was due to no fault of composition of the water but rather to electrolytic action. It was suggested that this arose from the electrolytic couples of the iron and its oxides, to the variation in the temperature of the water, or more likely to stray currents from some source. To remedy the trouble zinc plates may be hung within the jackets near the water-inlet to neutralize any free acid in the water. These plates may be 0.75 in. thick by 8 by 16 in. in area.

In a further discussion of the subject (*Bimonthly Bulletin*, A. I. M. E., Feb., 1909, 227) R. L. Lloyd stated that he was able at Cananea to overcome corrosion by the addition of commercial arsenic oxide to the water, putting into it 1 kg. weekly.

Dust-Flue.—At the works of the Southwest Smelting and Refining Company, Oro Grande, N. M., there has been installed a brick flue of catenary, or beehive, section 8 in. thick having a V-shaped bottom.[2] The flue is 16 ft. wide, 18 ft. total hight, 14 ft. above the ground and 200 ft. long. The 200-ton copper-matting furnaces of the company, 40x168 in. in size, yield about 5 per cent. of flue-dust, which is caught in this flue. Along the middle of the flue at the apex of the doubly sloping bottom is set an endless-chain scraping-conveyer. The return line of this conveyer is shielded to protect it from settling flue-dust, but the bottom line travels the length of the flue removing flue-dust as it accumulates at the apex of the V. The conveyer discharges to a bucket-elevator which puts it into a cylindrical steel tank. From this the dust is drawn off at

[1] *Bimonthly Bulletin*, No. 20, A. I. M. E., 179.
[2] *Mines and Minerals*, XXVIII, 436.

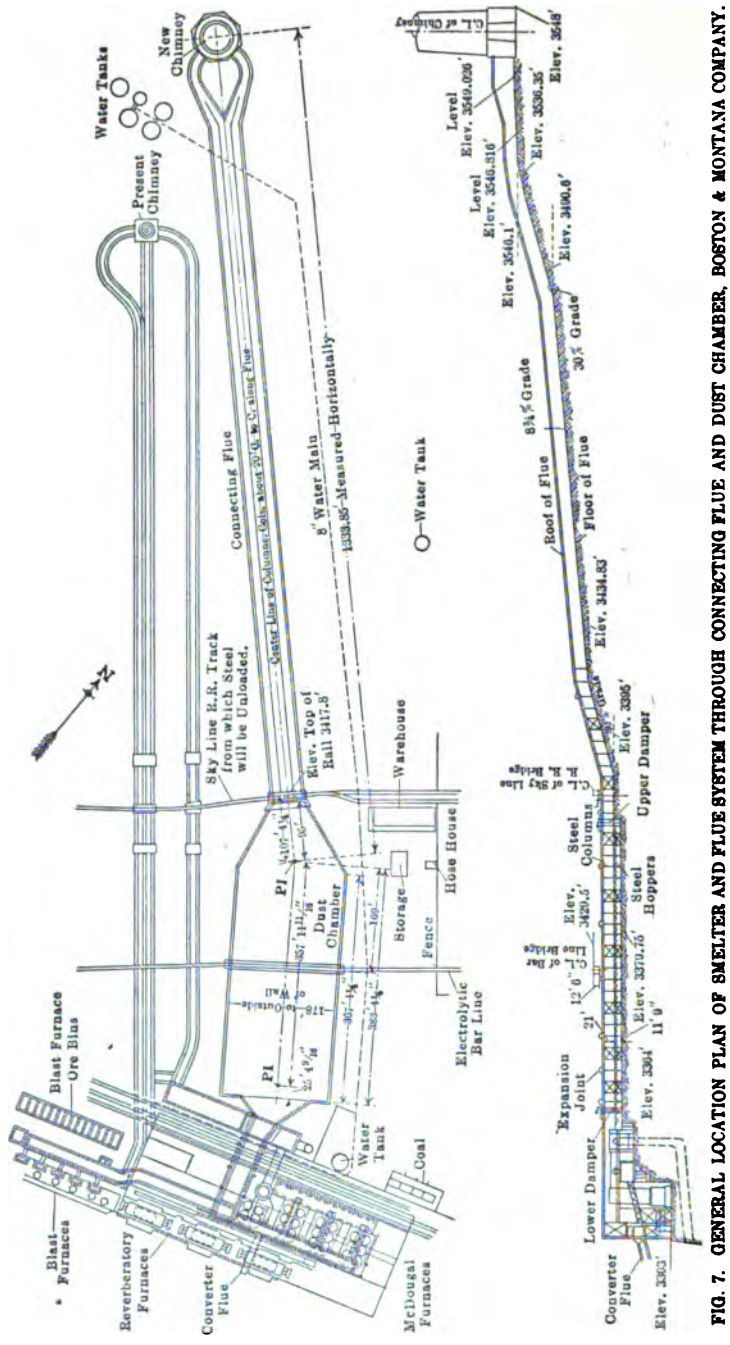


FIG. 7. GENERAL LOCATION PLAN OF SMELTER AND FLUE SYSTEM THROUGH CONNECTING FLUE AND DUST CHAMBER, BOSTON & MONTANA COMPANY.

the bottom as required and briquetted. A 5-h.p. motor drives this machinery. The system dispenses with the necessity of using a hopper-bottomed flue, and the work of removing the flue-dust. Such a flue would also be more expensive in first cost than the one here described.

Flues and Stack at Great Falls, Mont.[1]—The Boston & Montana Company topped its 506-ft. stack, Oct. 23, 1908. The smeltery is situated three miles from Great Falls, Mont., and has a capacity of 4500 tons daily. It was required to construct a system which would take care of 4,000,000 cu.ft. of gases per min., having a temperature of 315 deg. C.

General arrangement: In Fig. 7 is given a plan of the works, and a sectional elevation of the flue. Between the smelting works and the dust-chambers the main-flue is rectangular, 33x21 ft. in the clear, but having a hopper-bottom with a clear space below, so that the flue-dust can be withdrawn into cars. The dust-chamber is 176 ft. wide and 478 ft. long, narrowing as shown in the plan-view to the connecting flue which leads to the stack. This chamber is 21 ft. high, and has its bottom composed of hoppers, set 12 ft. above the ground. Beneath the flue are laid 21 parallel tracks for electrically operated tram-cars, which receive the flue-dust from the hoppers. It is divided longitudinally into two. Either side can be shut off by means of cast-iron dampers, opened or closed by motors, being thus accessible for repairs or for cleaning out. Fastened to the top of the I-beams composing the roof of the chamber is a heavy copper-wire net, from which are suspended long iron wires set 4 in. apart, forming a suspended net-work. The moving gases deposit flue-dust upon these wires. All the wires below the upper support are connected to shaker-beams operated by a motor. The dust, accumulating on the wires, is thus shaken off and falls into the hoppers which form the bottom of the flue. The connecting flue is rectangular, 48 ft. wide by 21 ft. high from the ground, and is divided into three, so that any one of them may be cut out and by-passed for cleaning out.

The chimney has an octagonal base 46 ft. high with a taper of 8 per cent. Above this the section is circular, the first 180 ft. above the base having a taper of 7 per cent., the next 100 ft. a taper of 4 per cent., and the remaining 180 ft. a taper of 2 per cent. The thickness of the wall varies from 60 in. at the base to 18 in. at the top. The inside diameter at the base is 66 ft. 6 in., at the top 50 ft. As a protection against corrosive gases the stack is lined (upon the completion of the chimney) with 4-in. acid-proof blocks laid with an acid-proof mortar containing soda and asbestos wool, and with a 2-inch air-space between them and the stack. The top of the stack is laid with acid-proof terra-cotta blocks with interlocking joints, all in acid-proof mortar.

[1] *Eng. and Min. Journ.*, LXXXVII, 156; *The Mineral Industry*, XV, 250.

The materials of the stack weigh in the aggregate 17,000 tons. To handle this, and to run up the chimney without interruption, the scaffold was arranged as follows: A square tower, large enough to go within the 50 ft. interior was made of 12 posts, each 10x10 in., and in 16-ft. lengths. Opposite each of the four flue-openings, two of the 10x10 posts, to which were spiked 2x4 in. oak strips, served as guides of a platform-elevator 6x9 ft. in area. The center span between opposite posts was 30 ft. The cathead, which carries the 14-in. hand-sheaves of the elevator, was built of two 3x10 in. fir sticks for the sheaves. Three of the elevators were run by electric motors at 300 ft. per min., using ⅝-in. steel hoisting-ropes. The fourth elevator was a steam hoist of 35 h.p., which was run at a speed of 600 ft. per min. The posts were braced by 2x6 in. sway-braces, and tied by 3x10 in. timber, upon which rests one end of main-floor tie-beams, 8x8 in. These beams were left in place as the scaffold rose, the outer end entering the putlock holes in the stack. On these rested 6x6 floor-beams, and these were covered by 2x10-in. scaffold planks laid immediately inside the stack to the width of 6 ft. The elevator posts were naturally unsupported on the side toward the elevators. To take this thrust there were introduced intersecting trusses at every 30 ft. of hight.

The time for raising a new hight of scaffold of 17 ft. was three or four hours, and this was done in over-time so as not to interrupt the masonwork. For less raises wooden horses and benches were used. The stack was plumbed by a transit as well as by the use of a plumb-rule. Speaking tubes were put in for communication, as well as electric bells for signals. The chimney was designed and erected by the A. Custodis Chimney Construction Company, New York, the flues and chamber by the Boston & Montana Company.

Blast-furnace Smelting Plants.

Mammoth Copper Mining Company.[1]—This, a subsidiary company of the United States Smelting, Refining and Mining Company, has a blast-furnace and converting plant at Kennett, Cal. The works treat, by semi-pyritic smelting, the massive sulphide ore of the Mammoth mine, to which is added silicious ore obtained outside, limestone, and 3 to 4 per cent. of coke. The works are situated 15,000 ft. from the mine, and at a level 2300 ft. below it. The ore was formerly sent down by means of a Riblet aerial wire-rope tramway, but, because of the sticky nature of the ore adhering to the bucket, and because of the uncertain nature of the tramway itself, this was abandoned in favor of an incline on which are operated 20-ton cars in balance. The incline is 4000 ft. long, having a drop of 1700 ft. vertically, and from it the ore is taken by an electric

[1] *Min. and Sci. Press*, XCVI, 30; *Min. World*, XXIX, 309.

tramway to the smelting-works. Silicious ore and coke are brought in by railroad, and limestone comes from the quarry over an electric tramway. The cars run over the top of the receiving-bins, dropping their loads into them.

The ore from the Mammoth mines is a massive pyrite averaging 3.6 per cent. copper and $1.50 per ton in gold and silver. To the charge of each furnace is added, say 100 tons of silicious ore daily, a good deal of which comes by rail from the Quartz Hill mine. The ore from the mine carries 93 per cent. silica and from $1 to $6 per ton in gold.

The works are motor-driven by power supplied by the Northern California Power Company, and 3000 h.p. are used. There are five furnaces, three of 42x180 in., and two of 50x180 in., having a united capacity of 1750 tons daily. They have water-jacketed tops in place of the ordinary brick tops. Some incrustation forms on the water-jacketed surfaces, but this flakes off, and does not, as in the case of a brick top, firmly adhere and build up to form a heavy incrustation. The smaller furnaces have forehearths 6 ft. 9 in. diameter. The slag from one flows into a "swivel pot" which serves to direct the flow to the slag-cars standing on a track 8 ft. below the slag-floor, so that there is no interference of the cars, and of the locomotive which moves them, with smelting operations. There is a continuous flow of matte from the forehearth into an endless-chain pan-conveyer. The matte, largely cooled in it, is at the end of the conveyer received into barrows and taken away for weighing, dumped, cooled with a spray of water, and finally loaded into cars.

The main-flue is of brick, 200 ft. long of 156 sq.ft. area, followed by a flue-chamber 64 ft. long and of 456 sq.ft. area, terminating in a self-sustaining stack 200 ft. high by 16 ft. diameter. The flue has a hopper bottom by which the dust is drawn into cars which travel on a track beneath. These cars are wheeled to an elevator and there raised and dumped into a storage-bin. This bin leads to a belt-conveyer to the briquetting machinery.

Blast for the furnaces is furnished by seven Connersville blowers, each of 124 cu.ft. capacity per revolution, and having a speed of 132 r.p.m. Of the original three blowers, two are arranged to run at a constant speed, while the third is run by a variable-speed motor.

The 20- to 30-per cent. matte was formerly shipped to Utah, its excess-iron having been utilized for the fluxing of silicious ore, but now the plant has a converter-department of two stands, with eight shells 96x 150 in. The converters are supplied with air by a blowing-engine of 10,000 cu.ft. per min. capacity, at a pressure of 15 lb. per sq.in., driven by a 750 h.p. motor. The relining-plant at the end of the converter-building has two Ingersoll-Rand tamping-machines, each carried by a steel

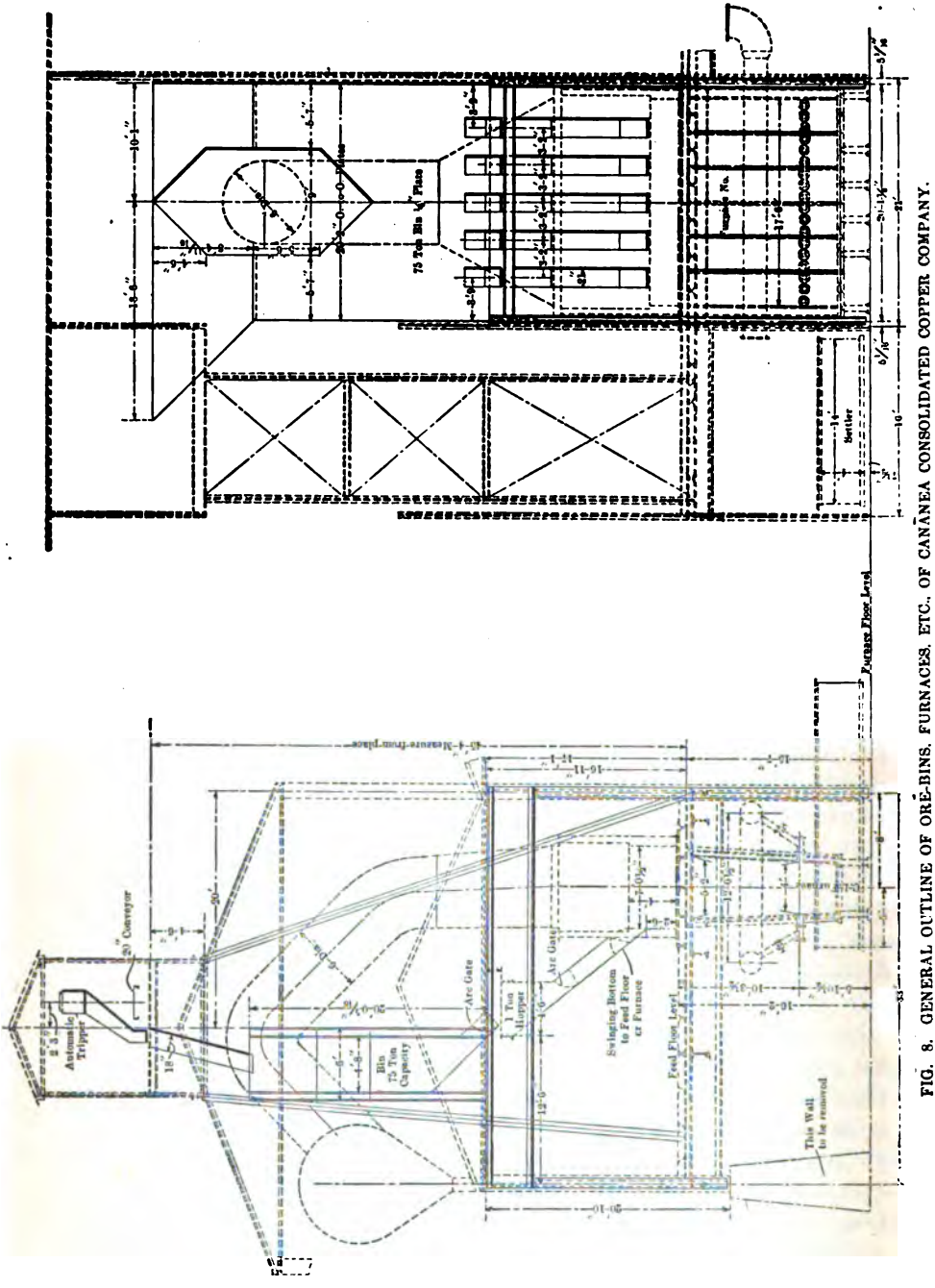


FIG. 8. GENERAL OUTLINE OF ORE-BINS, FURNACES, ETC., OF CANANEA CONSOLIDATED COPPER COMPANY.

jib-crane. The fume from the converters is taken by a dust-chamber 200 ft. long by 407 sq.ft. area, terminating in a self-sustaining stack, 150 ft. high by 12 ft. inside diameter. Two electric cranes, each with a main hoist having a capacity of 50 tons, and two auxiliary hoists of 15 tons capacity are provided for the converter building.

Cananea Consolidated Copper Company.—There are eight furnaces each 48x210 in. at the tuyeres, the details of which are given in the article cited.[1] They have 20 per cent. more area than the former furnaces, and smelt 90 per cent. more charge. Fig. 8 gives a general outline of the ore-bins and furnaces. The already fluxed charge from the bedding floor, described at length in THE MINERAL INDUSTRY, XVI, 356, is delivered at each furnace by a 20-in. conveying belt and an automatic tripper to a 75-ton storage-bin. From this bin the charges are withdrawn as needed at each side of a furnace to five one-ton hoppers. The chute from the hoppers swings so that a charge, if desired, can be dropped upon the feed-plate instead of into the furnace, and fed from the plate so as to keep the furnace properly trimmed; the feeding is all done at one side of the furnace. This method of charging the ore has proved to be a success.

The distance from the slag-floor to the feed-floor is 16 ft. 2 in., and the smelting column 10 ft. 3 in. Figs. 9 and 9a represent the two elevations of the furnace. The sole-plate stands on 12 cast-iron columns, the side-jacket, 12 ft. 7 in. high by 2 ft. 11 in. wide, resting directly on these. At each end there are two end-jackets, 9 ft. 7¼ in. high, beneath which come, to fill the remaining space, the cast-iron breast-jackets 5 ft. wide by 3 ft. high. On two of the side-jackets are tap-holes, 6 in. diameter, set 14 in. above the bottom of the jacket. In case it is desired to drain the hearth, one or both of these may be opened. Commonly they are plugged with clay. There are three tuyeres to each jacket, or 36 to the furnace, set 11⅛ in., center to center, 3 ft. above the bottom, and 5¼ in. in diameter, and each has its own shut-off valve. The tuyeres are plain cylindrical ones with a ball-and-socket connection to the blow-pipe; they have slag-escapes and peep-holes plugged with wood. The tuyere-cap hinges down, and the tuyere is shut off by its own valve when it is desired to punch it.

The inner sheets of the jackets are of ⅝-in., the outer ones of ⅜-in. flange steel. The sole-plate on top of, and within the jackets, is lined with 9 in. of brick set on end. To provide for loosening the sediment which accumulates at the bottom of the end-jackets, a scraper is provided actuated by a rod which comes out of the jacket through a gland. By occasionally reciprocating this scraper, the mud is loosened.

[1] *Eng. and Min. Journ.*, LXXXV, 754, 841, 867; LXXXVI, 603.

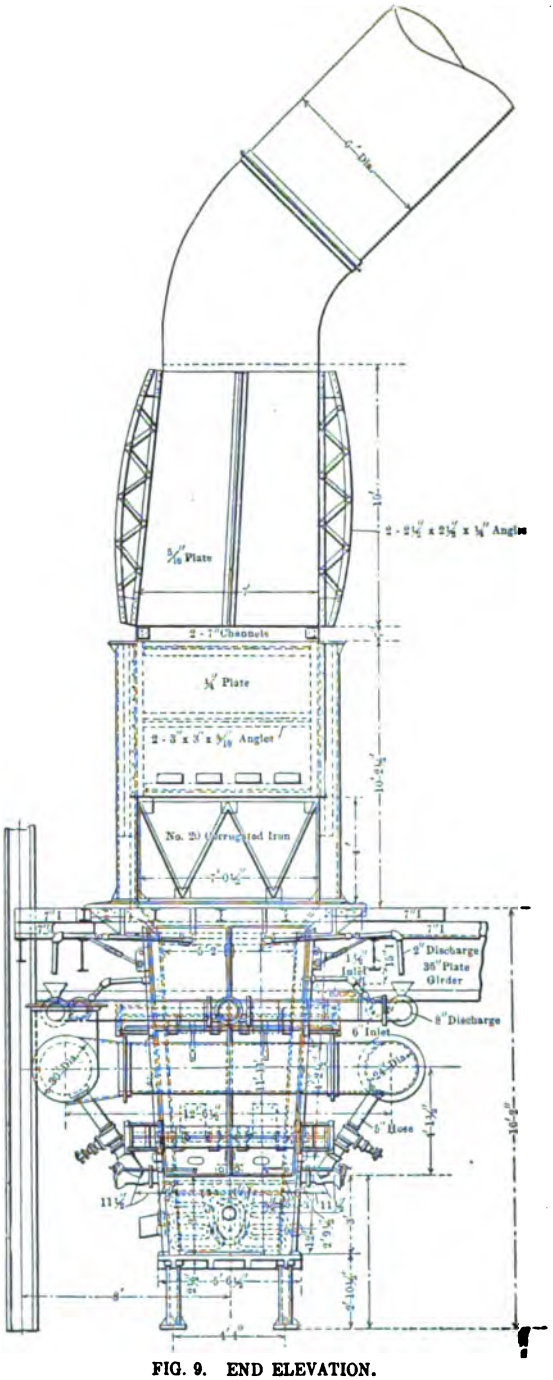


FIG. 9. END ELEVATION.

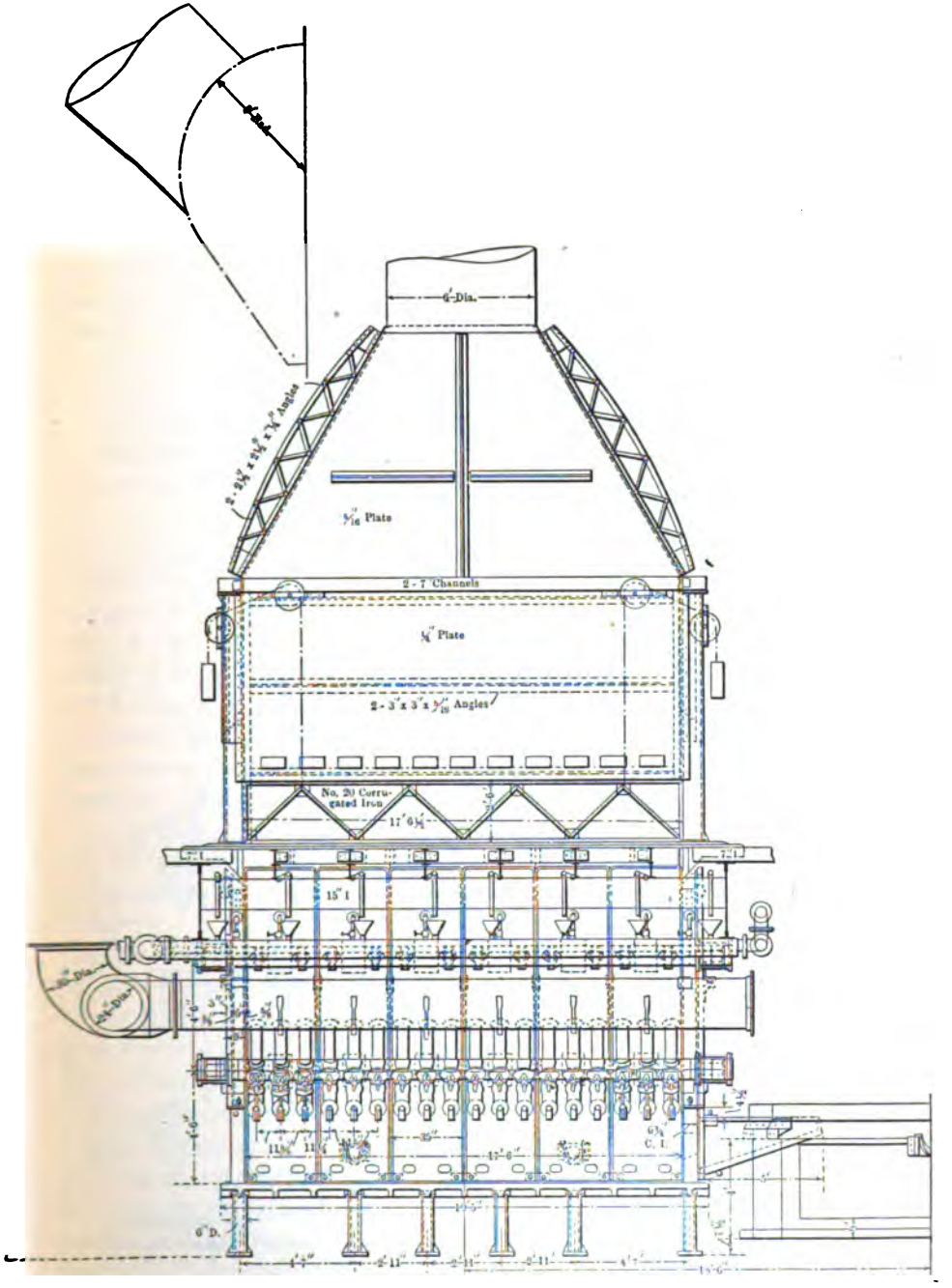


FIG. 9-A. SIDE ELEVATION.

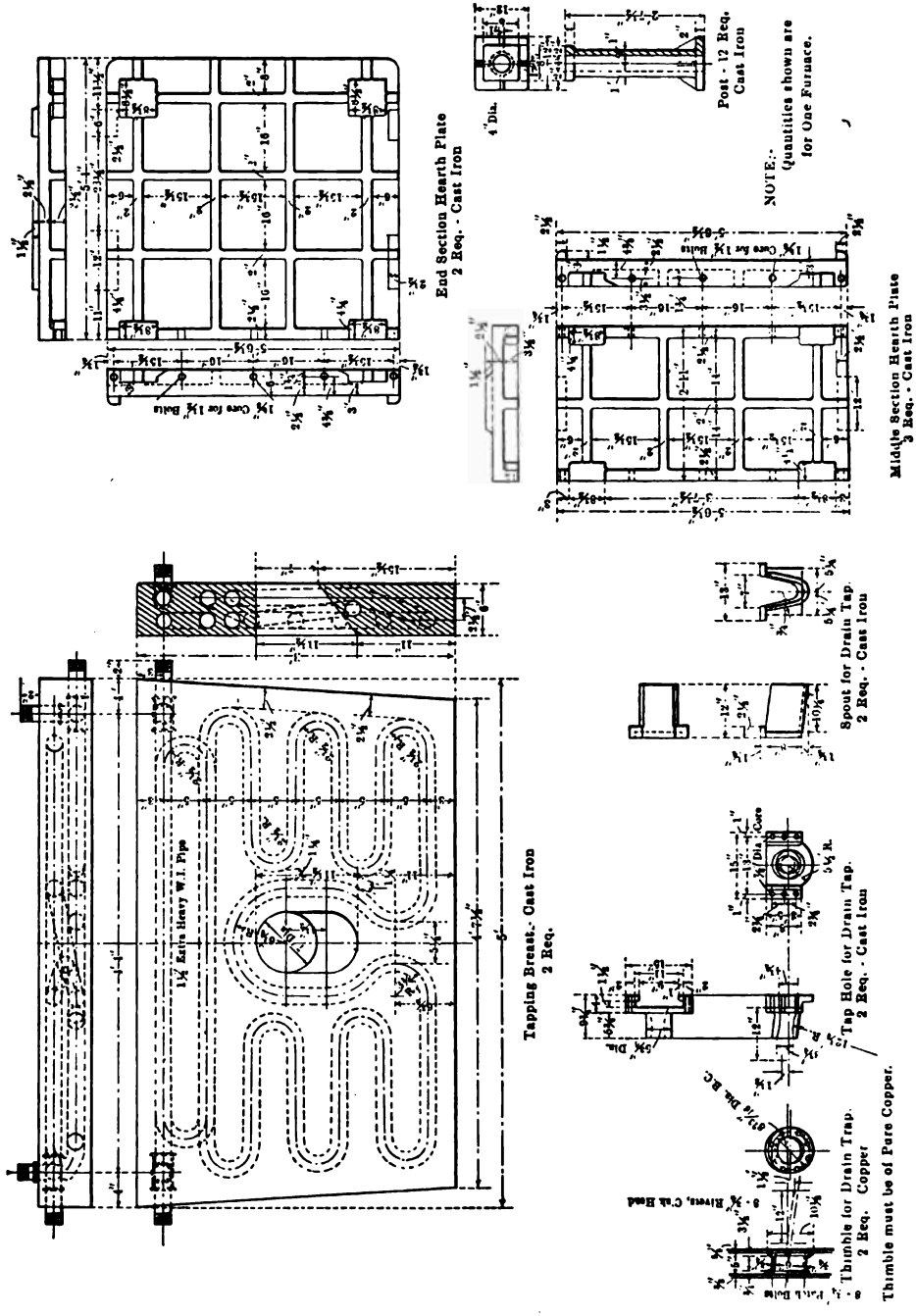


FIG. 10. CAST-IRON DETAILS, LOWER PART OF FURNACE, CANANEA CONS. COPPER CO.

The tapping breast, Fig. 10, is of cast iron, having within it a 1½-in. extra-heavy wrought-iron pipe, bent as shown, and with the iron cast around it at the foundry. Against it is set a trapped water-jacketed copper matte-and-slag spout, Fig. 11, 5 ft. long by 2 ft. deep inside at the breast, which rests upon the forehearth, and is fastened by two tie rods to the breast-jacket. This traps the spout to the amount of 9 in., to which is to be added the depth due to the formation of a slag-crust on the spout.

The door-sill has a sloping front over which the ore slides into the furnace. It fits immediately above the jackets, so no brickwork is needed. Above the feed-floor the closed top of the furnace is double, with a 9-in. air space, the air circulating inside to keep this space cool. Ten feet above the feed-floor the plates are single, braced to stiffen them, and terminating in a down-take 6 ft. in diameter. The feed-door 17 ft. 6½ in. wide by 2 ft. 6 in. high, is balanced and slides vertically to open it.

The bustle-pipe is large, 24 in. in diameter, branching from a 30-in. pipe. The forehearth is 14 ft. wide by 22 ft. 6 in. long, having rounded ends, and is 4 ft. deep. It is lined with a thickness of 9 in. of chrome-brick with a backing of 12 in. of ganister or converter lining, making 21 in. in all. The matte tap-holes are also of chrome-brick. Above the brickwork, the sides of the forehearth when running are built up with a wall of clay called locally "mud," so that the actual depth of the forehearth becomes 5 ft. 6 in. The peculiarity of the setting of the forehearth is that the slag enters at the side of it but near one end. The two slag-spouts are at the other end. It is the practice at these works to pour the converter-slag into the settler. To keep the pouring hole open, it had been the custom to drop pieces of cast iron in, but this had to be discontinued, because it ate out the bottom beneath the hole. With a present estimated capacity of 1,000,000 tons annually, the labor-cost for smelting has been reduced to one-fourth of what it was with the older furnaces.

A new blower (No. 10), of 300 cu.ft. displacement, has been installed. In place of two 42-in. crooked blast-mains, which caused by friction a loss of 160 h.p., a straighter 72-in. main has been put in. Fuel-oil has been tried out at the reverberatory furnace, and gives good satisfaction. The reclaiming machines, with some minor changes, are working well. One reclaiming machine with two men can handle 3000 tons of ore daily.

Luster Mining and Smelting Company.—The works of this company are situated at Santa Maria del Oro, Durango, Mex., and treat ore from the adjacent Magistral mines.[1] It is a three-furnace plant, making copper matte, each furnace being 40x168 in. and with a hight from tuyeres

[1] *Min. and Sci. Press*, XCVII, 843.

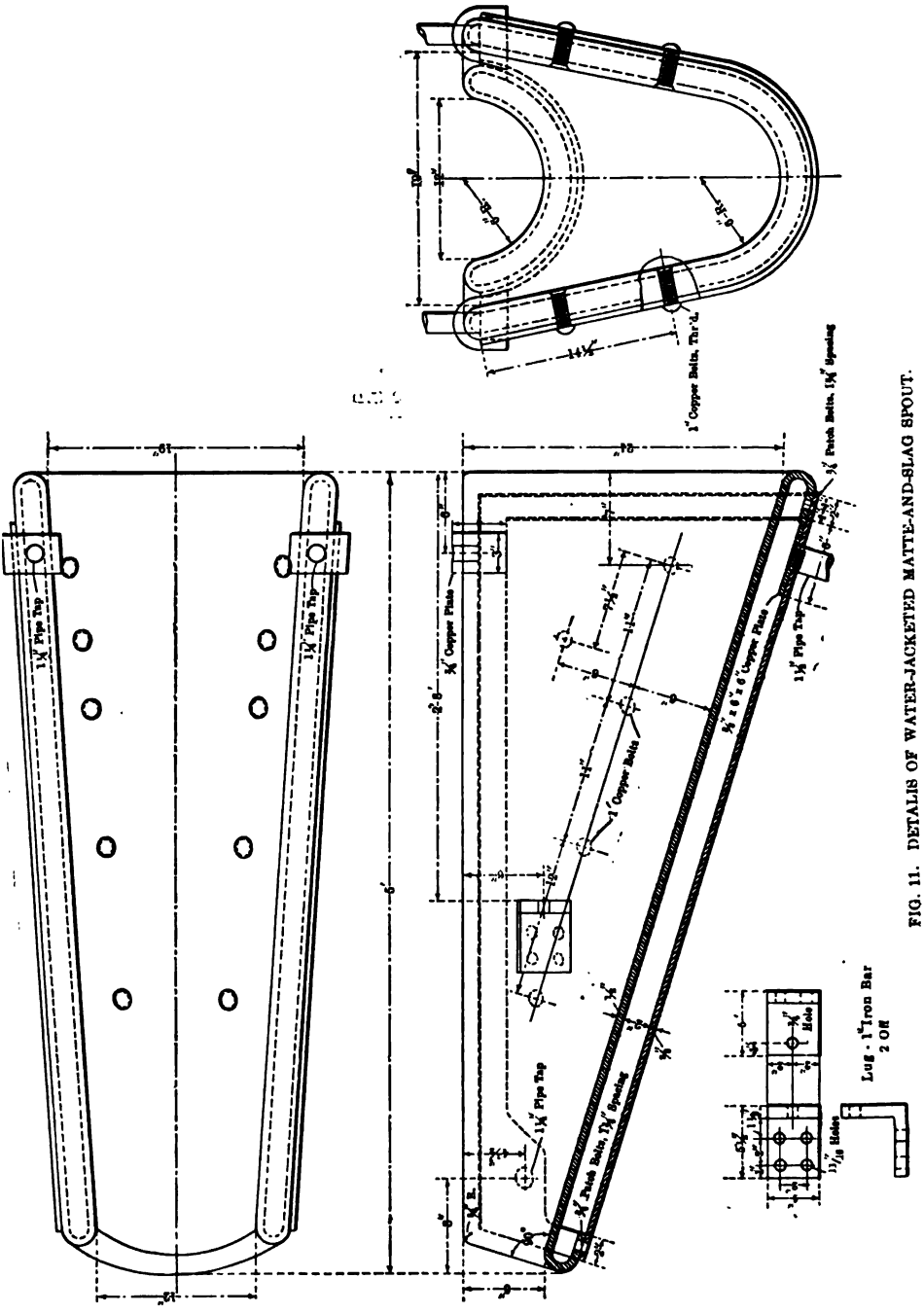


FIG. 11. DETAILS OF WATER-JACKETED MATTE-AND-SLAG SPOUT.

to charge-floor of 10 ft. For 7 ft. the jackets are vertical (no bosh), and above them comes the distributing or feed-plates running from the top of the jackets at an angle of 45 deg. to the feed-floor 3 ft. above. The coarser ore, fed on these plates, naturally rolls to the middle. There are both side and end doors, so that the furnace-charge can be properly trimmed or regulated. The top of the charge is generally held at the top of the jackets, or 7 ft. above the tuyeres, though, when decidedly basic, it may be raised to the feed-floor level, thus roasting off sulphur before it gets far down in the furnace. The sole of the furnace is lined 30 in. thick with ordinary brick, except that chrome brick is used near the tap-holes.

There are two circular forehearths, 6 ft. in diameter by 3 ft. deep, inside dimensions, to each furnace, the shell made of $\frac{3}{8}$-in. steel plates lined with the local fire-brick. These linings will last one month on an average. Each forehearth has two slag-spouts set at 120 deg. from one another at the front side. The maximum hight of matte in the settler is 30 in., and it is drawn down to 20 in.

Water is supplied to the jackets from a tank by a 6-in. pipe under a head of 18 ft. The jacket-water is returned over a cooling-tower to the supply-tank. Each furnace uses 700 gal. per min., which escapes at a temperature of 160 to 180 deg. F. Of new or fresh water, each furnace needs but 60 gal. per min., and this includes that for cooling and for sprinkling on both tap and feed-floors.

There are 16 tuyeres $4\frac{1}{4}$ in. in diameter, eight on each side, set at 10-in. centers, each with a quick-opening gate-valve. The bustle-pipe is of No. 8 iron, 24 in. in diameter. At the power house are three positive-blast blowers, motor-driven by 150-h.p. electric motors. Two of the blowers are No. $8\frac{1}{2}$ Root, one of the No. $7\frac{1}{2}$ Connersville type. The ordinary pressure is 11 oz. per sq.in. at the tuyeres, 15 oz. at the blowers, the difference being due to the resistances of the hot-blast stove through which the blast comes. Each furnace takes 12,500 cu.ft. of air per minute or 100,000 cu.ft. per ton of charge, figured on 180 tons daily capacity. This makes 500 cu.ft. of air per pound of charge.

McDonald hot-blast stoves are used, by which the blast is heated 70 to 80 deg. F. before entering the furnace. The down-takes enter at the top of the stoves, and the bulk of the flue dust is deposited in the space beneath the heater-pipes, this space being cleaned out periodically. From the stove there leads away a flue 7 ft. wide by 8 ft. 2 in. high, of a total length of 1100 ft., rising 136 ft. in that distance. Then comes the stack, only 15 ft. high, giving a total hight of draft-column of 151 ft.

The slag is caught in oval slag-pots of 16 cu.ft. capacity and is drawn out to the dump by mules. Matte-pots of 2 cu.ft. capacity are used, made

of mild cast-steel. Cast iron was found to be unsatisfactory, while steel has been found to last at least during the nine months that the works had been in operation. The local fire-brick is made on the ground, costing $15 per M., the clay being ground and the brick molded by hand. Imported fire-brick would cost $95 per M.

The furnaces are charged by two-wheeled buggies. The ores may be divided into four general classes, and from them the necessary self-fluxing mixture is selected and bedded to give a uniformly-working charge. The average constitutions of these classes are given in the accompanying

COMPOSITION OF ORE SMELTED AT LUSTER WORKS.

Class.	SiO_2. %	Fe. %	MgO. %	Al_2O_3. %	S. %	CaO. %	Cu. %	Au.
Oxidised ore	62	13	3	4	2	2	2	$12
Silicious sulphide	42	26	2	7	19	3	0.9	13
Sulphide ore	18	40	2	3	35	2	1.1	8
Limy ore	9	5	2	3	2	48	0.2	3

table. The furnace-charge consists of 4000 lb. dry weight of ore, 120 lb. of low-grade matte and foul slag (weight not counted), with 460 lb. or 11 per cent. of West Virginia coke. The slag composition is SiO_2, 46.9 per cent.; FeO, 34.4; CaO+MgO, 10; Al_2O_3, 6.4; S, 2.1; CuO, 12 per cent., with $0.88 gold per ton of slag. The slag is computed at 80 per cent. of the charge. The matte contains Fe, 57.1 per cent.; Cu, 15 per cent.; S, 27.9 per cent., and carries $186.30 gold per ton. About 85 per cent. of the sulphur is volatilized, and there is a concentration of 16 : 1. During four months' operations, with the drawback of insufficient blast, the actual recovery of copper was found to be 83.3 per cent.; that of the gold 87.4 per cent., figured on the raw ore. The normal recovery may be given at 84.7 per cent. of the copper and 89 per cent. of the gold.

Cerro de Pasco Mining Company.—The plant has a side-hill location, and is of steel construction throughout, covered with corrugated iron.[1] The ore from the various mines comes in by rail in 30-ton gondola cars, and is dumped from them into 24 storage bins. The ore is drawn from these bins into two-ton charge cars, the limestone for fluxing being added in the cars.

There are three furnaces, 56x180 in., with a united capacity of 500 tons daily. Each furnace has 12 tuyeres to the side, delivering air under 18 to 20 oz. pressure to the sq. in. The furnaces tap into forehearths, 18 ft. diameter, having a capacity of 30 tons. Each of the intermediate settlers is common to two furnaces. The furnace gases pass by a concrete flue to a steel stack 185 ft. high.

[1] *Eng. and Min. Journ.*, LXXXV, 763.

The matte from the settlers is withdrawn to a ladle handled by the 80-ton traveling crane used in the converter building. There are four stands of converters, the converter shells being 72x168 in. having 14 tuyeres and operating under an 18-lb. blast.

The coke is made at the works in a double tank of 70 beehive coke-ovens, 35 to a side. The capacity of each oven is six tons of coal yielding daily two tons of 48-hour coke. The coal is washed to reduce the percentage of contained ash.

There are under construction several 19x60-ft. reverberatory furnaces which are to take the calcines from Klepetko-Evans-McDougal roasters. The coarse ore, which is screened from the fine, goes to the blast-furnaces.

Douglas Smeltery, Fundicion, Sonora.—The ore is delivered to the smelter from the mines (30 miles distant) by wagon-trains drawn by traction-engines.[1] An engine costs $12,500 and is equipped with a derrick so that it can load and unload heavy weights. This was found a great convenience during the construction of the works. It will draw a load of 45 tons of ore from the mine to the smelter in one day, the train consisting of an engine and five trucks, each truck carrying three steel four-ton buckets. The train-crew consists of three men (engineer, fireman, and brakeman). Wood is used for fuel. Water is obtained from wells sunk along the route, and but 12 minutes are consumed in taking water at any well.

On arriving at the works, the wagon-train runs alongside of a small receiving-bin. The buckets containing the ore are lifted off the car by an electrically-operated overhead traveling-crane and dumped into the bin. The ore from this bin is transferred to the regular storage-bins, being drawn off from the receiving-bin into five-ton side-dump steel tram-cars, using a 30-in. narrow-gage industrial railway operated by Davenport locomotives, hauled up a trestle to the storage-bins and dumped into them.

The initial equipment is one water-jacketed blast-furnace, 44x160 in., 13 ft. 6 in. from tuyeres to feed-floor, set on a foundation 3 ft. above the ground-floor level. There are eight lower and eight upper welded-steel water-jackets. The lower row has on each side two wide and one narrow middle jacket, this latter being shorter to leave room for a tap-jacket, so that the tapping is done at the middle of the long side. The blast is delivered from the blowers by a 36-in. spiral-riveted pipe to 24 tuyeres, 12 to each side.

The settler is a steel shell 15 ft. in diameter by 5 ft. high, lined with magnesite brick. The matte pots are of 35 cu.ft. capacity; the slag-pots of 45 cu.ft., or five tons each.

[1] *Eng. and Min. Journ.*, LXXXV, 303; LXXXVI, 413. *The Mineral Industry*, Vol. XV, 354.

The furnace is connected by means of a sheet-steel down-take to a balloon-shaped dust-flue, 150 ft. long, 12 feet wide by 15 ft. high, which discharges into a self-supporting steel stack 177 ft. high, 14 ft. diameter at the bottom and 8 ft. at the top. This is long enough to take care of two additional furnaces, or for a plant of 1000 tons daily capacity.

Charging of the furnace is done as follows: Ore, fuel, iron ore and limestone are drawn from the storage-bins into one of the four compartments of the charge-cars, the calculated quantity being measured into the compartments. From the charge-cars the four ingredients, thus measured, are dropped into a hopper delivering into the boot of an inclined bucket-elevator and thence into bins of 30 tons capacity, one at each end of the furnace. Above the charge-floor it is received into another large hopper, whence it is drawn as desired into buggies and by them charged into the furnace. It is thus seen that the charge is well mixed before going into the furnace. It is also in contemplation, when more furnaces are installed, to shoot from the feed-hopper directly to the furnace without using the charge-buggies.

To lessen the expense of a larger sampling plant a novel method is used. A Vezin sampler taking out one-tenth, and an inclined motor-driven bucket-elevator have been mounted on trucks to move on the narrow-gage railroad tracks outside the bins to the unloading point. The ore to be sampled is put through the sampler and discharged into the regular side-dump cars. These then go to the sampling-mill. The rejected portion is delivered to other cars and thence to the main storage-bins.

Fines, concentrates and flue-dust are thus treated: They are put into bins, and thence fed to the boot of an elevator which mixes them with lime in a pug-mill to a stiff mud. This lime, which is burned at the works, contains some gold and silver. The pug-mill is set so high that its product can fall into a charge-car set beneath. The wet mixture is fed without drying to the blast-furnace.

Power plant: The blast is furnished from a No. 9 Root blower (116 cu.ft. per rev.) direct driven. The water piping throughout is exposed, there being no danger of freezing in this climate.

Smelting costs: It is computed that 17½ tons of ore are handled per man employed. The smelting costs are:

For unloading, sampling, and yard charges: Labor, $0.40; wood for locomotive, $0.01; tools and supplies, $0.01; steam power, $0.15; total, $0.57. Direct charges: Labor, $0.33; coke, $1.96; wood and sundries, $0.01; tools and supplies, $0.01; repairs, $0.03; steam power, $0.14; total, $2.48. Indirect charges: Administration, $0.06; taxes, etc., $0.01;

general expense, $0.10; laboratory expense, $0.03; salaries, $0.20; total, $0.40. Total smelting costs, $2.88.

Summary of all charges: Mining, $3.99; development, $1.25; transportation, El Cobre to Fundicion, $2.21; sampling, handling and yard-charges, $0.57; smelting, $2.88; refining, transportation, and selling commissions, $3.09; federal-government taxes, etc., $0.24; total, $15.73. The metal values received are: 5.65 per cent. copper at 13c. per lb., $14.69; gold and silver, $9.63; total, $24.32. Deducting all charges ($15.73), the net profit will be $8.59.

The limestone contains $1.20 per ton, which covers the cost of mining and smelting it, so no charge need be made for that item. The costs above given are likely to be reduced with increased tonnage and with closer operating.

Great Cobar Smelting Works.—This is one of the most modern of Australian plants, having a capacity of 2000 tons daily with three furnaces in operation.[1] The article here abstracted gives a perspective view of the furnace showing constructive details, also an elevation of a sampling plant, having a capacity of 50 tons per hour, in which every fifth car coming from the mine is sampled.

There are three 56x240-in. blast furnaces arranged end to end in the furnace-building with spaces between where the forehearths are situated, these settlers being 18 ft. diameter by 4 ft. 6 in. deep. Settlers are also set beyond the end-furnaces. In this way any furnace may discharge at either end to a settler, as may sometimes be important to do in case of failure of a given settler. The furnaces are set 9 ft. high above the ground-floor on a masonry foundation. This gives an ample fall of the slag to the slag-pots, and for drawing off matte for converter use.

A furnace has a heavy base-plate on which stands the lower jackets 10 ft. high, and on these upper jackets, also of the same hight, independently supported. Allowing a distance of 6 ft. from the floor level to the base-plate this gives a distance of 35 ft. from ground-level to charge-floor. Each furnace has 40 tuyeres, 20 on each side, at 12-in. centers. The slag-spout is trapped, and of the usual water-jacketed type.

The furnaces are charged by side-dump cars which run on tracks of 36-in. gage, one at each side of the furnace and handled by electric locomotives. The ore is stored in hopper-bins and is drawn off over hopper-scales.

Fig. 12 is an elevation of the furnace building, the converter-building flues and stack of the plant. The slag from the furnaces is handled in pots holding 25 tons each, mounted on four-wheeled trucks and hauled by an electric locomotive. Two tracks approach each settler, so that a slag-

[1] *Eng. and Min. Journ.*, LXXXV 960.

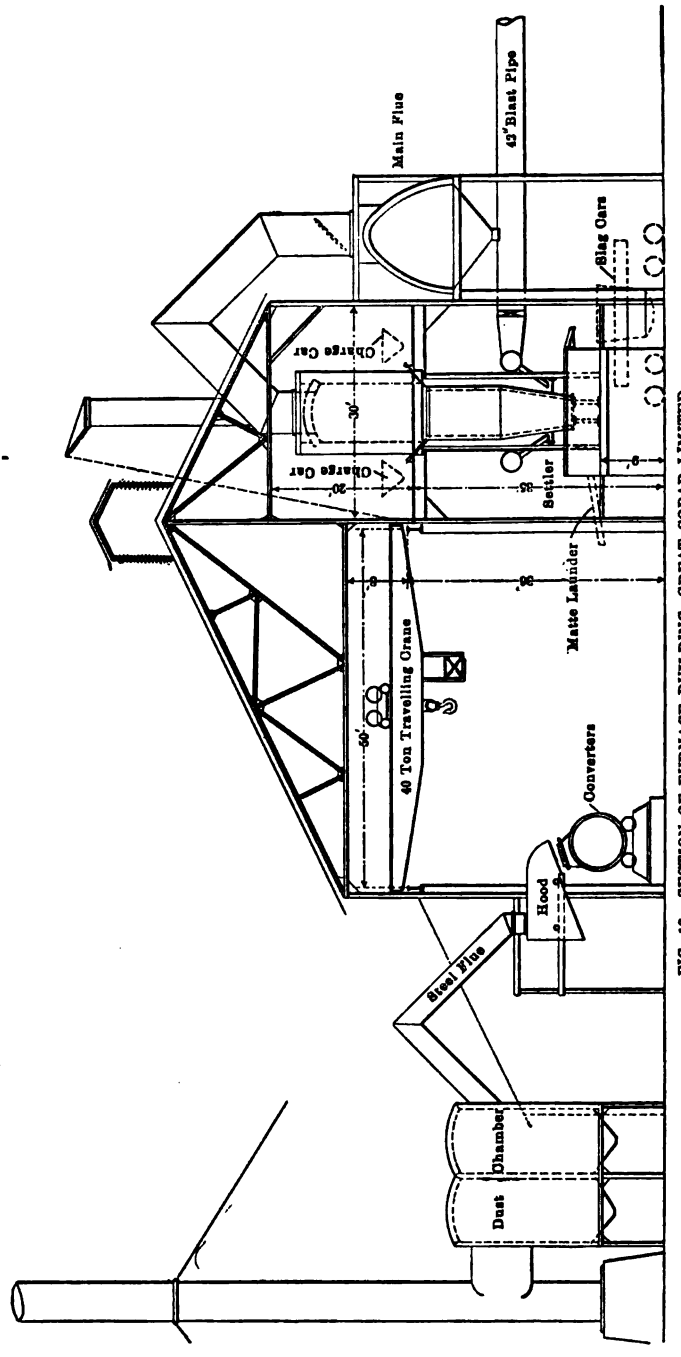


FIG. 12. SECTION OF FURNACE BUILDING, GREAT COBAR LIMITED.

car can be set in place and be filling, while the full one is taken away. The flue-dust is carried by a double inclined down-take to a balloon-shaped sheet-steel flue, lined with brick and having a continuous hopper-bottom, the whole set so high that the flue-dust can be drawn off without interference with the slag-cars. Railway tracks about the works are of 4 ft. 8½ in., or standard gage.

The converter plant is provided with two stands of converters, 84 in. diameter by 124 in. long, electrically operated, and with five extra shells. There is a 40-ton traveling electric crane of 50-ft. span running the full length of the building (176 ft.). The matte from the settlers is tapped into 10-ton ladles, which are taken to the converters by the traveling crane. The converter-copper is also poured into similar ladles and transported by the crane to the copper-casting room adjoining and near one end of the converter building. Here the copper is poured into a 15-ton reverberatory furnace. When full this is poled and cast into anodes. Silicious gold and silver ore is used for converter linings. Two 6-ft. wet-pans are used for mixing, these being set at the end of the converter-building. The lining is tamped in with an air-driven tamper suspended from a jib-crane.

The gases from the converter-plant go through the hood, and by a steel connecting flue, to the hopper-bottomed brick dust-chamber and thence to an independent stack 180 ft. high. The chamber is of steel construction with brick walls, and the steel hoppers below are high enough so that the dust can be drawn off to cars set and operated by a locomotive passing beneath. The top of the chamber, however, is of steel plates to increase radiation. (It appears that corrosion, due to the gases, may make trouble here.)

The power-plant is set 400 ft. away from the smelter-building, and the plant is designed to effect as great an economy as possible, coal being expensive. For the furnaces are provided three direct-connected, No. 10 Connersville blowers (300 ft. displacement per rev.), and expected to operate at a pressure of 42 to 48 oz. per sq.in., each blower being independently connected to its own furnace. At the same time there are provided cross-connections so that any blower can be connected to any furnace, or so that the pressure in the blast-mains can be equalized. To save space the blowers and their engines are set at an angle of 45 deg. to the walls of the power-house. Air is supplied to the converter by a duplex blowing-engine of 10,000 cu.ft. capacity per minute at 15 lb. pressure per sq.in. There are also two pair of compressors, each of 2500 cu.ft. per minute, compressing to 80 lb. per sq.in. Electric power for the plant is furnished from three 300-k.w. direct-connected generators, two being in operation and one always in reserve. Steam is supplied at

170 lb. pressure from a battery of six 500-h.p. Babcock & Wilcox water-tube boilers fired by chain-grate stokers. The coal for the stokers is supplied from steel hopper-bins overhead, and into these bins the coal is delivered by a pan-conveyer, which runs the whole length of the building over the hopper-bins, then outside the building and passing under a receiving hopper into which the coal is discharged from the railroad cars. The same conveyer on its return runs under the boiler room and receives ashes discharged from the stokers. These ashes are received into steel hoppers placed over the railroad tracks at one end of the building, and thence drawn to railroad cars. The smelting and converting building is of steel construction with corrugated-iron roofs, but without the side or end-walls found necessary in a colder climate.

Granby Smelting Works.—This works at Grand Forks, B. C., underwent extensive improvements in 1908.[1] They included the construction of additional ore and coke bins of 7000 and 4000 tons respectively; a sheet-steel chamber back of the furnace to replace the brick dust-chamber formerly used; enlargement of the brick flue-dust chamber near the stack; replacing the older wooden blowing-engine building with one of steel construction of larger size and having a cement floor; installation of two Connersville blowers, each of 30,000 cu.ft. per min. capacity, and each driven by two 150 h.p. motors; lengthening the converter-building to install three 84x126 in. converters in place of those formerly in use, to have a united capacity of 36,000,000 lb. blister copper yearly and to be electrically operated. There has also been added a two-cylinder Nordberg blowing-engine of 10,000 cu.ft. of free air per min. at 12 lb. pressure driven by electrical power. The eight blast-furnaces, now 44 in.x18 ft. long, are being lengthened to 22 ft., which will make their united capacity 4000 tons daily in place of 3000 as at present.

Balaklala Smelting Works.—This works treats copper ore from the Balaklala mines, Coram, Shasta county, Cal. It contains three water-jacketed blast-furnaces, 56x240 in. at the tuyeres; also a reverberatory furnace of 17x95 ft. hearth dimensions. The reverberatory furnace will be oil-fired, and is expected to handle 200 tons of calcines in 24 hours. The fine ore for this charge is roasted in McDougal furnaces. The main stack is 20 ft. diameter by 250 ft. high. Electricity is largely used for power purposes.[2]

Bully Hill Smelting Works.—This comprises two water-jacketed blast-furnaces, 42x200 in. hearth dimensions, with air-jacketed tops, and having a capacity of 350 to 400 tons each daily; also a reverbatory furnace of 75 tons capacity. Hot blast is used for the blast-furnaces, the stove

[1] *Eng. and Min. Journ.*, LXXXVII, 249.
[2] *Min. World*, XXIX, 310.

being heated by the waste heat from the reverberatory. In case this latter is not in use the stove is heated by oil. The stove has 72 U-pipes, 15 in. diameter.

The above is new equipment. In addition there is the old plant containing two furnaces, McDougal roasters and a converting plant, this latter ample to take care of the output of the furnaces.[1]

Tennessee Copper Company.—See article by Mr. Gottsberger in earlier pages of this chapter.

Converting—Copper Mattes.

Converting Copper Matte at Anaconda, Mont.—C. Offerhaus gives details of charging and operating the converters at Anaconda.[2] The matte, of a grade of 38 to 52 per cent. copper, taken either from the reverberatory or the blast-furnaces, comes in 10-ton ladles, mounted on cars, to the charging-floor of the converting plant, 23 ft. above the working-floor. There are three men on the charging-floor, two to dump the ladles, and one who takes at equal intervals three samples of the matte as it flows from the ladle to the converter, the dumping taking five minutes. Before, or during charging, a few lumps of coal are thrown into the converter to prevent the shooting-out of some of the molten matte. This expulsion of matte has been laid to imperfect drying, or, in the case of an already-used converter, to the reaction between the freshly-poured matte and copper oxide which has remained behind in the converter from the last charge. If the matte is charged hot this shooting is less likely to occur.

First period: During the first part of the blow dense white vapors, due to the volatilization of zinc and lead, leave the mouth of the converter. The color of the flame is red, but it soon turns to greenish yellow. A small or jumping flame indicates that the blast is not properly received, and that the tuyeres need punching. If the slag-crust is hard to punch a hammer is used to drive the $\frac{3}{4}$-in. punching-rod through, and this is then withdrawn with a "claw." Continuous punching is mostly needed if the matte is rather cold. The blast oxidizes iron to ferrous form; and this enters the slag; no higher oxide could endure in presence of the sulphides. Copper is oxidized to the cuprous form, but this, at the first stage, again reverts to matte, so that but little copper oxide gets into the slag. Observing the flame, one notices after some time, pale-blue flashes in the prevalent greenish-yellow; these flashes increase, and finally become permanent. Soon the pale-blue, which is probably a cuprous-silicate flame will dominate, indicating that practically all the iron has

[1] *Min. World*, XXIX, 311.
[2] *Eng. and Min. Journ.*, LXXXVI, 747.

been oxidized and gone to form slag, the copper remaining as cuprous sulphide (white-metal) of 70 per cent. copper. The average time up to this stage is one hour.

Pouring: The slag is now ready for pouring. The converter is turned, the blast cut off, and the slag poured into five-ton cast-steel ladles. It is difficult to distinguish the matte from the slag at this time; however, by pushing back part of the stream with a rabble every quarter minute or so, it is possible, after some practice, to distinguish the first drops of white-metal. This looks like boiling grease against the darker smooth surface of the slag which adheres to the rabble blade. As soon as the skimmer sees the matte on the rabble blade, he indicates to the converter man by a rapid to-and-fro motion of the rabble to pour more slowly or to stop altogether. The slag-ladle is taken by the traveling-crane, put into a hydraulically-operated cradle, and the slag slowly poured into an endless-chain slag-casting machine. Some white-metal settles out, during the pouring, and the slag-ladle is finally slowly drained. Considerable crust forms in the ladle; this is dumped out and the interior of the ladle then receives a lime wash, and is ready for use again. The chemical wear of the converter is at the tuyere openings; the mechanical wear is opposite, nearer the top of the converter. Copper to the extent of 2.39 per cent. (for a period of six months) still remains in the slag, mostly as drops of white-metal or matte, so that we may expect a heavy slag from a high-grade matte to contain more copper than the lighter slag produced from a lower-grade matte.

Second period: The slag has been poured off all but a thin layer, then one or two tray-loads of sweepings, consisting mainly of solid white-metal and other rich material from the converting operations, are put in the converter to cool it, the blast is turned on and the converter turned back into blowing-position. Cuprous oxide forms, and reacts with cuprous sulphide, copper being produced. It is found, if the converter is too hot, that the tuyeres become easily stopped by cuprous silicate and are hard to punch, while the reaction between the oxide and sulphide proceeds slowly. The desired temperature of the charge and the quantity of sweepings to add is judged by the color of the flame. This should be brownish, yellow or orange; a reddish flame denotes too low a temperature. If it is this latter, some coal is charged in. If the temperature is too high, as indicated by a white-yellow flame, and there are hard tuyeres, more sweepings are charged from a bucket with a dumping bottom, the converter remaining in blowing position. Another means of regulating temperature is by changing the pointing of the tuyeres by tilting the converter somewhat. The time needed for the second blow is 60 to 90 minutes. The critical time for the finish is recognized by examining the

copper, which on punching adheres to the punching-rod, or by the appearance of the flame. The brownish-yellow or orange flame becomes darker, takes a brown color, and slows down. Shots of metal from the converter no longer stick to the hood, but rebound from it, and are no longer dull but burning. At this moment all the sulphur is burned out, and but little copper oxide is present. It is difficult to recognize this point, and it takes long experience to do so.

Pouring the metal: The converter is turned down, and a small sample taken out. This varies in sulphur as follows: "black blister-copper" has the least sulphur and has considerable oxygen; "blister-copper" showing blisters and less oxygen has more; "clean copper" has the most sulphur of the three. The contents of the converter are now poured into a five-ton steel ladle lined with fire-clay. In pouring, a pole is used to hold back the slag produced during second period. The following gives the composition of this slag: Cu, 21.43 per cent.; SiO_2, 22.7; FeO, 39.4; S, 0.4 per cent. The ladle is transferred by the travelling-crane to a hydraulically-operated cradle, which pours the copper into the refining furnace.

The refining furnace will hold 20 to 30 ladlefuls of copper. The first of these may be so-called "clean copper," higher in sulphur, but the last ones should be carefully finished; otherwise much sulphur will cause excessive boiling in the furnace and some slopping over.

Very little flue-dust is made by a converter. The 11 converters at Anaconda made 140 tons of flue-dust in one month, which contained 71.23 per cent. of copper and 12.5 per cent. of sulphur. It is sent to the reverberatory furnaces.

It usually takes 60 minutes to blow a charge of seven tons of white-metal, but a cold charge will take longer. The temperature can be raised by charging a few lumps of coal. When cold, not all the slag separates, and then the matte fails to boil or to look oily on the rabble blade used for testing the charge. Another seven tons is then charged, and will give the necessary heat to separate the slag well. This is called "doubling." Where the matte is low-grade doubling may be necessary to furnish the needed amount of white-metal to finish properly. Indeed two white-metal charges may be united, and so a converter emptied ready for a new charge.

If, at the end of the first period, the pouring of the slag is delayed, white-metal will be oxidized, and cuprous oxide enter the slag or be set free to react on the white-metal. In consequence much sulphur dioxide may be evolved, the slag foams out of the mouth of the converter, and severe explosions may take place, either when the converter is in blowing-position, or when it is turned down to pour the slag. A skimming at

an earlier period, to avoid an accumulation of slag, will reduce the violence of the explosion. The foamy mushy slag thus produced runs high in copper, being a ferrous-cuprous silicate containing cuprous and cupric oxides mechanically mixed.

Foaming may occur during the second period, but it is then a less serious matter, and can be kept down by the addition of a few lumps of coal. Slag, during the second period, may be due to imperfect separation at the end of the first. If much sweepings is added, the iron present in it may add to the volume of the slag. The following is the analysis of a mushy slag formed during the second period: Cu, 27.9 per cent.; FeO, 29; SiO_2, 29; S, 3.4 per cent. This slag goes back into the next charge of the converter.

Beneath the foamy slag of the second period is copper, still retaining sulphur, and called "regal" or regule. An average analysis of it is as follows: Cu, 92 per cent.; Fe, 0.8 per cent.; S, 1 per cent. Overblowing during the second period rarely occurs, and then because the skimmer

RECORD OF SOME ACTUAL RUNS, ANACONDA WORKS.

	Time	
	Charged	Interval
First run:		
Matte charged	12.25 M.	
One ladle of slag poured (flame greenish yellow)	1.05 p.m.	40 min.
Blue flashes appear in the flame	1.25	60 "
All slag poured and white metal transferred to another converter	1.35	70 "
Second run:		
Matte charged	7.20 a.m.	
One ladle of slag poured (flame blue)	8.10	50 "
Rest of slag poured and one bucket of sweepings charged	8.30	70 "
Blister copper poured	9.25	125 "
Third run:		
Matte charged	8.25 a.m.	
Half ladle of slag poured (flame greenish yellow)	9.35	70 "
Blue flashes break through	10.15	110 "
Half ladle of slag poured	10.20	115 "
Rest of slag poured and one bucket of sweepings charged	10.35	130 "
One bucket of sweepings charged ready for pouring blister copper	10.55	150 "

Time is lost in waiting for matte, in repairing stalls, etc. Thus, on a specified day, the total time taken for charging was 39 min., waiting for matte, 82 min., charging converters, 31 min., stall down, 641 min., time blowing, 174 min.

may have mistaken a slightly overblown charge for a charge in the early stage of the first period. When a charge is overblown, copper-oxides are formed, and these slag with the lining, but the heat evolved drops, and contents of the converter, consisting of solid copper, copper oxides and copper silicate freezes. This material may be worked up in other converters, or may be put into the refining-furnace. A converter-lining will stand for five or six charges. If the skimmer thinks it will not stand

another charge, it is replaced by a freshly-lined converter. At the final charge all the second-period slag is drained from the converter. If possible, such a converter is washed with matte to remove adherent copper, which otherwise is hard to remove or cut out. The matte used in the washing is poured into a ladle and sent to another converter.

Copper and Silver Losses in the "Direct Copper Process," Mansfeld, Germany.—This modification of the Welsh process of treating high-grade copper matte or white metal is known at Mansfeld as reaction-smelting,[1] and consists in roasting the matte, then melting it down with raw matte of the same grade to produce anode-copper according to the reaction, $Cu_2S+2Cu_2O=6Cu+SO_2$. The melting down and skimming takes 16 to 18 hours, the poling and casting six to eight hours more, i.e., a charge is put through daily. To determine the losses by this method at Mansfeld, the results of a month's run, as below given, were taken.

There were smelted 1117.57 metric tons of sulphide and oxidized materials, consisting of raw white-metal with 76.5 per cent. copper and 131 oz. silver per ton, roasted matte of 73.5 to 74 per cent. copper and 105 oz. silver per ton, together with a small amount of by-products containing 46 per cent. copper and 46 oz. silver per ton, or in all, containing 832.117 tons of copper and 2501.857 kg. of silver. There was produced 758.927 tons of anode-copper containing 99.311 per cent. copper and 90 oz. silver per ton, and from the reverberatory slag (by smelting it in a blast-furnace) 80.85 tons of "cupola blocks," containing 84.05 per cent. copper and 30 oz. silver per ton. The blast-furnace slag, 27 tons, still retains 1.41 per cent. copper but no silver. These three products, the anode-copper, the cupola blocks and the slag, accounted for 821.273 tons of copper and 2439.084 kg. of silver. This would correspond to a loss of 1.303 per cent. of the copper and to 2.5091 per cent. of the silver contained in the reverberatory-furnace charge.

Machine for Casting Converter Copper.—This, as shown in *Eng. and Min. Journ.*, LXXXV, 903, consists of an enlarged Walker casting-machine 24 ft. in diameter, which carries 24 anode molds, as shown in the plan view, Fig. 13. In pouring or molding the copper, when a charge has been blown to blister at the converter, it is poured into a ladle which is transferred by traveling crane to the pouring apparatus shown in the figure. This consists of a vertical hydraulically-operated plunger, on top of which is pivoted a platform on which the ladle stands and to which it is secured. The whole arrangement is so adjusted that when the plunger rises, the ladle-spout maintains a uniform distance from the mold. The adjustment is such that in pouring, the spout moves across

[1] *Metallurgie*, 1908, p. 356.

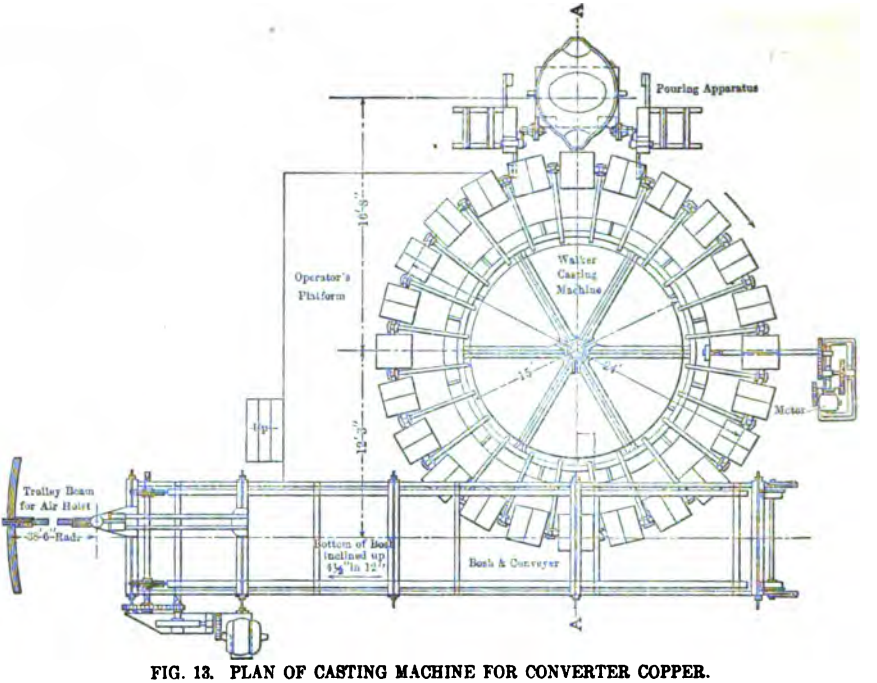


FIG. 13. PLAN OF CASTING MACHINE FOR CONVERTER COPPER.
(Mold Dumping Mechanism not shown.)

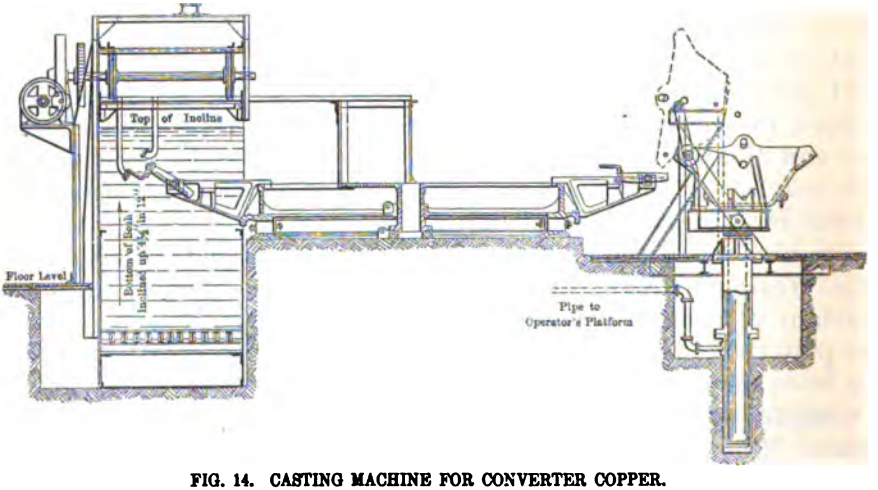


FIG. 14. CASTING MACHINE FOR CONVERTER COPPER.
(Section on line A-A of Fig. 13.)

the mold from front to back, thus changing the point at which the hot copper strikes the bottom and lessening the tendency to destroy the mold. The anodes drop into a water-filled sump or bosh, and are raised

by an endless-chain conveyer which returns overhead instead of (as commonly arranged) beneath. Directly over the discharge end of the bosh are pivoted two trolley beams 30 ft. long, running at their outer ends on a circular track. Each of these has its own air-hoist by which the ingots are picked up and delivered anywhere within the space commanded by the trolley beams. The plant will easily handle 600,000 lb. of copper in 24 hours if it be steadily supplied. Under ordinary working conditions it takes care of 300,000 lb. as delivered from the converters.

Kelly Slag-casting Machine.—This machine is used for casting converter-slag from the converter-plant at the Garfield works.[1] There are eight converters which can yield daily 320 tons of slag containing 3.5 per cent. copper and of a total value of $10@20 per ton. Fig. 15 is a

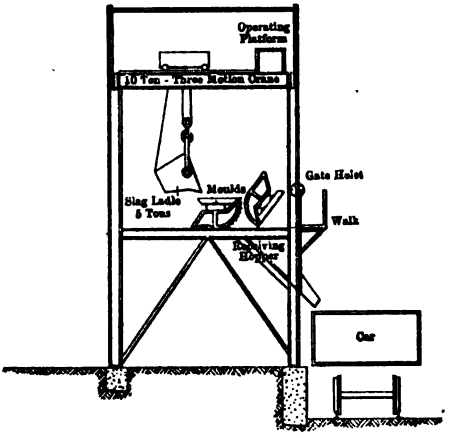


FIG. 15. KELLY SLAG CASTING MACHINE.

transverse elevation of the plant. A five-ton slag-ladle receives slag from the converter, transfers and pours it into a set of molds mounted on rocking frames. One frame is shown in the view receiving slag, and another in discharging position. There are 24 of these molds, each 10 ft. long, occupying a length of 270 ft. The united tray consists of four cast-iron molds fastened to a steel frame, which is in turn rigidly bolted to two curved cast-iron rockers. These rockers are so constructed that the molds assume the normal horizontal position when empty, and a dumping position when full and released. The tray is 10 ft. long, 4 ft. wide and 4 in. deep, and will contain 600 lb. of slag. Each mold is divided into four compartments by ribs 2 in. high. When the tray is released and falls into dumping position, it discharges into a brick-lined steel bin, and from the bin is drawn off into railroad cars, the tray meanwhile falling back

[1] *Eng. and Min. Journ.*, LXXXVI, 610.

into horizontal position. Twelve of these trays will take care of the contents of a 10-ton ladle and one used alternately with the other twelve. Two men per shift attend to all operations, and the labor-cost may be given at 8c. per ton.

Constitution of Copper Matte.—Prof. Chas. H. Fulton and I. E. Goodner review and discuss[1] the various binary systems of the metallic sulphides and of the various metals entering into the constitution of matte. The chief sulphides are Cu_2S, FeS, PbS, and ZnS, while the metals present in a free state, or nearly so, are iron and copper.

Iron sulphide, FeS, is a chemical compound; no subsulphide of iron exists. At 1200 deg. C decomposition begins to take place, and this becomes marked by the time 1500 deg. C. is reached, sulphur volatilizing and a mixture of FeS and Fe remaining. With pure FeS, in the binary system FeS-Fe, the pure sulphide freezes at 1171 deg. C.; the eutectic, containing 84 per cent. FeS and 16 per cent. Fe, freezes at 970 deg. C., while the freezing point of iron is 1540 deg. C. The mattes produced in smelting are derived from the higher sulphides, as FeS_2, FeS and $CuFeS_2$, and since the temperature in smelting exceeds 1200 deg. C., the final product, as regards iron sulphide, is the series FeS-Fe, the amount of iron set free increasing with the temperature reached.

This explains the production of "iron sows," found in semi-pyrite smelting when running silicious slags which need high temperature, the sows being the segregated iron from the matte. It is also probable, that at these high temperatures, iron sulphide, oxidized by the blast to Fe_3O_4, reacts thus: $2FeS+Fe_3O_4=5Fe+2SO_2$ with the production of iron.

Cuprous sulphide, Cu_2S, is the only subsulphide known. It is of the same type as that of FeS-Fe, and like it loses sulphur above its melting point. In the Cu_2S-Cu system, the pure sulphide freezes at 1135 deg. C., the eutectic (3.8 per cent. Cu_2S, 96.2 per cent. Cu) freezes at 1067 deg. C., while pure copper freezes at 1084 deg. C. Cuprous sulphide undergoes a dimorphic change at 103 deg. C. At this temperature copper, which had been in solution, is forced out as the so-called "moss copper" of mattes.

Lead sulphide, PbS, is the only sulphide, no subsulphides being known. The system PbS-Pb shows a curve dropping from the melting point of the pure sulphide, 1120 deg. C., to that of lead (325 deg. C.), no eutectic being known. Like other sulphides, PbS loses sulphur above its melting point, and is markedly volatile as a whole, but the relative loss of sulphur is not so marked as in the cases of iron sulphide and cuprous sulphide.

[1] *Bimonthly Bulletin*, A. I. M. E., Nov., 1908, p. 859.

Zinc sulphide has marked volatility at a high temperature; its melting point has not been determined, but has been computed by extrapolation at 1670 deg. C.

The binary system FeS-Cu_2S (copper-matte). The melting points of the pure sulphides have already been given, the eutectic being 79 per cent. FeS and 21 per cent. Cu_2S, and its melting point 860 deg. C. It has a range extending from 20 to 90 per cent. FeS, due to the existence on the one hand of mixed crystals consisting of 90 per cent. FeS with 10 per cent. CuS, and on the other of a mixture of 20 per cent. FeS with 80 per cent. Cu_2S.

Micrographic examination shows substances called D and E. The first (D), of a blue color, consists of Cu_2S-Cu, or this holding in solution FeS-Fe to the probable extent of 25 per cent. The second substance (E) of a brass color is FeS, in which is dissolved Fe to the extent of 2 per cent. When the matte contains 60 per cent. or over of copper, the FeS no longer shows, since it then becomes soluble in the Cu_2S-Cu; in fact, practically down to 40 per cent. copper, the eutectic is never found. Metallic copper is of common occurrence and may be found in matte of any tenor as thin sheets or wires, which under the microscope show as long thin seams. It invariably occurs in D. It usually increases with the tenor of the matte in copper, and is present in mattes which do not show its presence to the naked eye. Zinc sulphide may be present to a small extent as purplish crystals. (See also THE MINERAL INDUSTRY, XVI, 361.)

The binary system, PbS-Cu_2S. This is an important system in lead-copper mattes, with a eutectic of the composition Cu_2S, 51, and PbS 49 per cent., freezing at 540 deg. C.

The binary system, PbS-FeS. Weidman, who has investigated the system, places its eutectic at FeS, 25.8 and PbS 74.2 per cent., with the freezing point at 782 deg. C., but because of the practical insolubility of the constituents in one another, the eutectic ranges from end to end of the series.

The binary system, Cu_2S-Ag_2S. This consists of an unbroken series of mixed crystals, but has no eutectic. The melting point of Ag_2S is 835 deg C., and that of Cu_2S, 1134 deg. C. as before given. This unbroken series of mixed crystals implies powerful solvent action, and any silver is held in solution in mattes probably as silver sulphide.

The binary system, PbS-Ag_2S. Lead sulphide has an appreciable solvent action on silver sulphide, but less than cuprous sulphide, which when present will take it up to the exclusion of PbS.

The binary system, FeS-Ag_2S. This system has not been investigated, but we know that ferrous sulphide has practically no solvent action on

silver, or furthermore on gold. Metallic iron has however a powerful solvent action on gold, and even more on silver. The collectors of silver and gold in mattes are therefore cuprous sulphide and copper.

Zinc sulphide which may be present to the extent of 3 or 4 per cent. in mattes, is practically insoluble in cuprous sulphide, iron sulphide or lead sulphide; and silver sulphide is insoluble in zinc sulphide, an interesting fact bearing upon the condition of the precious metals in matte.

Copper-iron-lead mattes: These are in structure much like the copper-iron mattes, except that the eutectic is that of Cu_2S-PbS. The presence of lead sulphide in commercial leady mattes reduces the amount of FeS, so that in many instances all of this is in solution in the Cu_2S-Cu. These mattes belong to a ternary system, consisting of the sulphides Cu_2S, PbS and FeS.

Presence of iron and copper in mattes: The range of sulphur in mattes is from 17 to 24 per cent., much less than that called for by considering the metals as combined as the sulphides Cu_2S, PbS and FeS. We must infer therefore that copper and iron are present in metallic form. In ordinary mattes the iron does not show under the microscope, being in solution in the FeS, but, when present in quantity, it shows up plainly. Copper mattes made in semi-pyrite smelting with a slag of 46 to 49 per cent. silica, and therefore formed at a high temperature, show on cooling "moss-copper," even when of as low tenor in copper as 10 per cent. The copper shows itself suddenly when the matte is nearly cold, or has just reached the dimorphic point, 103 deg. C. However moss-copper, visible to the naked eye, is not ordinarily seen in mattes containing less than 30 per cent copper.

Matte as a carrier of precious metals: The substances in mattes which dissolve precious metals and give the matte a value as a collector of silver are metallic copper, cuprous sulphide, and in a less degree, lead sulphide. Metallic copper, cuprous sulphide and metallic iron have a powerful solvent action on gold. Zinc sulphide and iron sulphide have little or no solvent action on silver or gold. Hence the importance of having some copper present in matte to collect the precious metals. In view of the feeble solvent action of arsenical and antimonial speiss, their presence has no significance as a collector of precious metals.

Hydrometallurgy of Copper.

Extraction of Copper from Cupriferous Pyrite at Rio Tinto.—The details of the process have been given in THE MINERAL INDUSTRY, XII, 112, and XV, 288. The following are additional particulars.[1] The washing of the heaps serves three purposes, viz., (*a*) to keep the temper-

[1] *Min. and Sci. Press*, XCVI, 27.

ature down; (*b*) the water and air together acting on the pyrite give the reaction: (1) $3FeS_2 + 2H_2O + 22O = FeSO_4 + Fe_2(SO_4)_3 + H_2SO_4$ and the ferric sulphate reacts with cuprous sulphide thus: (2) $CuS + Fe_2(SO_4)_3 = 2FeSO_4 + CuSO_4$; (*c*) To dissolve the copper sulphate thus formed, and to carry it away to the precipitating-vats. At the vats, besides the fundamental reactions, (3) $CuSO_4 + Fe = FeSO_4 + Cu$, we have other reactions which use up iron: (4) $Fe_2(SO_4)_3 + Fe = 3FeSO_4$; (5) $H_2SO_4 + Fe = FeSO_4 + H_2$; (6) $Fe_2(SO_4)_3 + Cu = CuSO_4 + 2FeSO_4$.

It will be recalled that ferric sulphate is reduced as much as possible to ferrous form by passing the solution from the heaps through a bed of raw pyrite before it reaches the precipitating vats. This will reduce at least 90 per cent. of the ferric salt to ferrous form.

The following figures show the percentage composition of the copper-bearing solution as it enters the vats, as well as its composition on leaving them, the content on leaving being given in brackets: Cu, 0.2715, [0.0019]; FeO, 1.3908, [1.7202]; Fe_2O_3, 0.0610; H_2SO_4, 0.4874, [0.4129]; total solids, 7.0872, [6.9662]; specific gravity, 1.05818, [1.05718]. To precipitate one ton of copper there was used 1.26 tons of iron or 1.4 tons of pig of 95 per cent. Fe. The best results are obtained when the solution is slightly acid, since this accelerates precipitation and prevents the falling out of basic salts of iron from the solution. The solutions are sampled every four hours at different points in their path through the vats, so as to keep a check on the reactions.

The precipitation-vats, containing pig-iron, are arranged in sets or divisions of four to six in series as follows: Division 1 to 2, seven vats each 324 ft. long, 13 sq.ft. in section; 2 to 3, two vats, 230 ft. long, 19 sq.ft.; 3 to 4, three vats, 144 ft. long, 14 sq.ft.; 4 to 5, three vats, 410 ft. long, 14 sq.ft.; 5 to 6 is a parallel system of 700 sq.ft. total section. The copper-bearing solution traverses a mile of vats before the copper is all precipitated. The vats are lined with 3-in. plank, rope-calked, and coated with asphalt. The grade or fall of the tanks increases as hereunder: For the first 40 per cent, of the distance, 1 in 200; for the next 30 per cent., 1 in 150; for the next 20 per cent., 1 in 100; for the remainder of the distance, 1 in 50.

The presence of ferric sulphate or of free sulphuric acid in the liquors causes an unnecessary consumption of iron; hence, when the proportion of copper to acid becomes less, the liquors are made to flow faster, by increasing the grades as above specified. Basic sulphates and other impurities also tend to precipitate when the proportion of acid is high.

Pig-iron is not so effective a precipitant as scrap-iron. The great objection to the former is that the surfaces of the pigs become coated

with copper, and the reactions then proceed more slowly, the shell of copper protecting the iron. Scrap-iron presents larger surfaces to the action of the liquors. When the liquors, fairly strong in copper, first enter the precipitating system the copper is mostly precipitated in coherent form, but later, in a powdery state and accompanied with impurities. The consumption of pig-iron as compared with copper varies inversely as the quantity of copper in the liquors. The temperature of the liquors influences precipitation, it being more rapid in summer. A temperature of 100 deg. F. gives excellent results.

The following points have been established by the practice at Rio Tinto as effecting the saving of the pig-iron: (1) Solutions to be analyzed before and after precipitation so as to know how to control the work; (2) free acid to be eliminated as much as possible; (3) the grade of the vats and velocity of flow to increase as the copper in the liquors decreases by precipitation; (4) the pig-iron to be piled in open order to expose as much surface as possible; (5) a reaction of the liquors, such as would be produced by causing them to tumble through air, is objectionable, since it tends to oxidize the solution; (6) there is a point toward the end of the precipitation when the value of the iron consumed exceeds that of the copper recovered; (7) the warmer the liquors the faster the precipitation.

Precipitation of Copper-bearing Waters at Butte.—At this place, previous to the removal of the reduction works to Helena, Great Falls and Anaconda, the smelters and concentrators accumulated great masses of tailings and slag carrying appreciable quantities of copper, both in oxidized and in sulphide form, computed to contain on an average 2 per cent. of copper. The water from these tailings, entering Silver Bow creek, are quite acid, containing salts of iron and copper, the latter to the extent of 0.02 to 0.074 per cent. It is estimated that the precipitation plant at the Parrot property recovers $25,000 per month; the consumption of scrap-iron is large (four tons per ton of copper precipitated). The practice at Butte of letting the solutions tumble through a series of racks is considered wrong, and the experience at Rio Tinto shows that the scrap should be placed in wide and deep boxes to lessen the velocity of flow. Certainly, the entering and efficient waters should be frequently tested for ferrous and ferric iron, for copper sulphate, as well as for free acid.[1]

Copper-precipitating Plant of Copper Queen Consolidated Copper Company.—Mine-waters, containing 0.015 per cent. copper, are treated at a plant described as follows:[2] The plant consists of a double row of five

[1] *Min. and Sci. Press*, XCVI, 28. *The Mineral Industry*, XVI, 371.
[2] *Eng. and Min. Journ.*, LXXXVI, 853.

tiers of rectangular troughs on three sides and four tiers on the fourth side of a rectangular frame, the outside dimensions of which are 78x108 ft. made of 8x8-in. posts and 6x8-in. stringers. The troughs are 3 ft. wide by 20 in. deep, and for the precipitation of the copper are filled with old tin cans. In the bottom of the trough are set four longitudinal slats, 1x4 in., held up at every 2½ ft. by 2x4-in. cross-pieces, the deposited copper working down by and beneath the slats. The flow can be by-passed through either of the parallel troughs, the other being then cut out for a clean-up. The mine-water, entering by the upper tier of troughs, passes around the rectangle and comes back at a little lower point to the corner where it enters; it then flows down a slide to the next tier, and so in succession through them all, until at the last tier, it leaves by a flume to settling tanks 17x9x9 ft. These last tanks catch a few of the finer copper particles and some sediment. The total length of flow in the troughs is 1750 ft. and the grade or pitch of the troughs increases in the later tiers.

In the upper tiers of troughs the clean-ups are made every second day, but less frequently in the two lower ones. When cleaning up, the flow is shut off from one of the parallel troughs, the trough drained, the cans or scrap-iron washed clean with a hose, and the washings run to settling tanks; more cans are added to the trough as needed, and the flow turned in again. Once a month the trough is emptied of its contents and a thorough clean-up made; the troughs are then filled with a new supply of cans.

The settling tanks, which receive the sediment washed from the tin cans, are 25x40x2 ft. In these tanks the washings are allowed to settle for two hours when the clear supernatant water is decanted. The settled residue is a thick mud, yellow and containing, not only copper, but also iron oxide, scrap-iron, silica, etc. Once a month the wet mud from the settling tanks is bailed out with buckets, emptied into a tram-car, and transferred to drying tanks 18x25x2 ft. Here it stands for a month to partly dry, after which it is shoveled up and loaded into railroad cars for shipment. This product when shipped still retains 40 per cent. moisture. The dried sample assays 35 per cent. Cu; 6, SiO_2; 17, Fe; 13, Al_2O_3 and 1.5 per cent. S. When the flow is slow, 90 per cent. of the copper contained in the water is recovered as precipitate, and a little less with active flow.

Some Future Problems in the Metallurgy of Copper.

By Edward Dyer Peters.

For more than 25 years copper metallurgists have been so engaged in improving the mechanical features of their processes, and in trying to keep their plants somewhere near to the standards set by leaders

in the profession, that they have had little time to give serious attention to the perfecting of details. A generation that has seen the capacity of copper blast-furnaces increase from 30 tons per day to 3000; that has watched the reverberatory expand from 12 tons to 300; that has witnessed the evolution of the mechanical roaster, and the displacement of the Swansea blister process by the bessemer converter of the iron-master; such a generation has been too much occupied in inventing, in adapting, and in constructing, to give to every detail of the new conditions that close and intimate study which is demanded by modern industrial ideals.

The cheap and rapid handling of vast quantities of material is the most striking feature of our present metallurgical operations. Until modern mechanical appliances were introduced, the enormous furnaces and apparatus of the present period would have been impossible.

Retrospection based upon personal observation may be permitted, even in a brief scientific essay, if it be used strictly for the purpose of suggesting future improvements. Availing myself of this somewhat melancholy privilege, I desire to review rapidly certain of the more striking advances in copper metallurgy which have been made during the last generation, more particularly with the view of showing why they came about as they did.

Roasting belongs properly to the smelting operations, being often the first step in preparing a sulphide ore for fusion. Apart from the heat required to ignite the sulphides and to maintain the temperature suitable for the oxidizing reactions which we desire to effect, there is only one fundamental requisite for rapid and efficient roasting: namely, extensive sulphide-surface exposed to the action of oxygen. This is a somewhat far-reaching demand, and requires: (*a*) Small ore particles. (*b*) A vigorous air current, which shall constantly bring fresh oxygen to the surface of the ore, and shall also sweep away the residual nitrogen of the spent air, and the sulphur dioxide gas which would dilute the atmosphere in contact with the ore so that its oxidizing power would lose its efficiency. (*c*) Frequent stirring of the ore, for, after the removal (approximately) of the first atom of sulphur from the pyrite—which atom is driven off by heat alone—the oxidation of the ore proceeds with discouraging slowness so soon as the upper layer of sulphide particles has become partially oxidized.

The first two of these requisites were obtainable in our old-fashioned hand-rabbled roasting furnaces. The last one was so far from being obtained that the capacity of the furnace, in my opinion, was never over one-fourth of what it might have been had it been possible to fulfill this requirement. Placing it in the form of a concrete statement—without any attempt at accurate figures—I am inclined to think that my own

roasting furnaces used to do reasonably speedy work for about five minutes; then taper off rapidly for some seven minutes more; then wait, at less than 20 per cent. of their maximum efficiency, until the next period of stirring arrived.[1]

The invention and the success of the mechanical roasting furnace has to do almost entirely with this single point: how to stir the ore frequently and efficiency. This weak link in the chain of operations has been made strong and complete. Mechanical roasters have been—and still are—among the most important agents in the modern treatment of copper ores. They still, however, have two weak points which may possibly, in the course of time, relegate them to the long list of appliances which have served a useful purpose, and have finally been retired to make way for still more valuable inventions. These weak points are: (*a*) The ordinary roasting furnace yields its product in a pulverized form, unsuitable for direct smelting in the blast-furnace. (*b*) It wastes most of the natural fuel of the ore, there being sufficient heat capacity in ordinary sulphide fines to effect its own smelting as well as its roasting. In other words, the heat evolved by burning the sulphides during the process of roasting ought to be applied to their complete fusion. This is a time-worn statement that I shall use as a text later. What I wish to point out in this immediate connection is that this question of a more frequent stirring of the ore was the one glaring fault in our roasting practice of 30 years ago, and that, really, almost the entire energy and ingenuity of the inventors and users of automatic furnaces were focussed upon this single point. All the wealth of science and of modern mechanical skill has been brought to bear upon the exposing of fresh particles of ore to heat and oxygen as frequently as is needful—and has been successful.

Not many years ago the chief difficulty in smelting copper ores in the blast-furnace was the burning-out of the fire-brick lining of the shaft. If the furnace was driven at all rapidly it needed relining in a few weeks, and often in a few days. This was the weak spot in the blast-furnace department.

An ingenious man took advantage of the extraordinary heat-conductivity of copper to construct the critical portion of his furnace of slabs of the crude metal, sprayed with water on the outside. To avoid the danger and discomfort of this plan, a second ingenious man circulated his cooling-water through a narrow compartment enclosed between tight metallic walls—and the water-jacket furnace came into existence. With his mind thus freed from the constant fear of melting down his furnace, the metallurgist could now give his undivided attention to the next weak

[1] This rough estimate is based upon the examination of a considerable number of samples taken for the purpose of determining how often hand-roasters required stirring. The examination was conclusive in showing that they needed stirring very much more frequently than it was feasible to do it.

point in his process. This was, undoubtedly, the slow filling-up of his furnace crucible with iron sows, barium sulphide, zinc sulphide, and a gradual never-ceasing accumulation of minute quantities of various substances which the furnace had failed to digest.[1] These substances, if excreted as fast as they form, are comparatively harmless; but, if retained internally, lead irresistibly to the complete filling of the crucible and the end of the campaign.

The water-jacketed shaft enabled us to drive the furnace with such rapidity that we could produce a stream of molten material of sufficient volume to retain its heat and liquidity in an exterior forehearth. The interior crucible was eliminated; the accretions were transferred from the inside to the outside of the furnace; and the apparatus had now taken on its modern form in all essentials excepting size. The most skilled fireman would find it difficult to keep up the process of combustion with, say, six fragments of anthracite coal in an air-tight stove. With 600 fragments of similar coal, he might have to exercise equal care to prevent the melting of the stove. I am not sure, after all, that this was not the very weakest point in our old furnaces, both blast and reverberatory. They were too small. The proportion of wall surface was too great for the little mass of glowing material inside. The last *hundred* or two degrees of heat is what does the actual melting. The first *thousand* degrees is only getting ready for it. But, if we enclose a small heated mass within a large surface of cool walls, radiation and convection are so great that we cannot attain that last hundred or two degrees without an excessive expenditure of time and fuel. This point is especially noticeable in reverberatory smelting, and the brilliant results obtained by Mathewson at the Washoe smelter are due in great measure to the recognition of the fact that *mass* is one of the most important accompaniments of satisfactory smelting.

This is not the place to dwell upon the mechanical ingenuity which has been expended in enabling us to provide and handle the vast quantities of material demanded by the modern furnace, and to control the resulting products. I have used these illustrations to point out that our most important improvements in smelting have resulted from the elimination of certain radically weak points in the chain of operations. It is then evident that the recognition of other weak points which still remain will be an important aid for the future; and my present purpose is to indicate certain conditions which are not only unsatisfactory, but which seem to me susceptible of improvement along familiar and well-tried lines.[2]

[1] I am, of course, referring to the days before the introduction of the forehearth, and when we used an interior crucible, from which the metal was tapped at long intervals.

[2] I need scarcely say that I claim no originality for these suggestions.

Improvements in the mechanical concentration of ores, and the growing appreciation of the safety and advantage of large low-grade deposits are resulting in a large production of pyritic concentrates. It is probable that this product will continue to increase, and that its economical treatment will engage the attention of many skilled metallurgists. One of the most common methods, at present, of treating these concentrated sulphides is by roasting in mechanical furnaces, and smelting the roasted ore in reverberatories. This plan has the two serious objections already noticed: (*a*) The heat derived from the oxidation of the sulphides during the roasting process is mostly wasted. (*b*) The roasted product remains in pulverized form, and, consequently, has to be smelted in reverberatory furnaces.

This is not the place in which to discuss the comparative advantages of blast- and reverberatory-furnaces for the smelting of copper ores. I believe firmly that, under anything like ordinary conditions, and assuming that the ore is mainly in lump form, the blast-furnace is decidedly the more advantageous of the two. It is self-contained, cheap, easy to erect, easy to run, and requires few repairs. To save space, however, I am willing to waive all these claims, and to assume—for the moment—that the reverberatory can smelt copper ore as cheaply, easily, and efficiently as the blast-furnace. Let us assume, however, that the blast-furnace smelter is to receive at least one-half of his fuel supply without any cost to himself. This would certainly place the blast-furnace outside of all competition. It seems to me that, with the very large supply of sulphide concentrates which will soon be available at certain centers, a smelting charge might contain more than enough sulphur and iron to cut the coke bill in halves, providing that this natural fuel is expended judiciously, and always assuming that the sulphide concentrates can be put into lump form, suitable for the blast-furnace.

We know already how to do reasonably good pyrite smelting; so our entire attention may be devoted to our one weak point, viz., the pulverized condition of the sulphides. One of the most important questions for the present-day metallurgist to determine is, whether he can convert his finely divided sulphides into lump form without too serious sacrifice of their heat-producing power, and without too heavy an expense. Two familiar methods of effecting this result are: (*a*) Smelting the raw sulphides in a reverberatory into a low-grade matte. (*b*) Briquetting the raw sulphides with a few per cent. of lime. Both of these methods are too expensive, and the second one yields a product which is too weak for the blast and ore-column of the high furnace which we intend to use in smelting.

Quite recently a third method has been introduced which promises distinct advantages, and which, if it fulfils the claims of its adherents, appears to solve the difficulty. I refer to the roasting of pyritic ores by the heat derived from their own oxidation, and in such a manner that they agglomerate into a half-fused porous mass which is a peculiarly suitable material for the blast-furnace. Whether this process of oxidizing-agglomeration shall be carried out by actual pot-roasting or by the ingenious continuous method invented by Mr. Dwight has no bearing upon the present argument. It is enough to know that it is feasible, and that it can be accomplished at a moderate cost.

Most of the work hitherto done in this direction has been with the intention of removing as *much* of the sulphur content of the concentrates as is possible. It should not be difficult to modify the process so as to oxidize as *little* of the sulphur and iron as is compatible with the thorough agglomeration of the sulphide fines. If this can be accomplished, we should eliminate the weak point to which I have just referred. We should have for our blast-furnace an ideal material: a porous half-agglomerated compound of (mostly) iron and sulphur.

In order to determine how much of the fuel-value of the original sulphides still remains in this product, and how much of it we have lost by the pot-roasting operation, we must review the part which iron sulphides play in pyrite smelting.

A small proportion of coke—0.5 to 2 per cent.—is used in all pyrite smelting, even where the charge contains the highest possible proportion of pyrite. The reason for this apparently inconsistent practice is discussed at length in the familiar literature of the subject. So long as this addition of coke does not exceed a certain maximum (variable) limit, it does not affect the essential oxidizing reactions which are confined strictly to a zone extending from just above the tuyere level to an horizon several feet above this point. As there is no available oxygen left above this zone of combustion (focus); and yet, as a moderate amount of coke may be completely consumed before it reaches the focus at all, it is evident that the coke obtains its oxygen from some source other than the air-blast. Sticht investigated this point at the Mount Lyell furnaces, and found that the coke obtained its oxygen by reducing sulphur dioxide to sulphur.

If, now, more coke is added than can be burned in the manner just described, the excess carbon will descend into the smelting zone, or focus, and will consume the oxygen of the blast which ought to be used in burning iron sulphide. This modifies profoundly the regular operation of the process, and results in two evils: (*a*) A low-grade matte (due to the dilution of the regular product by the iron sulphide which has been

robbed of its oxygen by the encroaching coke). (*b*) An abnormally silicious slag (due to the fact that a certain amount of iron which ought to have been oxidized and combined with silica has now gone into the matte as iron sulphide).

We see, therefore, that here are two distinctly different varieties of smelting. Much of the argument which has arisen among writers on this subject is due to the fact that they were talking about two different operations, and that much of what is perfectly true about one of these operations is entirely false if applied to the other. We have one kind of smelting in which so little coke is used that it does not affect the oxidizing power of the focus. This may be called "True Pyrite Smelting." We have a second kind of smelting in which so much coke is employed that the oxidizing effect of the focus is diminished. This operation may be termed "Partial Pyrite Smelting." As I desire to discuss the behavior of iron pyrites unobscured by the influence of carbonaceous fuel, I will assume, for the moment, that we are speaking solely of "True Pyrite Smelting."

By far the most common and important sulphide with which the metallurgist has to deal in pyrite smelting is pyrite, FeS_2. The atmosphere of the pyrite furnace above the focus is neutral or reducing.[1] As we desire to study the fuel-value of FeS_2 in the process of pyrite smelting, we must first determine how pyrite behaves in a non-oxidizing atmosphere when subjected to a gradually increasing temperature, this being the situation in the upper portion of the pyrite furnace.

Exposed to a dull-red heat (without oxygen) pyrite loses about three-sevenths of its sulphur, and becomes a compound analogous to pyrrhotite, perhaps Fe_7S_8. This compound melts at about 925 deg. C., and with increasing temperature gradually loses a little more of its sulphur, until —at about 1200 deg.—it becomes FeS. As this is the lowest combination of iron and sulphur which can exist at these temperatures, it is evident that, if any further sulphur be expelled, some of the iron must remain uncombined with sulphur. This is, in fact, the case. As it approaches the intense heat of the focus, the FeS becomes a compound somewhere between FeS and FeS,Fe. Thus, considerably more than one-half of the sulphur content of the pyrite is wasted—so far as useful heat-production is concerned.

The true pyrite furnace runs on a very narrow margin of heat. Leaving aside the trifling addition of coke, the pyrite has to furnish heat enough not only to melt itself, but also to melt the required silica, as well as such lime, alumina, and other earths as may be present in the

[1] This point is established by Sticht's well-known analyses of the Mount Lyell furnace gases, and substantiated by the behavior of all normal-running pyrite furnaces.

ore, or as may be added to diminish the specific gravity of the slag. Sticht points out that there may be an opportunity to save a portion of the unfortunate waste of sulphur which has just been described; and that, small as this saving may appear, it occurs at such a vital point in the smelting operation that it would be likely to broaden considerably the present narrow margin of safety. He recognizes that we must inevitably lose one-half of the sulphur content of our FeS_2; but that, after this loss has been effected, the resulting FeS is a tolerably stable compound, and requires an appreciable time and a decided rise of temperature before more of its sulphur is driven off. He points out that at the melting point (950 deg.) of FeS the tension of the sulphur vapor may be held in check by a slightly increased pressure within the shaft and that, with increased blast and higher furnaces, the decomposition of FeS might be sufficiently retarded to permit it to reach the zone of oxidation, and thus exert its full value as a fuel.[1] This is a point, however, which belongs to the refinements of true pyrite smelting, and has little bearing upon the matter at present under discussion.

What I have been endeavoring to show is, that one-half of the sulphur-content of our pyrite concentrates is valueless as a fuel for smelting purposes. May we not utilize this valueless atom of sulphur to accomplish much of the agglomerating roasting of the pyrite fines? Careful experiments alone can determine this point; and while I do not for a moment believe that we can ever succeed in pot-roasting pyrite fines in so economical a manner that we shall burn exactly the one free atom of the sulphur belonging to the pyrite, and then stop—leaving our product in the shape of agglomerated FeS—I feel strong hope that we may be able to approximate this result, and that we may thus leave so large a proportion of unoxidized iron and sulphur in our pot-roasted sulphides that the amount of coke used in smelting them will be greatly reduced.

Owing to the absence of suitable ores, true pyrite smelting must always remain an exceptional operation. I have, however, felt it necessary to review rapidly its principal features in order to clear the ground for the modified process which promises to be of such practical utility: namely, partial pyrite smelting.

Let us examine the interior condition of the furnace in true pyrite smelting in order to compare it with the state of affairs which we shall bring about in partial pyrite smelting, when we add so much coke that it is not entirely burned by the oxygen of the sulphur dioxide, but, in part, descends unconsumed to the focus of the furnace, and begins to appropriate the oxygen of the blast. In true pyrite smelting the furnace shaft is filled with a skeleton of quartz fragments cohering only slightly

[1] Sticht. "Ueber das Wesen des Pyritschmelsverfahrens," *Metallurgie*, 1906.

at their surfaces, and practically infusible. This porous silicious skeleton may be compared to a vertical glacier standing with its foot in the fire, while its head is constantly renewed at the charging-door. It melts at the bottom; not primarily from the heat of the smelting-zone, but from the fluxing action of the FeO, which is formed with extraordinary rapidity as the streamlets of liquated FeS rush into the oxidizing atmosphere of the focus. One pound oxygen will oxidize x pounds of FeS. Two pounds oxygen will oxidize $2x$ pounds of FeS. Consequently, the rapidity with which the column of silica is eaten away at the bottom depends upon the volume of the blast; for not an ounce of silica will be removed until it finds its corresponding equivalent of FeO; and the FeO can only be formed by the oxygen of the blast.

It is misleading to regard the pyrite furnace solely as a smelting-furnace; to all intents and purposes it is a bessemer converter. The shower of melted FeS represents the bath of matte of the converter process, and the skeleton of quartz represents its silicious lining. The FeO takes up from this quartz skeleton exactly the quantity of silica that it requires to produce the silicate whose heat-formation corresponds to the momentarily-prevailing temperature, and then stops. This silicate is chemically inert, and no coaxing or driving can induce it to take up any more silica. If we persist in adding quartz faster than the FeO can digest it, the excess silica remains untouched until the furnace is blocked with it.

When the foregoing facts are appreciated, it will become evident that, in this kind of smelting, a hot blast is not only superfluous, but actually harmful.[1] There is no better fuel than FeS,Fe, burned at exactly the point where it is most needed. If the heat in the smelting-zone is insufficient, it is increased at once by burning more pounds per minute of FeS,Fe. A pound of coal devoted to increasing the blowing capacity will yield far greater heat-returns than a pound of coal consumed in preheating the blast. This is the result of long experience at Mount Lyell, and the pyrite work there never reached its full satisfactory development until the hot-blast stoves were discarded, the blowing capacity doubled, and the furnace hight increased to correspond with the new conditions.[2]

In turning to the commercially more important partial pyrite process, we must abandon our simple mixture of cupriferous pyrite and quartz, and consider what may be the principal classes of ore we shall be called upon to treat. If we are to do *any* kind of pyrite smelting, we must have

[1] As a matter of fact, it is likely to cause a hot top, a cold focus, and a low-grade matte.

[2] The Mount Lyell pyrite furnaces treat an ore of between 2 and 2½ per cent. copper. A matte of nearly 40 per cent copper is produced at a single smelting, the concentration being 18 to 20 into one. The conditions permit of using about two parts of heavy pyrite to one part of silisious material (containing about 70 per cent. silica), and 95 per cent. of the iron content of the charge is thus burned, and sent into the slag, at a single operation. R. Sticht's V.-Presidential Address to the Australasian Association for the Advancement of Science, January, 1907.

a certain amount of iron sulphide to begin with and, recognizing the scarcity of massive pyrite or pyrrhotite containing copper, gold, or silver, I have already assumed the presence of a large supply of tolerably clean value-bearing pyritous concentrates, which may be agglomerated by some form of pot-roasting into a porous mass of (approximately) FeS, without much wasteful oxidation. These agglomerated concentrates are the mainstay of our operations from a metallurgical standpoint, although it is quite possible that subsidiary dry silicious, or earthy, ores may surpass the pyritous concentrates in commercial importance.

These pyritous concentrates contain, probably, the copper that is needed to make our resulting matte an efficient collector for gold and silver; they may also carry a small, but welcome, value of precious metals themselves. They contain much of the iron which we shall need as the fusible base of our slag; and they contain a certain potential of heat which we intend to utilize to its fullest extent. The agglomerated pyrite concentrate is, then, the backbone of the enterprise; but, before we can proceed to the wholesale acquisition of miscellaneous outside ores, we must secure one other constituent for the charge, which is just as indispensable, though not so striking in its action, as the iron sulphide.

This second essential constituent is *free silica*. It is not enough to furnish the iron sulphide simply with a silicious ore containing even a heavy excess of silica above the amount required to flux its own bases. Some of the feldspars, for instance, contain 68 per cent. of silica, and could easily spare two-thirds of it, and still have more than enough left to make a suitably acid slag with their own bases. But they would have but a feeble effect in fluxing FeO in the pyrite process. Sticht states this point forcibly in the paper from which I have just quoted. He says: "The determining influence of silica is not often referred to; but its action follows fixed laws which one cannot over-ride, and which narrowly circumscribe the pneumatic processes. What we call *silica* is in reality simply the anhydride of an exceedingly strong acid which becomes active only in an igneous condition, but then seeks its base the same as any other mineral acid. As far as its chemical relations are concerned under the conditions existing in the converter (or pyrite furnace—E.D.P.), that element (iron) behaves exactly the same as it does at ordinary temperatures in the presence of ordinary free acids. Hydrogen sulphide will not precipitate iron in the presence of hydrochloric acid, for instance; and, similarly, iron will not form sulphides at high temperatures in the presence of free silica. Conformably, therefore, sulphur, already combined with iron can be driven out and burnt in an oxidizing atmosphere, when uncombined silica is present—the iron and silica uniting."

If, then, the silica in the smelting mixture is in combination with alumina or lime or other bases, its affinities are already satisfied. It holds out no inducement for the iron to break its connection with the sulphur; consequently, the iron enters the matte as ferrous sulphide. This brings out clearly the fact that, when estimating the possible constituents of our furnace mixture, we must (at least, mentally) divide its silica content into two distinct portions: (*a*) Free silica, upon which we shall depend to decompose ferrous sulphide (in the presence of oxygen) and form a ferrous silicate, of a silicate-degree corresponding to the temperature of the smelting zone, normally approaching a monosilicate and having a formation temperature of about 1200 deg. (*b*) Combined silica, which has no active effect in pyrite smelting.

Having provided for ores carrying a sufficient supply of free silica, and possessing already an ample stock of ferrous sulphide, our metallurgical horizon is now clear, excepting for the fact that a small addition of earthy bases is desirable to lessen the specific gravity of the heavy iron slag. As partial pyrite smelting is a process in which we are certain to have a large excess of these earths, this point need not delay us. We are now ready to proceed to the remaining (metallurgically non-essential) portion of our smelting mixture. This is likely to comprise the portion of our tonnage upon which we shall count for our main dividends. We may not hope to obtain much profit from the pyritous concentrates which have thus far received our chief attention. Such material as this is condensed flux and fuel, and, as such, is so desirable that the smelter may think himself fortunate if he can obtain a steady supply of such choice material on a margin that shall defray the cost of handling, and cover its own metallurgical losses. The miner will regard a reduction of smelter tariffs as a part of his legitimate profits, and as one of his main incentives to the exploitation of such low-grade material.

The profits of the smelter will be derived from ores which are hard and expensive to treat, and which, consequently, are expected to pay a comparatively high tariff. Such ores are the ordinary dry silicious, or earthy, non-concentrating gold or silver ores which are so common in so many mining districts, and which will become so much more common when the smelters can offer better prices for them. These ores usually contain more or less scattered pyrite, much quartz, blende, alumina, and other earths, and are generally so silicious that they are unwelcome to the ordinary smelter; yet, they are often the chief ores of many large districts.

Recognizing that if we intend to treat any considerable tonnage of these undesirable ores we shall be introducing into the furnace a far greater proportion of silica than the true pyrite furnace (with its mono-

silicate slag) can digest; recognizing, also, that (aside from the small amount of disseminated sulphides in these dry ores) we shall be loading our pyrite process with an inordinate proportion of dead material which produces no heat itself; it is plain that our original iron sulphide cannot furnish anything like the heat required to melt this new burden, and that we must have recourse to carbonaceous fuel to supplement our pyrite fuel. In other words, we abandon true pyrite smelting, and enter upon the domain of partial pyrite smelting: a process in which the aims, the means, and the results are all so different from true pyrite smelting as to require a considerable rearrangement of ideas.

One of the most notable differences in the aims of the two methods is that, while in true pyrite smelting we desire to slag as much *iron* as possible with a given weight of silica, in partial pyrite smelting we almost always strive to slag as much *silica* as possible with a given weight of iron.

Assuming—as is usually the case—that highly silicious material forms a considerable proportion of our available ore supply, we must first determine the lowest percentage of iron (or manganese) that our slag must contain to be reasonably fusible. Twenty per cent. of FeO in the slag is about as low as one would dare to start with, expecting, no doubt, to diminish this amount by several per cent. as the men become more skilled in the management of the furnace. This will reduce the proportion of heat-generating materials in the mixture to a low point, and will require the addition of an amount of coke which will interfere seriously with the oxidation of the sulphides and the consequent degree of matte concentration. This interference, however, is not fatal to the process. The coke, certainly, has the first call on the oxygen of the blast, and is consumed pretty thoroughly before the sulphides can begin to burn to any considerable extent. This means that, instead of two zones of action—as in the pyrite furnace where we have the upper, or neutral, zone and the lower, or oxidizing zone—we shall find it convenient to consider the shaft of the partial pyrite furnace as divided into three zones: (*a*) The upper, or neutral, zone—free from oxygen, as the air-blast before it reaches this horizon has had to run the gauntlet of both melted sulphide and glowing coke. (*b*) The middle, or coke-burning, zone—representing the upper horizon of the focus, and thus crowding the sulphide-burning horizon lower down. (*c*) The lower, or bessemerizing, zone—where the coke has been, in great measure, consumed, so that the air, as it streams through the tuyeres, has some opportunity to burn the FeS before ascending into the layer of glowing coke which stops its further influence upon the sulphides.

It will be noted, therefore, that the time and space available for the oxidation of FeS are exceedingly limited, and are regulated by the depth

in the shaft to which fragments of unburned coke can descend. It is also evident that any addition to the normal coke-charge (unless accompanied by increased blast, greater hight of furnace, and other suitable modifications) will have a profound effect upon the amount of FeS that will be burned. The sulphide-burning zone is all too limited at best; and if it be still further contracted by unconsumed coke pressing down into its upper boundary, the evil result will be felt in a surprisingly short time. The matte will become greater in quantity and poorer in quality, and the slag, robbed of its normal supply of iron, will become more silicious.[1] In order to avoid this evil, and thus to effect a proper degree of concentration by sending his iron into the slag instead of into the matte, the partial pyrite smelter is always trying to run with as little coke as possible. Consequently, he has no surplus heat to fall back on, and his furnace is always in a somewhat critical condition; although I am convinced that the employment of much larger furnaces, and the careful bedding of his ores will improve his situation materially. If he adds a little more coke to overcome some slight irregularity, his matte increases in quantity and decreases in quality, while his slag becomes dangerously silicious. If he lessens the coke, his matte improves, to be sure, but there is not heat enough generated to melt the great mass of inert earths with which the charge is burdened. It is under these circumstances that a moderate heating of the blast has its legitimate sphere. Only a little more heat is needed to make conditions safe and comfortable, and this may be supplied (without the fatal reducing influence) by a moderate preheating of the blast.[2]

One more point still remains for consideration before closing this discussion of partial pyrite smelting. Having learned from true pyrite smelting that the pyrite furnace "chooses its own slag," and that this slag approaches pretty nearly a ferrous monosilicate, how are we going to adapt it to the conditions just referred to, where we desire a slag at least as acid as a bisilicate, and containing only about 20 per cent. FeO? It is quite true that, even in the partial pyrite furnace, the ferrous sulphide, in the presence of oxygen and free silica, will form its proper ferrous monosilicate. Thus far we permit the process its freedom of action; but the moment after this ferrous silicate is formed we subject it to arbitrary and unnatural conditions which rob it of all further freedom of action and reverse the laws which it normally follows.

The presence of glowing coke in the focus of the furnace modifies all

[1] S. E. Bretherton, while superintendent of the Val Verde smelter a few years ago, first called my attention to the extreme delicacy of the balance which prevails in this type of smelting. If I remember rightly, he was only using about 3 per cent. of coke, and was making the high rate of concentration required to produce a shipping matte from gold and silver ores low in copper. The addition of one or two scoops of coke to each charge was soon followed by a great increase in the volume of matte, and by a dangerously silicous slag.

[2] In a plant large enough to warrant the outlay, it seems to me that Gayley's dry-blast, so astonishingly efficacious in iron furnaces, might be a great aid to the partial pyrite smelter.

Notes on the Hydrometallurgy of Copper.

By Ottokar Hofmann.

I. The Hunt & Douglas Copper Process in its Application to Work Copper Matte on a Large Scale.

The Hunt & Douglas process is based on very interesting chemical reactions, but so far as I am aware its application on a large scale has not previously been described. I had the opportunity to work the process on a commercial scale at the plant of the Consolidated Kansas City Smelting and Refining Company, at Argentine, Kan. In a short article, *Trans.*, A.I.M.E., Vol. XVI, T. Sterry Hunt describes three hydrometallurgical methods for the extraction of copper from its ores and products, devised by himself and James Douglas. The method which was introduced at Argentine he described as follows:

"In the second method referred to above, beginning with a neutral solution of copper sulphate, there is added enough common salt, or other soluble chloride, to convert the copper present into cuprous chloride (58.5 parts sodium chloride to 63.4 parts copper). Through the clear hot solution is now drawn or driven sulphurous acid gas derived from roasting sulphurous ores; this serves to convert the dissolved copper into insoluble cuprous chloride, with liberation of previously combined sulphuric acid, and the generation of one half as much more of the same acid by the oxidation of the absorbed sulphurous gas. The clear acid drawn from the cuprous chloride is then saturated with copper from oxidized ores, and the precipitation by sulphurous gas is repeated indefinitely. The insoluble cuprous chloride obtained in this process may be either reduced to the metallic state with iron, or treated with milk of lime; in the latter case cuprous oxide and chloride of calcium are formed. The resulting chlorides in either case serve for the chloridization of any further copper which may be dissolved. By the use of a solvent containing only a small portion of soluble chloride, any silver present in the ores is chloridized, but remains undissolved in the residue and may be extracted therefrom by solution, by amalgamation, or by smelting."

The material treated at Argentine was a lead-copper matte, averaging 39.55 per cent. copper; 11.26 Pb; 19.90 Fe; 1.88 Zn; 1.01 Mn; and 21.43 S. The silver varied from 200 to 300 oz. per ton. This matte was first crushed by a rock-breaker and then pulverized in a Krupp ball mill to pass a 50-mesh screen.

The roasting was done in two Pearce two-hearth furnaces. On the upper hearth the temperature was kept as low as the heat developed by the oxidation of the sulphur permitted. No fire was applied except after the ore had almost passed the whole circle of the hearth and come near

the slot through which it dropped to the lower hearth. There a very gentle fire was maintained to prevent the temperature from falling too low. In roasting sulphide ores or products, which owing to their nature and richness in sulphur have to be roasted first in their own heat and then finished at a gradually increasing temperature with the assistance of extraneous heat, care has to be taken that the temperature is not allowed to sink too low before the extraneous heat is applied; in such a case it is rather difficult to raise it again to the desired point and that requires considerable time and fuel. There is often a loss of silver by volatilization, which otherwise would not occur. I obtained the best results by regulating the roasting on the upper hearth so that the material commenced to ignite when it had moved about 8 ft. from the point at which it entered the furnace. By observing this precaution, the roasting was so much advanced by the time the material had reached the drop-slot that the oxidation of the sulphur did not create more heat. This point in roasting is readily observed by stirring the charge; if the particles thrown to the surface brighten and remain so for a short while, the oxidation still evolves heat; but if these particles are of a dead red color and begin to darken immediately, it is an indication that, in order to continue the oxidation, heat must be supplied. I found it to be of the greatest importance to have the roasting well advanced when the material left the upper hearth. When this was neglected, and the speed of feed increased so that the matte, after having dropped to the lower hearth, still created heat by oxidation, the finished product was invariably insufficiently roasted.

I endeavored to maintain a gradually increasing temperature on the lower hearth up to the point of discharge. However, I did not succeed in accomplishing this until I inserted an additional fire-place, and then not as well as I desired. The Pearce roasting furnaces of standard construction have not enough fire-places on the lower hearth to permit a fine and delicate roasting. The fire-places are too far apart, and the material which attains a red heat in the immediate neighborhood of one fire, cools below red heat before it reaches the next, where it is again heated. This condition is undesirable; it would be a valuable improvement of the Pearce roaster if the lower hearth was provided with a greater number of fire-places, smaller but closer together. By this arrangement more moderate fires could be maintained and the temperature regulated to suit the material under treatment. The insertion of only one additional fire-place caused a marked improvement in the final results.

In order to regulate the final heat, tests were made at intervals of the material before and after it passed the last fire. The samples were sifted and washed in a small dish to determine if any cuprous oxide had formed.

The presence of cuprous oxide is readily detected by its pink color. It often happened that although the material was free from cuprous oxide before passing the last fire, it could be plainly detected after passing it. This was always an indication that the last fire was too hot. It was important to avoid this condition, because by too high a temperature cupric sulphate, of which quite a percentage was formed during roasting, was decomposed into cuprous oxide and sulphuric acid, and the matte was discharged before the cuprous could be oxidized to cupric oxide. This test had to be made, not in order to prevent the loss of acid, because in the Hunt & Douglas process more acid is made than needed, but for the reason that when cuprous oxide is treated with dilute sulphuric acid, only half of its copper can be dissolved as cupric sulphate; the other half changes into metallic copper, which being insoluble, will remain in the residues.

Even with the greatest care it is impossible to roast a leady matte free from small lumps. They form in the very early period of the process before any additional heat is used, but as a rule, being usually porous, they will be found well roasted. These lumps, however, are very undesirable in the subsequent operation, as they retard the solution of the cupric oxide in dilute sulphuric acid. The roasted matte, therefore, had to be crushed. From the roasters the matte was automatically conveyed to revolving cooling tables, then fed into a ball mill with 50-mesh screens. From the ball mill it was elevated and conveyed to the storage bins.

Solution.

The dissolving of the cupric oxide had to be done in agitating tanks, it being impracticable to conduct the operation in tanks with filter bottoms. When roasted copper matte is brought in contact with dilute sulphuric acid, or even water, it cements and hardens to such an extent that it cannot be handled with shovels, if not previously loosened with picks or bars. This hardening of the material prevents to a great extent the free percolation of the solution; this causes much delay and also makes the discharging of the tank a rather difficult task. Even while charging an agitating tank, it is necessary that the acid solution be kept in lively motion and that the matte be introduced in a gradual stream and not charged with a shovel; otherwise hard chunks are formed.

The dissolving was done in wooden agitating or stir tanks 12 ft. in diameter and 6 ft. deep, provided with a strong hard wood propeller, which entered and was driven from above. These tanks were about two-thirds filled with acid solution, containing 9 to 10 per cent. free acid, to which some wash water was added. This acid solution, which resulted in the process, always contained 2 to 2½ per cent. copper.

The agitator was set in operation and a jet of steam introduced through a lead pipe entering from above and fastened close to the side of the tank. The roasted matte was then brought from the storage bins in cars, which were half covered and provided with a slot through which, by tilting the cars, the matte could be uniformly charged into the tank. The addition of matte to the dilute sulphuric acid produces considerable heat which aids the solution of the cupric oxide and diminishes the amount of steam required to maintain the pulp at the desired temperature.

After a certain amount of matte had been added, the pulp was frequently sampled; these samples were filtered and the filtrate tested for free acid. When the solution was nearly neutral the charging of the matte was stopped, but the agitation continued until the solution became neutral or almost neutral. This operation was performed with care in order to avoid an excess of matte, which would have enriched the residues with copper. In mixing the acid solution with wash water, care was taken to have enough free acid present, so that the resulting solution would contain from 6 or 7 per cent. copper.

The neutral solution, together with the residues, was discharged through an outlet near the bottom of the stir tank, into a large lead-lined cast-iron pressure tank; thence, under an air pressure of 40 lb., it was forced through large filter presses with 4x4 ft. wooden plates. When charged, each press was capable of holding five tons of residues. When a press was filled, compressed air was applied to blow out as much as possible of the strong solution which had been absorbed; then the residues, while still in the press, were washed with water.

Below the presses there were two rows of tanks, one to receive the strong liquor, the other the wash water. Some of the tanks were assigned to the stronger portion of the wash water which went back to the process, while the remainder were used as collecting and settling tanks for the weak wash water; this was sent to the scrap iron tanks for the precipitation of the contained copper. The strong liquor and the strong wash water tanks were connected with a pressure tank, placed on a lower level, by means of which the liquid could be forced up to the stir tank level. On opening the filter presses the washed residue cakes dropped into wooden push cars; they were then wheeled to an opening in the press floor, through which they were dumped directly into railroad cars. The residues which were rich in silver and lead were delivered to the lead smelting department.

The next operation was to chloridize the sulphate solution. For this purpose the strong solution was elevated, by means of a pressure tank, into a stir tank, used only for this purpose. Then the solution was tested for copper, its volume measured, the total copper in the charge calculated, and as much common salt added as was required to convert the copper

present into cuprous chloride (58 parts of sodium chloride to 63.4 parts of copper). The solution was agitated, heated, and then discharged into storage tanks; from these tanks the chloridized liquor was elevated and charged into so-called reducing towers for treatment with sulphur dioxide gas.

Precipitating the Copper with Sulphur Dioxide Gas.

The towers in which this part of the process was carried out were made of steel and lined with lead; the bottoms were cone-shaped. There were four towers, but their dimensions were not in conformity with the requirements. They were too short, too large in diameter (about 12 ft.), and the steel was not stiff enough for the great weight and the oscillation caused by the injected gas. I replaced them later by more suitable towers. The tops were tightly closed, provided with manholes, inlet pipes for the solution and an outlet pipe for the gas. The cone-shaped bottoms were provided with gas inlet pipes, a steam pipe and a discharge pipe. The outlet for the gas was connected with a main pipe which discharged into a wooden stack. The gas from each tower passed directly into the stack. This arrangement caused a considerable loss of gas. By tests it was found that a gas, which on entering a tower contained 7 per cent. sulphur dioxide, contained 4 per cent. when discharged, so that there was a loss of 57.1 per cent. I experimented and connected two towers so that the gas, after passing through the first, was made to pass through the next tower. By this alteration the loss of gas was reduced to 25.4 per cent., equal to an increased utilization of 31.7 per cent. The light construction of the towers, however, made the increased pressure in the first tower unsafe and I had to disconnect them. This experiment demonstrated that the precipitation of the copper in the tower under pressure was considerably quicker than in those without pressure.

The gas was furnished by three revolving cylinder furnaces, of which two were kept in operation and one in reserve. These furnaces were lined and provided with ribs for continually raising the ore, dropping it in a shower. The front end of the furnace was closed but the cover was provided with air registers, and two discharge openings for the roasted ore; the latter opened and closed automatically at each revolution of the furnace. The back end of each cylinder projected a few inches into a small dust chamber, which again was connected with a system of dust chambers. Through the roof of the small chamber, in a slanting position, entered the feed pipe of the furnace. The feeding was done by a very short screw conveyer connected with a hopper; the speed of the conveyer could be regulated. The material consisted of iron pyrites concentrates, rich in

gold, from Colorado. These concentrates were dried on steam slabs on a floor above the furnaces.

The last dust chamber was connected with a heavy lead pipe about 24 in. in diameter, and was strengthened with iron rings, to which the pipe was fastened. The entire length of this pipe, about 150 ft., was cooled by a spray of water, so that the gas was cool before it entered the pumps. There were two large double acting steam gas-pumps, of which, however, one was sufficient to do the work; the other was kept in reserve. The cylinder measured 27½ in. in diameter and the piston had a stroke of 28 in., so that each stroke furnished about 19 cu.ft. of gas. We had to regulate the speed in order to obtain a good roast of the concentrates and at the same time produce as strong a gas as possible. In order to fulfill both conditions, we found that the resulting gas could not be made to contain more than 5 per cent. sulphur dioxide. Frequent gas tests had to be made in order to maintain this percentage. Sometimes there was an increase in strength up to 7 per cent., in which case the roasting was not satisfactory; but more frequently it dropped below 5 per cent., which caused a slower precipitation.

The pumps forced the gas through a lead-lined receiver, in which a great deal of sulphuric acid condensed and had to be drawn off daily. The gas entered the tower under a heavy perforated lead cone, which divided it into small bubbles. Cuprous chloride was precipitated in small white crystals, while sulphuric acid was set free. The reaction was most energetic in the beginning, while the solution was neutral or contained only a small percentage of acid, became more sluggish in proportion as the percentage of acid increased, and stopped entirely when the copper contents of the liquor was reduced to about 2 or 2½ per cent. This remaining copper could not be reduced no matter how long the charge was kept under treatment with gas. The acid continued to increase slowly but the copper did not diminish.

The reactions in the Hunt & Douglas process are given as follows:

$$2CuSO_4 + 2NaCl = CuSO_4 + CuCl_2 + Na_2SO_4$$
$$CuSO_4 + CuCl_2 + SO_2 + 2H_2O = 2CuCl + 2H_2SO_4$$

According to these equations all the copper should be precipitated as cuprous chloride, but in actual practice, as before mentioned, this is not the case. A somewhat better result was obtained by the use of a moderate excess of salt. It may be possible that the following reaction also takes place: $2CuCl_2 + SO_2 + 2H_2O = 2CuCl + 2HCl + H_2SO_4$, and that the formation of hydrochloric acid accounts for all the copper not being precipitated. The more hydrochloric acid there is formed, the more copper will remain in solution. The hydrochloric acid dissolves cuprous chloride, forming cupric chloride, which again is decomposed by sulphur dioxide, forming

cuprous chloride, hydrochloric acid and sulphuric acid. This may also account for the fact that in an extended run the sulphuric acid continues to increase while no more copper is precipitated.

Sodium chloride as a chloridizer for the sulphate solution is not the most suitable chemical for this process, as a large quantity of sodium sulphate is formed which goes into solution. As it is necessary to use the solution over and over again, on account of the sulphuric acid which is formed therein, more and more sodium sulphate is formed. In a short time the solution becomes so saturated with this salt that it crystallizes out whenever conditions are favorable. This happens at different stages of the process, causing much annoyance and lessening the merit of the process. It was especially aggravating in the operation of the filter presses. When the matte residues, together with the strong liquor, were forced into the press, the filtration in the beginning was free and satisfactory, but soon grew less so until finally it stopped entirely, although the press was not one-quarter filled with residues. On opening the press it was found that the chambers were partly filled with a sloppy mass containing many fine crystals, while the filter cloth was densely covered with them. The only way of clearing the press was to force water through it. This not only caused much delay, especially as this application of water often had to be repeated, but caused the making of a large quantity of wash water, from which the copper had to be precipitated with scrap iron. Sometimes it happened that the press could be filled without any trouble, in fact the chamber filling was quite firm; but as soon as water was used to wash the residues, the filling shrunk in volume, and the frames which previously were quite full, after washing were only a little over half filled. This great annoyance induced me to use, instead of salt, the calcium chloride formed in converting the cuprous chloride into cuprous oxide by boiling with milk of lime. The resulting calcium chloride, however, was not a strong enough solution to be used directly, containing but 9 per cent. chlorine; also it would have diluted our copper solution too much, so I was obliged to concentrate it. In using calcium chloride, cupric chloride and calcium sulphate were formed, the latter being precipitated. Though this chloridizer made necessary an additional filter press operation, to separate the calcium sulphate from the solution, it was by far preferable to salt, as it left a clean solution, free from undesirable salts. This would have enabled me to do away with the great filter press trouble. However, the actual results differed from my expectations. I found that from a sulphate solution chloridized with calcium chloride, only about half of the copper in solution could be precipitated as cuprous chloride with sulphur dioxide. When the change from salt to calcium chloride was made, I prepared for the work a new acid solution, free from sodium sulphate. Several attempts

were made with the same unsatisfactory results. I finally added salt to the new solution and the results were at once better. From that time on chloridizing was done so that three-fourths of the required chlorine was derived from calcium chloride and one-fourth from sodium chloride. After adopting this proportion we had but little trouble with the presses; the filtration was free.

I stated previously that the calcium chloride solution resulting from the conversion of the cuprous chloride was not strong enough, it being necessary to concentrate it before using. To reduce this additional cost I attempted to make the original calcium chloride solution stronger. Instead of using water to make milk of lime, I used the weak calcium chloride solution in order to double the strength in chlorine, but this scheme was not practicable. I found that caustic lime, when treated with a calcium chloride solution, does not form milk of lime, but forms a sandy, semi-transparent mass. However, I attained the desired result by preparing milk of lime with water, allowing the lime to settle, decanting the clear water and replacing the same by weak calcium chloride solution. The condition of the slacked lime was not changed and by repeating this procedure I succeeded in increasing the proportion of chlorine from 9 to 24 per cent.

When the precipitation in the tower was completed, the cuprous chloride, together with the acid solution, was discharged into a system of seven cone-shaped lead-lined iron tanks. These tanks were so arranged that the liquor flowed from one to the other, and from the last into special storage tanks, to be in readiness to dissolve a fresh lot of matte. These cone-shaped tanks served a double purpose; to give the cuprous chloride an opportunity to settle, and to cool the solution. This liquor when hot holds in solution a large amount of cuprous chloride which precipitates out as the temperature falls. The temperature of the liquor in the first cone was 56.5 deg. C., and in the six following cones it was 48.0, 47.0, 45.0, 41.5, 39.5, and 37.0 deg. C., so that the temperature of the last cone was 19.5 deg. lower than that of the first. This cooling proved to be sufficient as no further precipitation took place in the storage tanks.

Below the level of the cones were three vacuum filters, into which the former could be discharged by opening the bottom valve. Into these filters the cuprous chloride was allowed to drain, there receiving a thorough washing. The washed precipitate was then converted into cuprous oxide.

Conversion of the Cuprous Chloride into Cuprous Oxide.

The conversion was done with milk of lime. The lime was slacked in a flat box and collected in settling tanks. From these tanks the milk of lime of proper consistency was charged, by means of a steam syphon,

into a stir-tank and heated with a jet of steam. The washed cuprous chloride was gradually charged, its color changing from white to red. Cuprous oxide and calcium chloride were formed. As calcium chloride dissolves cuprous chloride, the calcium chloride which is formed in proportion as the conversion progresses will dissolve some of the freshly charged cuprous chloride. On this property of the calcium chloride is based the test by which the conversion is conducted. After a certain amount of cuprous chloride has been added to the milk of lime, charging is interrupted, the stirrer, however, being kept in motion. About 10 minutes later, a sample is taken in a wide-necked flask suspended by a copper wire. Part of this sample is filtered and nitric acid and then ammonia added to the filtrate. If the blue color appears, some cuprous chloride is still dissolved in the calcium chloride solution. The agitation is then continued for 15 or 20 minutes, when another sample is taken. If the blue color appears again, more milk of lime is added gradually and at intervals. After each interval a test is made until the blue color ceases to appear. The last part of the operation has to be conducted carefully to avoid an excess of lime.

The pulp, consisting of cuprous oxide and calcium chloride, was forced by means of a double pump into a Johnson iron filter press, where it was well washed. The cakes were dumped on a lower floor and dried on steam slabs. When dry, the cuprous oxide was carted to the copper smelting department and dumped into bins, conveniently arranged on the charge floor of a cupola furnace. In this furnace it was reduced to metallic copper.

The cuprous oxide always contained from 4 to 5, and sometimes as much as 11 oz. silver per ton. Some of the silver undoubtedly came in with fine particles of matte residues, which were still suspended in the solution when it was charged into the towers for treatment with sulphur dioxide, although the resulting cuprous chloride was clear white and did not show any coloration. However, when I inserted more settling tanks for the solution, the cuprous oxide contained considerably less and more uniform amounts of silver. A sample of cuprous oxide containing 6.25 oz. silver per ton, when leached with a solution of sodium hyposulphite, still contained 5.5 oz. silver, so that only 0.75 oz. per ton could be extracted by that solution. This test was made with cuprous oxide produced before the additional settling tanks were in use; afterward the cuprous oxide did not contain over 2 or 3 oz. silver per ton. The resulting calcium chloride solution gave with sodium sulphide a dark precipitate which consisted mostly of lead sulphide, with only a trace of copper and no silver.

In smelting the cuprous oxide in the cupola, very strong and offensive fumes were formed. These fumes were so obnoxious that they became

a regular nuisance and often interfered with the work in other departments. These fumes were white but when very strong assumed a reddish tinge. They consisted chiefly of volatilized cuprous and cupric chloride, some hydrochloric acid and flue dust of cuprous oxide. An investigation showed that the cuprous oxide still contained 1 to 2 per cent. chlorine, notwithstanding the fact that it was subjected to a very thorough washing in the press. This chlorine could not be removed or reduced even by an extended washing. This was not due to the presence of cuprous chloride, caused by an insufficient quantity of lime being used in the conversion, for, even if for the sake of information an excess of lime was used and an unusually long time given for the conversion, the above stated percentage of chlorine was always found in the cuprous oxide.

If caustic lime is brought in contact with a weak solution of calcium chloride containing, say, 9 per cent. chlorine, it heats very rapidly. The lime slacks for a short time and falls to pieces, but does not form milk of lime. As before stated, it disintegrates and assumes the appearance of gravel. The smaller pebbles and the edges of the larger ones become transparent. A thick milk of lime mixed with a solution of calcium chloride becomes crystalline. Pieces of caustic lime brought into a strong solution of calcium chloride do not slack. The transparent substance formed, which is probably an oxychloride, dissolves in boiling water, but not very readily, and not at all or very slowly in water of ordinary temperature. This was most likely the cause of the trouble we encountered, the transparent substance being the chlorine carrier in the cuprous oxide. During the process of conversion, the calcium chloride in solution gradually increased in strength and, acting on the lime, formed this substance which remained in the cuprous oxide and which we could not remove by washing with water of ordinary temperature.

As washing with hot water, for different reasons, was impracticable, my next attempt was to destroy the very obnoxious character of the furnace gases and at the same time to prevent the heavy loss of copper which took place in smelting. For investigation, I placed on grates in the flue large lumps of caustic lime. After 24 hours some of the pieces were removed and examined. It was found that a crust had formed about $\frac{1}{16}$ in. thick, which could be peeled off. It consisted of calcium chloride, some carbonate of lime and some flue dust containing cuprous oxide, which gave it a reddish tint. In water it disintegrated into a powder, calcium chloride going into solution. The following table shows the analyses of several samples of this crust.

After leaving the lime in the flue for three weeks, the crust had increased only to the thickness of $\frac{1}{4}$ in. Much of the lime in the cooler part of the flue had been changed into carbonate by the carbon dioxide of

the furnace gases; thus energetic action was prevented. This method, even if it had been successful, would have been impracticable.

ANALYSIS OF CRUST FORMED ON LUMPS OF CAUSTIC LIME.

	Cl Soluble in Water.	Cl Insoluble in Water.	Cu.
	Per Cent.	Per Cent.	Per Cent.
1. From cooler portion of the flue; crust much decomposed	15.1	1.2	22.0
2. From cooler part of the flue; crust less decomposed	14.6	0.4	10.2
3. From hotter part of the flue	9.4	0.4	6.6
4. Portion next below the outer crust of pieces taken from the hotter part of the flue	1.5	0.2	0.7
5. Two in. beneath surface of a piece taken from the hotter part of the flue	0.2	0.0	0.0

It was next attempted to destroy the obnoxious character of the furnace gases by passing them through a shower of milk of lime. Not far from the stack there was an abandoned wooden tower, which was provided at different levels with strong wooden grates. At the foot of this tower tanks were arranged for making and receiving milk of lime. Two of these receiving tanks were connected with a force pump. The flue was connected with the wooden tower. Coarse limerock was placed on the different grates to detain the milk of lime in its downward course as long as possible. At the top several perforated pipes, through which the milk of lime was forced by means of the pump, were so arranged as to furnish an even spray. The bottom of the tower was made tight and the outlet made to convey the stream into one or the other of the two tanks, so that the milk of lime could be passed through the tower as often as desired.

The effect of the milk of lime was very gratifying. The strong and offensive odor of the gases disappeared entirely. The color of the milk of lime turned gradually darker and became finally olive green and very rich in copper. No copper escaped with the gases. When the olive green pulp was filtered the filtrate contained 1½ to 2 per cent. chlorine. The evaporation was great and water had to be added frequently to maintain the same volume of pulp. This method proved itself successful and was permanently installed.

I had made a good many analyses and notes during these experiments, but to my regret these records, together with many other notes and drawings, were lost during the great Kansas City flood of 1903.

Treatment of the Wash Water.

In the course of the process a great deal of wash water was made, principally from washing the copper matte residues and the cuprous chloride. The latter, which contained from 1 to 2 per cent. copper, was collected in a number of large tanks, from which it was drawn to be subjected to special treatment for the recovery of the copper. Wash water

containing 2 per cent. copper, and as much as practicable of the weaker portion, went back to the process and was used instead of water.

When I was placed in charge of the works the arrangement for recovering the copper from the wash water consisted of a large number of tanks, 6x6 ft. x 3 ft. deep, filled with scrap iron. The tanks were placed in two rows and so connected that the wash water flowed from one tank into the other. The solution was heated by means of steam jets. On a track between the two rows of tanks, a car was run to bring in scrap iron, and during clean-up to wheel out the cement copper; this latter was dumped into a storehouse. No provision was made to wash the cement copper. The outflowing solution of the last tank was frequently tested and when free of copper was allowed to run to waste. If a trace of copper was found the influx was reduced. No attempt was made to save the iron salts.

In the storehouse I found a pile of about 200 tons of cement copper. This had accumulated because it was in a condition which made it impossible to smelt it without an excessive loss of copper. This cement copper contained 56 to 59 per cent. copper, of which a large part was present as cuprous chloride and oxychloride; the latter predominated and was formed from the cuprous chloride by exposure to the air. The cuprous chloride in the cement copper was formed by the following reaction: The copper contained in the wash water was present partly in the form of chloride and partly as sulphate. Iron precipitated the copper from both solutions in a fine state (cement copper), forming ferrous chloride and sulphate respectively. Cement copper in contact with a warm solution of cupric chloride precipitates cuprous chloride. When this cuprous chloride comes in immediate contact with iron, metallic copper and ferrous chloride will be formed, but that part of it which settles on spaces between the iron, or is imbedded in the cement copper, remains unchanged. If then discharged and exposed to the air it changes chiefly to oxychloride. The first few tanks through which the wash water passed contained not only the most cement copper, but also the most cuprous chloride.

To produce a clean cement copper, free of chlorine, I constructed in sections a trough about 200 ft. long, 12 in. wide and 14 in. deep. All sections were placed horizontally, but each succeeding one was placed 3 in. lower. The outlet of each section was 2 in. lower than the inlet. In some of the sections a compartment was made by inserting across the width of the trough two boards 6 in. wide. These were placed about 12 in. apart to allow for the insertion of a steam jet. This part of the trough was tightly covered for 2 ft. on either side of the jet to prevent the solution from being splashed out by the steam. The 200 ft. of trough was arranged in U shape to avoid too long a building, and to make the handling of the material easier. After passing through the last section, the solution

flowed through a few scrap iron tanks to precipitate any copper which might be present as sulphate. Each section of the long trough was charged about 4 in. deep with cement copper, evenly spread. The wash water from the main department flowed into a circular tank 8 ft. in diameter and 5 ft. deep, thence through an overflow into the first section of the long trough. The purpose of the circular tank was to heat the wash water by means of steam jets before it entered the long trough. It gave also an additional, though not very effective, opportunity for the settling of particles of matte residues which might not have settled in the proper wash water storage and settling tanks. As the wash water resulting from the different operations of the process contained cupric sulphate in addition to the cupric chloride, salt water was added in the storage tanks to convert the cupric sulphate into chloride.

By passing the wash water through and over the cement copper, the copper was precipitated as cuprous chloride. Once a day the cement copper in the troughs was gently worked in order to bring fresh particles to the surface. At one side and below the level of the troughs a revolving barrel, 10 ft. long and 6 ft. in diameter, was erected; when a large part of the cement copper in the troughs had changed to cuprous chloride, it was removed from the troughs and charged into the barrel, to which small scrap iron, water and salt were added. By means of a steam pipe, the pulp was slightly heated to start the reaction; then the steam turned off. This pipe entered the barrel through one of its axles and was bent downward to reach into the pulp; it was kept in position by a stuffing box. Very soon an energetic reaction took place, the heat developed causing the pulp to boil violently. The steam found an outlet through another pipe inserted through the opposite axle. The salt was added to dissolve some of the cuprous chloride; this caused the reaction to start more quickly. The cuprous chloride changed into cement copper and the iron into ferrous chloride. The latter had the same effect as salt and dissolved cuprous chloride, thus assisting the process.

After the steam ceased to escape, the barrel was stopped and a sample taken. The sample was filtered and the filtrate tested. When the blue color could no longer be obtained with nitric acid and ammonia, the pulp was ready to be discharged. The conversion was completed in from 6 to 10 hours. Below the barrel was a square tank with a filter bottom, fitted on two opposite sides with rails which extended beyond the tank. On this track was an iron 8-mesh screen, fastened to a wooden frame and provided with four wheels. The screen covered the whole top of the tank. In discharging the barrel, the copper cover of the manhole was removed and the barrel gradually turned by means of a crow-bar. When the charge was out, the inside of the barrel was rinsed with water. The screen retained

pieces of iron, while the cement copper and solution passed through to the filter of the tank. By means of a stream of water, the iron on the screen was separated from adhering cement copper and returned to the barrel to serve as part of the next charge. The screen on wheels was pushed away from the tank, the outlet under the filter opened, and the iron solution allowed to drain off. Then warm dilute sulphuric acid was permitted to flow in. As soon as the acid appeared at the outlet the latter was closed. The acid was applied to remove basic salts, to prevent their formation as far as possible, and to dissolve any small pieces of iron which had passed through the screen. After several hours the acid was removed to a special tank to be used again. The copper was then thoroughly washed in the tank to free it from acid. The resulting cement copper was of a very clear color and unusually pure, containing 99 per cent. copper and but a trace of arsenic. It was melted in the refining furnace.

This method gave such good results that three more barrels were erected. The method eliminated the conversion of the cuprous chloride into cuprous oxide by milk of lime, the smelting of the product in the cupola furnace, the treatment of the furnace gases, the very unclean manipulation of the scrap iron tanks, and the additional treatment of the cement copper to free it from cuprous chloride.

Silver Contained in the Copper Produced by the Hunt & Douglas Process at Argentine.

The copper matte which was treated by this process contained from 200 to 300 oz. silver per ton. The final copper product contained from 6 to 8 oz. silver per ton. An investigation showed that the main portion of this silver was brought into the copper by insufficient settling of very fine and rich particles of matte residues, while the balance came from silver being dissolved in the solution.

Examining the scrap iron tanks, through which all the wash water passed, I found that the cement copper in the first tank contained 28 oz. silver per ton. There were only two settling tanks for the wash water and therefore not enough time could be given for settling. I inserted four more settling tanks, so that the time for settling was much longer. After operating for two weeks, I found that the cement copper contained only 6.6 oz. silver per ton. Before installing these four additional tanks, a sample of the wash water just before it entered the scrap iron tank was taken and allowed to stand for several hours. Notwithstanding the fact that the liquid appeared perfectly clear after a few hours, a red-brown coating of extremely fine matte residues formed at the bottom. After proper opportunity was given for the fine particles of matte residues to settle, by increasing the number of tanks for the strong solution as well

as for the wash water, the silver in the copper produced was reduced to about 3 oz. per ton. The surest way to free the different solutions from matte particles would have been to pass them through common filters, but the volume of solution in circulation was rather large and the filtering arrangement would have required great space and additional labor.

The greater portion of the silver still found in the copper, however, came from silver dissolved in the solution. I convinced myself that the solution contained silver by the following experiment: Two tanks (Fig. 1), 4x8 ft. x3 ft. 4 in. deep, were placed together and connected with

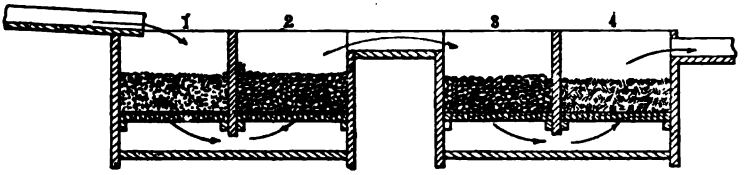


FIG. 1.

an overflow. A partition which did not reach down to the bottom was inserted in each tank, thus leaving a communicating space. By these partitions four compartments, 4x4 ft., were formed, each of which was provided with a false bottom perforated with ½-in. holes. In compartment 1 was arranged a sand filter; compartments 2 and 3 were charged with granulated copper, and 4 with cement copper. The intention was to desilverize the solution. By this arrangement the solution entering compartment 1 was filtered and freed from all solid particles. It passed the granulated copper in 2 upward, in 3 downward, and in 4 upward, keeping the copper always covered with solution to prevent oxidation by the air in case the influx should be interrupted for a while. Samples were taken and it was found that the granulated copper contained 3.1 oz. silver per ton, and the cement copper only a trace of silver. It was arranged that the strong liquor (the solution resulting from dissolving the matte in the stir tank before chloridizing) from the settling tanks could be passed through this system after due time had been given for solid particles to settle.

After the system had been in operation two days, samples were taken from compartments 2, 3 and 4. The sample from 2 contained 10.4 oz. silver (gain in 2 days, 7.3 oz. per ton); that from 3 contained 9 oz. silver (gain in 2 days, 5.9 oz. per ton); 4 contained 3.6 oz. silver (gain in 2 days, 3.6 oz. per ton). After five days' operation samples were taken again. It was found that the granules were cemented together with cuprous chloride, the cement copper so much so that a pick had to be used to loosen it to take the sample. Part of each sample was boiled with

milk of lime to remove the cuprous chloride. The cuprous oxide formed was separated from the metallic copper and all three products were assayed for silver, viz.: the sample of granulated copper just as it was taken from the tank with cuprous chloride on; the granules after boiling with lime; and finally the cuprous oxide. The same was done with the cement copper. The results were as follows: Sample 1, from compartment 2; granules with cuprous chloride, 16.8 oz. silver per ton (gained 13.7 oz. per ton); same after boiling with lime, 7.2 oz. silver per ton; the resulting cuprous oxide, 24.6 oz. silver per ton. Sample 2, from compartment 3; granules with cuprous chloride on, 18.6 oz. silver per ton (gained 15.5 oz. per ton); granules after boiling with lime, 6.1 oz. silver per ton; the resulting cuprous oxide, 25.6 oz. silver per ton. Sample 3, from compartment 3; cement copper with cuprous chloride, 5.2 oz. silver per ton (gained 5.2 oz. per ton); after treating with lime, 2.4 oz. silver per ton; the resulting cuprous oxide, 2 oz. silver per ton.

A few days afterward this experiment was stopped. The cuprous chloride continued to form and cemented the copper so closely that it finally prevented the percolation of the solution almost entirely. The object of this experiment was to desilverize the solution by an inexpensive method and to produce a copper free from silver. A mistake, however, was made in selecting this method by not taking into consideration the fact that the acid stock solution always contained from 1 to 2 per cent. chlorine; for it was not possible to convert all the copper into cuprous chloride with sulphur dioxide. Therefore, in again using the acid solution for dissolving the cupric oxide from the matte, the resulting sulphate solution had to contain chlorine, and the formation of cuprous chloride was to be expected.

It was difficult to understand why the solution resulting from the treatment of the roasted matte with acid stock solution contained silver. All the silver which had dissolved as sulphate should have been precipitated as chloride by the chlorine which the stock solution contained, and should have remained in the residues. On the other hand, a chloride solution containing only 1 to 2 per cent. chlorine is too weak to dissolve silver chloride. This is at least my experience in leaching silver ores.

Preparations for Smelting Cement Copper Which Contains Cuprous Chloride and Oxychloride in Large Quantities.

I mentioned elsewhere that when I took charge of the works, I found in the store house about 200 tons of accumulated cement copper which, on account of the large percentage of cuprous chloride and oxychloride it contained, was not suitable for smelting. The loss of copper by volatilization would have been enormous and the furnace gases too obnoxious

to be allowed to escape into the air. The only way to get this material in proper condition for smelting was to free it from chlorine. By boiling it with milk of lime, the cuprous chloride was changed into cuprous oxide and calcium chloride; the latter went into solution and could have been easily removed, but the oxychloride was not decomposed, so that boiling with milk of lime was of no practical value.

When the cement copper containing subchloride and oxychloride was heated to a dark red or even below that, strong, heavy vapors were expelled, causing a great loss of copper; but if before heating pulverized or dry slacked lime was mixed in, no fumes were evolved and no copper was lost, even if the charge were heated to bright red. The chlorine which had combined with the lime, forming calcium chloride, could be easily removed by leaching with water. With an addition of 10 to 15 per cent. lime, only very light fumes came off at a red heat. With 20 per cent. lime no fumes were evolved. When pulverized crude limerock was used instead of caustic lime and the mixture was heated, heavy fumes were expelled, until later in the operations, when the carbonate of lime had lost its carbon dioxide; but by that time the charge had lost a large portion of its copper.

The sample which was roasted with caustic lime was leached with warm water. The filtrate was quite milky, depositing a white precipitate. The clear blue solution, when diluted with water, again became milky, depositing an additional precipitate. This white precipitate consisted principally of lead, iron and copper and was rich in silver. The residues contained only 0.5 per cent. chlorine, as against 13 per cent. before roasting and leaching, which means that over 96 per cent. of the chlorine was removed.

This experiment was repeated on a somewhat larger scale. In a small experimental roasting furnace 500 lb. of the green cement copper was roasted, with an addition of 20 per cent. of caustic lime. The two materials were thoroughly mixed before charging. The roasting was done at a dark red heat, but before this point was reached the charge commenced to turn black from the formation of cupric oxide. No fumes could be noticed, not even at the end of the operation when the heat was raised for a short time to bright red.

The roasted cement copper was divided into two parts; one was leached with hot, the other with cold, water, in small tanks made for the purpose. The filtrate in both cases was milky, but that resulting from leaching with hot water was more so than the other. A solution of calcium sulphide was added to the filtrates in order to precipitate all the metals dissolved therein. The dark precipitate was dried and assayed for silver and gold. That from leaching with hot water contained 632.9

oz. silver and 0.29 oz. gold per ton, and that from leaching with cold water contained 409.2 oz. silver and 0.10 oz. gold per ton.

The roasted material contained 15.6 oz. silver per ton, while the residues after leaching with hot water contained 3.94 oz., and those leached with cold water contained 6.1 oz. silver per ton. The results with respect to the removal of the chlorine were almost the same as those obtained on a small scale in the laboratory. In roasting, the silver was converted into silver chloride, dissolved, and extracted by the more concentrated part of the calcium chloride solution.

On account of the rather satisfactory results obtained by this method, I concluded to use it in working the 200 tons of green cement copper. The material was roasted in a large reverberatory furnace in the lead smelting department, with an addition of 20 per cent. lime in lots of one ton each. A small but conveniently arranged temporary leaching plant was erected in a vacant building of the works. The roasted and leached material was dried and pulverized, along with the roasted matte in the ball mill, and dissolved in the stir-tanks. The precipitate obtained by adding calcium sulphide solution to the filtrate, containing 400 to 600 oz. silver per ton, together with the matte residues, was sent to the lead smelting department. In this way the 200 tons of green cement copper were worked to advantage. Later after some alterations, I used this temporary leaching plant for the manufacture of zinc sulphate from the lead-zinc oxide of the refinery.

II. *A Modification of the Hunt & Douglas Process.*

To simplify the operations and to avoid the saturation of the solution with sodium sulphate and its attendant disadvantages, I worked out and successfully introduced the following modus operandi:

(1) The process was started with a stock of dilute sulphuric acid. By treating the roasted matte in the usual way in the stir tank, a clean sulphate solution was obtained which filtered well in the press.

(2) The sulphate solution was chloridized with hydrochloric acid, of which in starting a stock on hand is required. By chloridizing the sulphate solution with hydrochloric acid, cupric chloride is formed and sulphuric acid is set free. No foreign salts are introduced and the solution remains clean, while by chloridizing with sodium chloride the solution becomes quickly saturated with sodium sulphate.

(3) The cupric chloride solution containing the liberated sulphuric acid was then treated in a stir tank with cement copper. The cupric chloride, by the action of metallic copper, is reduced to cuprous chloride, while the sulphuric acid remains unchanged. A steam jet is used to hasten the reaction. An excess of cement copper serves the same purpose. When

the filtrate of a sample shows no reaction for copper, the operation is complete and the pulp is drawn into a pressure tank and forced through a filter press. The filtrate, which is now a clean sulphuric acid solution, is elevated to storage tanks, whence it is used as required to dissolve the cupric oxide of a new lot of roasted matte. To produce as little wash water as possible, the solution absorbed by the matte residues and by the cuprous chloride is forced out by compressed air; this works very well, as the filtering capacity of both materials is not lessened by the formation of crystals. For the same reason the subsequent washing is quickly done, requiring comparatively but very little water to accomplish it.

(4) The washed cuprous chloride was treated in revolving barrels, in the same manner as described above, but care was taken that not more water was added than necessary, so that as strong a ferrous chloride solution was produced as practicable.

(5) The ferrous chloride solution was evaporated to dryness in the same iron pans which were formerly used for concentrating the calcium chloride solution.

(6) The solid ferrous chloride was charged into retorts, which were provided with inlets for steam and air. When heated, heavy fumes of hydrochloric acid were formed; these were passed through a cooling arrangement, in which a large portion was condensed. This condensed acid was strong and contained 35.6 per cent. chlorine. The acid fumes which were not condensed were made to pass through two towers made of stoneware pipes and filled with coke. The gas escaping from the first tower entered at the bottom of the second. To avoid the accumulation of too much water in the stock solution, I used cupric sulphate solution instead of water as a spray for the coke, thus chloridizing the solution. The solution, as a rule, after passing through the towers, contained an excess of hydrochloric acid. This condition, however, was properly adjusted by adding sulphate solution before the treatment with cement copper.

(7) The resulting cement copper was very pure, containing from 90 to 94 per cent., and more, copper. This was smelted in the refining furnace. No obnoxious gases were evolved, as in the case of the cuprous oxide. There was, of course, a loss of sulphuric, as well as of hydrochloric acid, which was caused chiefly by the wash water; these had to be replaced from time to time to keep up the volume of the stock solution. However, this shortage was not great and the cost of replacing it was by far less than that of the eliminated operations.

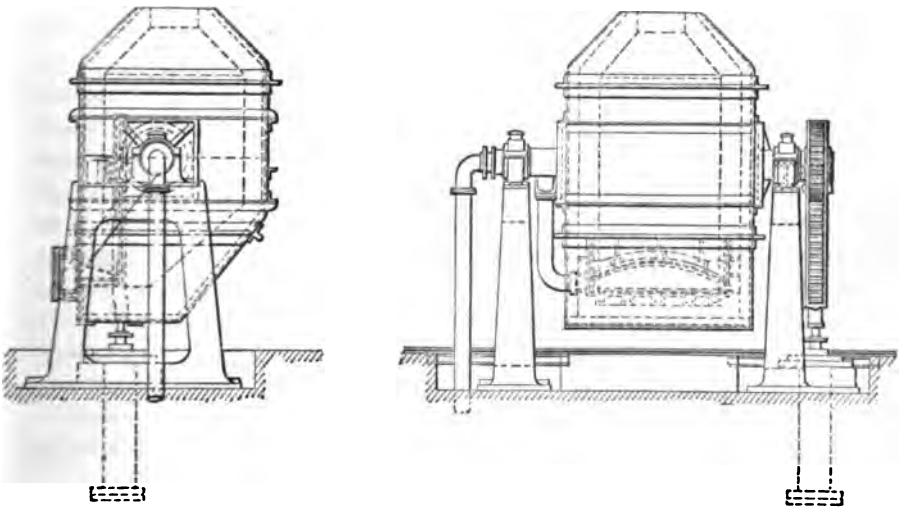
This modified process was used for some time, until I received instructions from the company for the necessary alterations of the works to prepare for the more profitable manufacture of blue vitriol. This was done by my method of producing this material direct from roasted copper matte.

Pyrite Smelting by the Knudsen Method as Practiced at Sulitjelma, Norway.

By E. Knudsen.

Much time and much money have been spent on the problem of pyrite smelting, and but slow progress has been made. This may partly have depended on the diversity of ore, necessitating different treatments; what is suitable for one ore may not be applicable for another. In the development of the Knudsen process at Sulitjelma, this was well observed, for experiments with very different ores were made during the course of the year and furnished a good guide.

Outline of Method.—The Knudsen process depends on the fact that when sulphide ores are roasted under the influence of a powerful air current the ore melts partially, and the pure metallic sulphide flows out and gathers on the hearth of the furnace. The construction of the furnace is shown in the accompanying engraving. The idea is to accumu-



KNUDSEN FURNACE, 8 TO 10 TONS.

This represents the furnace as first constructed by the author. The larger furnaces now used are of a modified form, which has been found more satisfactory.

late a small bath of melted ore as soon as possible by means of the air blast; by the production of a strong oxidation in this bath a very high temperature is produced which then acts on the remainder of the charge.

The temperature attained in this process is so high that sulphide ore, even rich in silica, becomes entirely fluid. The tremendous heat, particularly in the last period of the concentration, renders it necessary to employ very thick linings of refractory material. The heat remaining

in the furnace after the matte and the slag are removed is sufficient to kindle the new charge. The amount of coke or coal employed is very small, ranging from 0.5 to 1.5 per cent. of the weight of the ore, according to the nature of the latter. This quantity of coke is not particularly necessary for carrying out the process, but it serves to hinder the stiffening of the first, small, metal bath by the blast. The introduction of the air plays an important part in the process.

To charge the furnace, the estimated quantity of coke or coal is first introduced; this is then subjected to a strong air blast, whereby the fuel becomes red hot. The upper half of the furnace is connected with the hopper, which is then opened, and the charge falls into the furnace. The air blast is then maintained at a pressure of about 5 lb., the tuyere being well cleaned so that it does not form a nose. When the bath of melted iron sulphide reaches the tuyere, the air pressure is increased to 10, 15 and even 22 lb. in some cases. In from 1½ to 2 hours, the whole charge is ordinarily melted and concentration begins; in from 3½ to 4 hours a matte is ready, containing from 40 to 50 per cent. of copper. The furnace is then tipped horizontally and the contents poured out into the forehearth.

Experience at Sulitjelma.—It is now six years since the first experiments on a large scale were made at Sulitjelma and progress is slowly but surely being attained. At first a furnace of about seven tons capacity was erected; then one to treat 10 to 12 tons, and finally one that had a capacity of from 15 to 20 tons. When I say 10 to 12 and 15 to 20 tons, the reason for this is that the ores are not of the same bulk, on which account it would be more correct to state the furnace size in terms of its interior volume, e.g., 6 cu.m. and 10 cu. meters.

The durability of the furnace can be seen in the following statement.

DURABILITY OF FURNACE.

Run.	Magnesite brick used in repairing furnace.	Number of charges.	Tonnage smelted.
1	2,184	81	734.4
2	1,456	139	1,399.8
3	1,880	96	942.1
4	1,220	86	859.7
5	1,975	70	628.4
6	1,522	109	1,013.8
7	1,960	218	1,733.7
8	2,150	189	1,407.4
9	2,076	124	1,000.8

The small furnace, in the period from Feb. 18, 1907, to April 9, 1908, made nine runs which, while they treated different tonnages, still showed an increasing duration.

The larger furnace generally gave a longer run, apparently in proportion to its capacity, and it required less repair. The big furnace permitted 179 to 202, and even up to 459 charges per run. While the smaller furnace required for repairs approximately 1½ magnesite brick per ton of ore, the maintenance of the large furnace needed only one brick per ton. It is to be remarked that after removal the old brick were crushed and used for lining the hearth in the slag receptacle.

Nature of Ores Smelted.—The ores of Sulitjelma may be divided in two classes: Pure silicious ore and ore that is partially chloritized. The average analyses of each class are given in the accompanying table.

ANALYSES OF SULITJELMA ORES.

	Silicious.		Chloritised.	
Copper	6.47%	6.45%	5.15%	5.11%
Iron	32.74	32.90	25.64	27.10
Sulphur	33.87	39.90	22.64	25.30
Silica	16.75	16.20	31.38	26.30
Alumina	10.17	8.50	12.21	12.20

Our experience has shown that when a silicious ore is smelted, an easily fusible slag results, which, on being retained for about 20 min. in the horizontally placed furnace, gives a slag containing only from 0.5 to 0.6 per cent. copper. When, however, a large quantity of the chloritized ore is treated, the slag becomes thick and must be held for a longer time in order to separate completely the particles of matte blown into it by the blast. For this purpose a forehearth is provided into which the charge is poured and allowed to remain for about 90 min., after which the slag is run off and granulated while the matte is tapped into a Manhés converter.

Certain other ores had the disadvantage of decrepitating, which caused the charge to fall into a more or less powdery state, and then fuse together. If a considerable part of the charge should then fall into the molten bath at once, sharp explosions would occur, which, to the uninitiated, would seem dangerous; the only damage, however, was the freezing of the bath, if the falling mass of charge were too large. Whenever this accident occurred, we would tip the furnace to the horizontal and put a few kilograms of coke and 200 to 400 kg. of crude ore in range of the tuyeres; on replacing the furnace vertically and turning on the blast, the charge would again become normal. We have, in fact, smelted charges consisting of one-third lump ore, one-third fine ore and one-third dust, with success, but when so much fine material is introduced into the furnace the charge is bound to sinter somewhat.

We also experimented with slimes from the Elmore vacuum process, briquetted for smelting, with excellent results. The Elmore slimes were mixed with 4 per cent. of milk of lime, pressed with a primitive ramming apparatus, and placed for 12 hours in a drying chamber. The briquets were thereby rendered so firm that they withstood the hardest knocks, and did not fall apart when introduced into the Knudsen furnace. In 2½ hours of smelting, a matte was obtained with 38.5 per cent. copper. When much fine ore has to be treated, it is best to divide it, taking the smaller portion to make into briquets, drying but not burning them, before putting them into the furnace. This insures a quiet operation without unpleasant disturbances.

We have smelted ores high in silica and low in sulphur and iron, for example, with 20 to 21 per cent. of sulphur, 32 per cent. of iron and 40 per cent. of silica. These ores smelted satisfactorily and gave a clean slag, but they were hard to kindle on account of the small amount of sulphur in them; this fact required the addition of 2 per cent. of coal and coke, but as soon as the bath of iron sulphide came in front of the tuyeres the oxidation of the iron furnished the necessary heat.

Experimental Results.—The composition of the matte during these experiments ranged from 45 to 56 per cent. copper, and the slags of a number of charges contained, respectively, amounts of copper and silica

SLAG ANALYSES.

Copper.	Silica.	Copper.	Silica.	Copper.	Silica.	Copper.	Silica.
0.52%	34.46%	0.48%	34.47%	0.53%	34.97%	0.46%	32.00%
0.61	36.91	0.40	34.35	0.63	33.66	0.42	31.49
0.49	37.68	0.38	35.35	0.65	32.56	.....	.....

shown in the accompanying table. If the matte and slag be allowed to remain for a time on the hearth, and it has been indicated that a duration of 1 to 1½ hours is best, the matte particles can then be successfully separated. It is advantageous to place some pulverized, pure silicious ore on the hearth in order to render the slag cleaner. It also has a good effect on the matte, which is sometimes produced too profusely if it has formed a crust on the walls, and for this reason has had to be tapped more slowly. A matte with 65 to 70 per cent. of copper might be obtained, which, however, is not well adapted for the Manhés converter; by adding this raw ore, the copper contents can be brought down to 55 or 60 per cent., which can then be bessemerized very well.

Composition of the Slag.—As previously stated, the fluidity of the slag is the deciding factor. With a well acting furnace and with ores giving an easily fusible slag, no forehearth was found to be necessary.

But in metallurgical processes things so often take place unexpectedly, that I do not consider it advisable to work without the forehearth.

The slags at Sulitjelma are commonly monosilicates, with from 28 to 32 per cent. silica, 56 to 59 per cent. iron oxide, 10 to 12 per cent. of alumina, about 1.5 per cent. sulphur, and from 0.3 to 0.6 per cent. of copper.

Experiments were made by allowing the slag to flow into large iron pots, and fine results were obtained for a day at a time, until suddenly the slag would become thicker, and the separation of matte particles would be incomplete. Such surprises do not occur in a forehearth heated by generator gas. I have observed that slags with 32 to 36 per cent. silica carry the least amount of copper. We have had slags that, on being granulated, contained only 0.29 per cent., but ordinarily they varied from 0.4 to 0.6 per cent, in copper.

Composition of the Matte.—The matte ordinarily produced at Sulitjelma has a copper content of 50 to 55 per cent, and contains 20 to 21 per cent. of sulphur. After tapping into the converter, the first hour's treatment produces a copper 99.25 to 99.5 per cent. pure. The copper contains some silver.

The greatest difficulty of the process, when conducted in a small furnace, is to secure a matte which shall not vary too widely from the average. Wall accretions are more likely to occur, involving a longer blowing; and, thereby, a richer matte unsuited for the Manhés converter. With a large furnace, this last difficulty is less pronounced, since the charge sinks regularly and permits the formation of a matte with any desired copper content. With charges low in iron, it is important not to blow too long, making a matte high in copper, since there is danger of freezing the charge.

When the charge has been sufficiently blown, the furnace is emptied at once, and after a few minutes the matte runs into the converter. We have often made a finished bessemer copper within five hours after charging the furnace, and sometimes within 4½ hours.

Lining of the Furnace.—The durability of the lining depends altogether on its construction. The magnesite brick must be closely united, so that the joints are reduced to a minimum, and special mortar must be used. The fragments of the brick from the lower part of the furnace are used partly for repairing the upper part when necessary (which is not often), and some are broken up for lining the forehearth. Magnesite is especially necessary for the lower part of the furnace, up to 1.25 m. above the tuyere. It is the back of the furnace that suffers most. We have, for example, repaired the large furnace twice, above the tuyere, with 600 to 800 brick, and with these two small repairs have smelted

9000 tons of ore. The magnesite mortar is mixed with tar and applied very warm.

Air Blast.—The blast pressure is variable, ranging from ¼ to 1 atmosphere, and often to 1⅛ or 1½ atmospheres. A good deal depends on this as to how the charge starts. When care is exercised in the beginning, and not too much air is given during the first hour, a charge will generally settle down in a regular manner, and will then work well with a pressure of one atmosphere. When the charge hangs back and crusts are formed, it is necessary to use a higher pressure, whereby the melted bath is strongly agitated and the crusts go into solution.

The power necessary for blowing a Knudsen furnace, of about 12-cu.m. capacity, is as follows: In the first 1½ hours, not far from 40 h.p. is needed; in the next 1 to 1½ hours, 80 to 100 h.p. and in the last hour of the blast, 150 to 160 h.p., and in a few rare cases, 200 h.p. A furnace of 12-cu.m. capacity treats 20 to 25 tons of ore, and inasmuch as at least four charges in 24 hours can be reckoned, the power required is not large per ton of ore smelted, particularly as a pause of about one hour takes place between every two charges, for cleaning the furnace and burning the coke and coal, when practically no power is consumed. Having steam facilities, the feed water can profitably be warmed by gases given off from the furnace, as I have practically demonstrated.

Waste Gases.—The gases given off in the beginning of the process are not very hot, but they soon increase in temperature; at a distance of 10 m. from the mouth of the furnace, we have measured, with a pyrometer, temperatures of 600 to 780 deg. C., toward the end of the process. The radiation from the lining of the furnace, after it has been emptied and again set upright, is so great that we have measured 180 deg. in the outlet channel distant 10 m. from the mouth of the furnace.

The percentage of sulphurous acid in the gas has often been determined. From the nature of the process, the percentage is somewhat variable, but the gases from a charge show an average between 6 and 7 per cent., by volume, of SO_2+SO_3. The amount has been determined to be from 2½ to 11½ per cent. by volume, and was, in the last instance, 9 to 9½ per cent. dioxide, and about 2 to 2½ per cent. trioxide.

Fuel.—The coke and coal consumption depend on the cross section of the lower part of the furnace, and not on the size of the charge. Thus if a 20-ton furnace be charged with only 12 or 14 tons, the same amount of coke must be used as though it were filled with 20 tons. As previously stated, with ores difficult to kindle the coke consumption rises to 2 per cent., while ordinarily it is less than 1 per cent.

For a charge of from 20 to 25 tons, there are required, per ton: 0.1 ton of coke; 0.1 ton of coal; 15 pieces of magnesite brick, to repair the

furnace; and 20 pfennings (5c.) worth of tools, tar, etc. The heating of the forehearth also requires, per 24 hours, about 2¼ tons coal and

SMELTING RESULTS AT SULITJELMA

Date.	Charge No.	Charges in Tons.		Time.	Elmore-Slimes charged on Hearth.	% Copper.		Generators.	
		Ore.	Coke and Coal.			Slag from Hearth.	Matte from Hearth.	Tons.	
								Coal.	Wood.
				h. m.					
26/9	120	11.2	0.2	4.00	.....	0.36	66.4		
26/9	121	10.4	0.2	3.35	.....	0.46		2.5	0.300
26/9	122	11.1	0.2	3.40	.....	0.48			
26/9	123	12.4	0.2	3.45	.....	0.50	47.7		
27/9	124	12.3	0.2	4.00	.....				
27/9	125	10.5	0.2	4.45	.....	0.43	50.5	2.1	0.250
27/9	126	10.4	0.2	3.50	.....				
27/9	127	11.8	0.2	4.15	.....	0.47	40.7		
27/9	128	10.4	0.2	4.40	.....				
28/9	129	10.5	0.2	4.20	.....	0.59	34.4	2.4	0.275
28/9	130	11.0	0.2	4.20	.....				
28/9	131	9.9	0.2	4.50	.....	0.60	65.4		
28/9	132	9.8	0.2	3.40	.....				
29/9	133	10.8	0.2	5.15	.....	0.55	73.3		
29/9	134	13.6	0.2	4.25	0.2	0.62		2.5	0.300
29/9	135	11.9	0.2	4.35	0.5	0.41			
30/9	136	11.9	0.2	5.15	0.5	0.45	(a)60.0	2.4	0.325
1/10	137	13.3	0.2	5.00	0.6	0.49		2.1	0.275
1/10	138	12.6	0.2	3.10	0.4	0.55	63.9		
2/10	139	13.3	0.2	4.00	.....	0.52	54.0		
2/10	140	13.4	0.2	4.30	0.7			2.3	0.300
2/10	141	13.5	0.2	4.20	0.5	0.42	62.6		
2/10	142	11.2	0.2	3.35	0.5				
3/10	143	14.1	0.2	3.50	0.5	0.48	48.2		
3/10	144	11.1	0.2	4.35	0.5	0.60	68.0	2.3	0.300
3/10	145	13.6	0.2	4.25	0.5	0.58	59.2		
3/10	146	11.0	0.2	4.40	0.5	0.47	44.7		
4/10	147	11.0	0.2	4.30	0.5	0.44	60.8		
4/10	148	12.4	0.2	4.30	0.5	0.60		2.3	0.350
4/10	149	14.0	0.2	5.00	0.5	0.50	58.5		
4/10	150	13.5	0.2	3.35	0.5				
9/10	151	13.6	0.2	6.00	0.5	0.60	55.0		
9/10	152	14.3	0.2	4.20	0.5				
9/10	153	13.3	0.2	4.05	0.5	0.54	45.72	2.6	0.350
9/10	154	13.9	0.2	4.15	0.5				
9/10	155	11.2	0.2	3.40	0.5	0.51	56.7		
10/10	156	13.9	0.2	3.20	0.5				
10/10	157	12.0	0.5	4.05	0.5	0.41	49.9	2.6	0.375
10/10	158	13.9	0.2	4.20	0.5				
10/10	159	14.0	0.2	4.15	0.5	0.40	42.6		
11/10	160	14.9	0.2	4.20	0.5	0.42	44.0		
11/10	161	11.3	0.2	5.00	0.5	0.40	45.0	2.2	0.325
11/10	162	14.2	0.2	5.35	0.5	0.47	51.5		
11/10	163	13.9	0.2	3.35	0.5				
12/10	164	15.4	0.2	5.10	0.5	0.52	49.2	2.0	0.300
12/10	165	13.7	0.2	5.25	0.5				
13/10	166	13.5	0.2	4.35	0.5	0.53	51.3		
13/10	167	13.6	0.2	3.45	...			2.4	0.325
13/10	168	13.6	0.2	3.45	...	0.56	56.2		
13/10	169	11.8	0.5	4.00	...	0.50	53.6		
14/10	170	11.5	0.2	3.05	0.5	0.47	62.6		
14/10	171	8.3	0.2	3.50	0.5				
14/10	172	12.2	0.2	2.35	...	0.40	49.5	2.2	0.300
14/10	173	11.4	0.2	2.55	0.5				
14/10	174	12.4	0.2	4.10	0.5	0.55	49.4		
15/10	175	13.6	0.2	3.55	0.5	0.46	44.7		
15/10	176	13.6	0.2	3.50	0.5			2.0	0.300
15/10	177	13.6	0.2	4.00	0.5	0.50	50.9		
15/10	178	9.0	0.2	3.45	0.5	0.47	48.5		

(a) Matte from the Knudsen furnace, 68 per cent. copper.

¼ ton wood. It is to be noted that one forehearth is sufficient for two or even for three Knudsen furnaces.

Labor Employed.—The running of the Knudsen furnace requires two men per shift. These can attend to the charging, smelting, pouring out and also the cleaning. It is their duty to see that the charge is placed in the hopper over the furnace, ready for use, and that the coke and coal are on the grates. A foreman can easily look after two or three Knudsen furnaces. The generator and the forehearth can also be attended by two men per shift.

Cost of Operation.—The cost of smelting with a plant of two furnaces of 20 tons each, and one forehearth, with a generator, at normal prices for labor and materials, is about three marks (74c.) per ton of ore smelted, for the Knudsen furnaces, and 0.70 mark (17c.) per ton for the management of the forehearth. Air compression is not reckoned in, nor the chief repairs that are occasionally necessary to the hearth and the furnaces. In this statement, wages are reckoned at five and four marks per shift and the coke at 35 marks per 1000 kg., coal at 20 marks per 1000 kg., and magnesite bricks at one mark apiece.

It has been thoroughly proved that in the Knudsen process the loss of copper is less than in the former process of roasting and smelting of roasted ores in water jackets. This copper loss has been reduced by about 45 per cent. The accompanying smelting report covers a run at Sulitjelma from Sept. 26 to Oct. 31, 1908; in every case the charge stood for 1½ hours in the forehearth.

During the last three years, all smelting ores in Sulitjelma have been treated in Knudsen furnaces. The rich slags, however, were smelted in a water jacket, but this practice will be discontinued. During 1907, at Sulitjelma, 10,141 tons of ore were smelted, and all of this in Knudsen furnaces. The cost was as follows, in Norwegian crowns ($0.27) per ton: Unloading, transportation and carrying of the ore, coke, etc., into the works, 0.73; lining the Knudsen furnaces, 1.69; smelting (including air compression), 1.48; repairs, etc., 0.48; total, 4.38. For 1908 the cost of smelting appeared to be less.

As previously explained, the small furnace is uneconomical, for it requires the same attendance as the large one, and the amount of combustible material is about the same. It is therefore advantageous to use the larger furnace and to connect two of them to one hearth, thus insuring a cheaper production per ton of smelted ore. Experiments on the smelting of pyrrhotite ores containing nickel are well under way; copper ores, also, containing zinc blende have been smelted to a matte without difficulty, and with good results.

[Prof. H. O. Hofman in *Bull.* A. I. M. E., January, 1909, makes the following comment respecting the Knudsen process: "In this process, it will be remembered, copper-bearing pyrite is smelted in a converter

lined with dolomite. Practical experience has shown that the more basic —i.e., the richer in iron—the slag, the more quickly will the lining be slagged, while slags with a reasonably high percentage of silica do not corrode the lining as rapidly. The reason for this peculiar fact now finds its explanation in the formation of readily fusible calcium ferrate. The experiment given on page 68 has shown that magnesium ferrite does not form at a temperature as high as 1500 deg. C.; hence, the richer in magnesia the dolomite, the more resistant will it prove to slagging."—EDITOR.]

NEUTRALIZATION OF SMELTER SMOKE.

BY F. T. HAVARD.

In the conference of the counsel with Judge Hunt in the Anaconda smoke case, E. P. Mathewson described with much lucidity the processes in use for recovering arsenic from the smeltery fumes and dealt with the advantages, and the more numerous disadvantages of the methods proposed for overcoming what is styled the "smoke nuisance." Mr. Mathewson said that the arsenic in the ores of Butte ranges up to 2½ per cent., usually occurring in but small quantities. He stated that 90 per cent. of the ore which comes to the smeltery is sent to the concentrator where 50 per cent. of the arsenic is removed and thrown on the dump. Of the arsenic which reaches the smeltery in the ore, some is dumped with the furnace slags, a part is recovered in refinery drosses, and the rest goes into the flue in the form of white fume, consisting principally of the oxides. Arsenic is recovered from the fumes by redistillation; the method employed at Anaconda is the most thorough known. There are only two other smelters in the United States which collect the arsenic from the fume; one being at Everett and one at Salt Lake.

In telling the court of the methods by which arsenic might be extracted, Mr. Mathewson classified them as follows: *A*—Cooling processes: (1) water spray; (2) admission of air; (3) radiation; (4) freezing. *B*—Filtering processes: (1) bag house; (2) friction; (3) centrifugal gas cleaners. *C*—The Cottrell method. In commenting on the spraying method he emphasized the difficulty of disposing of the acid mud formed, and stated that a plant sufficient for the Washoe smeltery would cost $3,000,000. He disposed of the bag-house method by saying that a sufficiently capacious plant would cost $2,750,000, that the expense of its operation would amount to $1850 per day, with a recovery of arsenic to the value of $204, and that the life of the bags would be short. The installation of a radiation-cooling system would cost, in his estimation, $1,200,000, and would not be efficient. For the freezing system the pipes

alone would cost $4,000,000, and the cost of operation would amount to $10,800 per day. The system, further, has not proved practicable. A plant using zinc oxide as a neutralizer would cost $3,000,000, and would require 500 tons of zinc ore daily. With regard to the Cottrell method, Mr. Mathewson said that its application was not practicable on a commercial scale, except under very peculiar conditions. He concluded that if the friction system now used at Great Falls were successful, Anaconda would adopt it, although its installation might cost $2,000,000.

Recent Progress in Electrolytic Refining of Copper Bullion.

By Arthur L. Walker.

At the end of 1905, the capacity of the electrolytic copper refineries in the United States was taxed to the utmost. The quantity of pig copper produced in this country, requiring electrolytic refining, and the amount imported from Mexico, British North America, United Kingdom, and elsewhere was slightly in excess of 775,000,000 lb., which was then the output of the various plants. A small proportion of the copper produced in Michigan was treated by electrolysis, the only plant refining Lake copper in this way being that belonging to the Calumet & Hecla Company at Buffalo, N. Y. The next year and a half saw important additions to some of the largest refineries, namely, the Raritan Copper Works at Perth Amboy, N. J., the American Smelting and Refining Company's plant at Perth Amboy, N. J., the United Metals Refining Company's works at Chrome, N. J., and the Nichols Copper Company's works at Laurel Hill, N. Y., the additional capacity being about 389,000,000 lb. of copper per annum, bringing the total up to 1,164,000,000 lb., which for the last two years has been far in excess of the requirements. Consequently, during 1908 absolutely no extensions were made to the copper refineries. The following table gives some data in regard to the plants now in operation.

But little is known of the inner workings of the refinery of the Nichols Copper Company, except that its method of using the series system with cast anodes $\frac{3}{8}$ in. thick has been very successful. Wooden tanks lined with pitch are used and the solution is kept comparatively cool to prevent the pitch from flowing.

The original plant of the Raritan Copper Company had a capacity of nearly 13,000,000 lb. of copper per month, which was doubled by the building of the new plant in 1906-7. The power department of this latter unit (designed to generate power for the tank room at a low cost) is admirably arranged. It is claimed[1] that an indicated horse power is developed with 11.4 lb. of steam or with $1\frac{1}{2}$ lb. of buckwheat coal.

[1] *Electrochem. and Met. Ind.*, VI, No. 5, May, 1908.

ELECTROLYTIC COPPER REFINERIES IN THE UNITED STATES. (a)

Name of Plant.	Location.	Annual Capacity. 1908. Refined Copper, Pounds.	System.	Size of Anodes.		Current Density Amperes per sq. ft.
				Width in.	Depth in.	
Nichols Copper Company	Laurel Hill, N. Y.	300,000,000	Series	10	59	.....
Raritan Works	Perth Amboy, N. J.	300,000,000	Multiple	24	36	20
American Smg. & Ref. Company.	Perth Amboy, N. J.	144,000,000	Multiple	36	24	15
U. S. Metals Refining Company.	Chrome, N. J.	144,000,000	Multiple	36	36	21
Baltimore Cop. Roll. & Mfg. Co.	Baltimore, Md.	130,000,000	Series	24	22	15
Balbach Smg. & Ref. Co.	Newark, N. J.	48,000,000	Multiple	24	36	19
Boston & Montana Copper Co.	Great Falls, Mont.	48,000,000	Multiple	24	35	20–40
Tacoma Smelting Co. (b)	Tacoma, Wash.	18,000,000	Multiple	30	28	18
Mountain Copper Co. (b)	Oakland, Cal.	3,000,000	.....	.....	.....	.....
Chicago Copper Refining Co.	Blue Island, Ill.	2,000,000	Multiple	.....	.....	.....
Calumet & Hecla Mining Co. (c)	Buffalo, N. Y.	25,000,000	Multiple	27	29	10?
North American Lead Co.	Fredericktown, Mo.	2,000,000	.....			

(a) *The Mineral Industry*, Vol. X, 1901; Vol. XV, 1906; Vol. XVI, 1907. *Electrochem. and Met. Ind.*, Vol. VI, No. 6, June, 1908.
(b) Not operating at present.
(c) Refines Lake copper.

The new tank house building is 584 ft. long by 149.5 ft. wide, divided into three bays running lengthwise. In this building 1188 depositing tanks,[1] placed in 108 nests of 11 tanks each, are arranged for the new system of distributing bars, whereby an enormous amount of copper in the plant construction is saved. Each tank contains 24 anodes and 25 cathodes, the anodes being spaced 4⅝ in. center to center. A fairly rapid rate of circulation, 4 gal. of solution per tank per minute, is maintained, permitting the use of an electrolyte which is allowed to become quite impure. The use of diaphragms between anodes and cathodes, as suggested by F. L. Antisell (U. S. patent 875,641) is being tried at this plant with gratifying results. An extensive test along these lines is being made, and the outcome will be watched with interest. The diaphragm consists of rotary cut veneer placed in a frame. It is claimed by this system that a very high current efficiency is obtainable, and the loss of silver in the cathode is reduced to 10 per cent. of the normal, while the circulation of the electrolyte is not interferred with.

The silver refinery,[2] which treats the slimes from both the old and new tank rooms, is well arranged for the convenience of men and for safeguarding the material treated. After screening out the metallic copper, by means of a 40-mesh screen, the slime is boiled with concentrated sulphuric acid in lead-lined tanks, being agitated while boiling with paddles attached to a vertical shaft. The shaft and paddles are made of hard lead, containing 10 to 12 per cent. antimony. In boiling the slime with sulphuric acid in this manner, the copper content is reduced from about 20 per cent. to less than 1 per cent. in 24 hours, only sufficient copper being retained to insure no solution of the silver. This reduces

[1] *Electrochem. and Met. Ind.*, VI, No. 6, June, 1908.
[2] *Electrochem. and Met. Ind.*, VI, No. 7, July, 1908.

the amount of slag made in the doré silver melting furnaces to a minimum, and materially decreases the cost of handling by-products. The sulphate of copper solution from the dissolving tanks is transferred to the electrolyte in the tank room department. The slime is pressed, washed, dried, melted and cast into doré bars for parting, which is performed in porcelain tanks arranged according to the system devised by William H. Thum that has been in operation at the Balbach works for many years. The fume from the doré melting furnaces is passed through a bath of water by which means volatilized metals are condensed and caught. The system is very effective when the gas is passed through the water slowly enough to allow the latter to come in contact with the condensed particles in the gas.

The United States Metals Refining Company's plant at Chrome, N. J., was fully described in THE MINERAL INDUSTRY, Vol. XV. This article calls attention to the large anodes used, 3x3 ft., giving 18 sq.ft. of anode surface. The change in the size of the anodes was provided for by increasing the hight of the sides and ends of the tanks by 1 ft. The plan of working these large anodes has been successful and has resulted in a considerable decrease in the amount of labor required for handling. The new machine[1] for charging cathodes into the casting furnaces is operating very successfully. It handles loads of 5000 lb., picking up plates from a car and placing them in the furnace without difficulty. The copper can be packed better than by hand as it is piled more regularly and 300,000 lb. is charged in 75 minutes. This rapid charging, in connection with rapid casting on the machines, makes it possible easily to get out these large charges of fine copper.

At the American Smelting and Refining Company's plant at Maurer, near Perth Amboy, N. J., the addition to the tank-room capacity installed in 1906 was laid out on exactly the same plan as the tank room built in 1901-2. In making the anodes in this tank room 3 ft. wide by 2 ft. deep, heavy slime, due to the rich copper treated, has less chance to lodge on the cathodes in falling to the bottom of the tanks. This plant has treated much richer material than any other one, and the precaution was deemed necessary.

The Baltimore Copper Rolling and Manufacturing Company's plant at Canton, near Baltimore, which is now controlled by the American Smelters Securities Company, operates what is known as the Hayden system. The copper is cast into cakes which are then rolled into sheets 11 in. wide and ¼ in. thick, sheared into 24-in. lengths. These are placed in wooden frames which hold two sheets one above the other. These anodes, or stands as they are called, therefore consist of two sheets of

[1] *Eng. and Min. Journ.*, LXXXVI, 867.

copper, the dimensions being 22 in. high, 24 in. wide and ¼ in. thick. The stands are placed in slate tanks, each tank being divided into six compartments separated by slate partitions. In each compartment are about 135 stands. The current is delivered from the bus bar to an anode at the head end of each compartment and is conducted away from the cathode at the tail end of each compartment. It, therefore, passes through the six compartments of the tank in multiple and through the plates in each compartment in series.

Of the 12 refineries mentioned in the accompanying table the first five produce 85 per cent. of the total. Three of these five operate the multiple system and two the series. The advantages and disadvantages of these two systems have been mentioned in a general way.[1] Briefly, the claims for the series system are: (1) For a given amount of copper deposited less power is required. (2) Less copper and the precious metals it contains are required to conduct the process. (3) Less scrap is produced. (4) Less space is required for a given capacity.

The advantages claimed for the multiple system are: (1) Ability to treat copper of any quality, no matter how impure. (2) Less loss in precious metals in the cathodes produced. (3) Ability to handle material in larger units and with less cost for labor. (4) Requires less care in maintaining the quality of the electrolyte.

In the series system, where the current passes through the tank from one plate to the next one, dissolving the copper from the rear side of one anode, and depositing it on the front of the succeeding one, there is a heavy leakage through the solution under the plates, fully 30 per cent. of the current passing directly through the solution, instead of through the plates. The resistance between the plates is, however, very much less than it is between the electrodes in the multiple system, being only about one-half, or 0.15 volts per couple as against 0.3 volts in the latter. As a result, in the series system a given weight of copper will require only 7/10 as much power as in the multiple system, or a given amount of power will produce 1.4 times as much copper. In practice this would mean a saving of about 50c. or more per ton, depending upon the local conditions. As against this, however, the cost of casting thin anodes, in one case, by hand dipping from small furnaces of 60 tons capacity is very much greater than when casting anodes for the multiple system, by mechanical means entirely, from 150-ton furnaces. In the other case the cost of rolling thin anodes more than offsets the saving in power for depositing. Also the smaller cost for construction of a power plant for the series system is largely or entirely offset by the increased cost of casting equipment for making thin anodes or the rolling mill for rolling anode sheets.

[1] *Electrochem. and Met. Ind.*, VI, No. 6, June, 1908.

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COPPERAS.

The production of copperas (sulphate of iron) showed a slight increase in 1908 as compared with 1907. As in former years, the United States Steel Corporation furnished the bulk of the output. Practically the entire production was obtained by the makers of iron and steel sheet and wire as a by-product in acid cleaning. Wickwire Bros., of Cortlandt, N. Y., produced no copperas during 1907, but again entered the field as a producer in 1908. The E. I. DuPont de Nemours Powder Company, of Wilmington, Del., did not produce in 1908. Other producers were S. P. Wetherill Company, Philadelphia, Penn.; Pennsylvania Salt Manufacturing Company, Philadelphia, Penn.; and the Stauffer Chemical Company, San Francisco, Cal.

PRODUCTION OF COPPERAS IN THE UNITED STATES.
(In tons of 2000 lb.)

Year.	Short Tons.	Value.	Year.	Short Tons.	Value.	Year.	Short Tons.	Value.
1894	(a)	(a)	1899	13,770	$108,508	1904	16,956	$118,692
1895	14,118	$99,846	1900	12,374	96,517	1905	21,103	147,721
1896	11,170	52,062	1901	22,586	112,336	1906	22,839	228,390
1897	11,924	50,565	1902	19,784	118,474	1907	26,771	294,481
1898	11,285	58,105	1903	20,240	121,440	1908	35,334	388,674

(a) Statistics not collected.

The above table includes only the copperas recovered as a by-product, and disregards the production as an intermediate product in the manufacture of venetian red by certain paint makers.

Market and Prices.—The New York market was practically unchanged throughout 1908. Quotations were 55c. per 100 lb. for copperas in bulk; 65@75c. per 100 lb. in barrels; and 60@70c. per 100 lb. in bags.

CORUNDUM AND EMERY.

As a producer of corundum and emery the United States has practically withdrawn from the field. During 1908 there was no corundum mined, although a small production of emery was reported. The accompanying table shows the production and imports for a period of years.

STATISTICS OF CORUNDUM AND EMERY IN THE UNITED STATES.

Year.	Production. (a)		Imports.				
			Grains.		Ore and Rock.		Other Mfrs.
	Short Tons.	Value. (b)	Pounds.	Value.	Long Tons.	Value.	Value.
1897	2,193	$111,810	520,095	$20,022	5,209	$107,644	$2,211
1898	3,742	207,430	577,055	23,320	5,547	106,269	3,810
1899	3,970	228,570	723,229	29,124	7,435	116,493	11,514
1900	5,030	247,100	661,482	26,520	11,392	202,980	10,006
1901	4,305	146,040	1,086,729	43,217	12,441	240,856	10,926
1902	4,251	104,605	1,665,737	49,107	7,157	151,959	13,776
1903	2,542	64,102	3,595,239	109,272	10,884	194,468	17,829
1904	1,932	57,235	2,281,193	109,772	7,054	138,931	11,721
1905	(c)2,315	19,677	3,209,914	143,729	11,072	185,689	17,936
1906	(c)2,147	22,780	4,655,168	215,357	13,840	286,386	19,105
1907	(c)1,069	12,294	4,282,228	186,156	11,235	211,184	15,282
1908	790	10,360	1,845,366	89,702	8,084	146,105	12,592

(a) Statistics of the United States Geological Survey for 1901–1903. (b) Values have not much significance owing to the wide variation in the quality of the materials combined in the totals. (c) Emery only.

Practically the entire supply of corundum and emery during 1908 was imported from Canada, Turkey and Greece. American emery is not in favor with manufacturers of abrasive wheels. During 1908 about 15 plants in the United States manufactured abrasive products from imported material.

Corundum and Emery in Foreign Countries.

Reference was made in Volume XVI of The Mineral Industry to the corundum and emery industries in Greece and Turkey.

Canada.—Practically all of the pure corundum is obtained from the province of Ontario. Production has declined during the last three years, that of 1908 being less than one-half the production for 1906. The output for 1908 was valued at $96.62 per ton. The production and value of the Canadian output for a period of years is shown in the accompanying table.

THE CORUNDUM INDUSTRY OF CANADA.

Schedule.	1900	1901	1902	1903	1904	1905	1906	1907	1908
Production, tons...	60	534	1,137	1,119	1,665	1,644	2,274	1,892	1,039
Value.....	$6,000	$53,115	$83,871	$87,600	$150,645	$149,153	$204,973	$177,922	$100,389
Number of men....	35	68	95	186	202	.....	.....	.....	.....
Wages paid.....	$10,000	$30,406	$34,674	$106,332	$ 139,548	.....	.....	.....	.....

India.—Corundum is found at the three following places: (1) Patarknang, a village belonging jointly to the states of Mariaw and Rambrai, a few miles west of the Raciang river; (2) at Nongmaweit village; (3) on the Raindu river, a tributary of the Someshveri river. The material at Patarknang is coarse and is not regarded as valuable. The second and third deposits are close together and are situated a few miles west of the U Blay river, near its source. The material from this section is of good quality. At Nongmaweit the material occurs in small lumps and may be easily handled. It is obtained chiefly by quarrying. On the Raindu river the corundum occurs close to and in the river bed in large blocks, some of which weigh several tons. It is here that the best quality of material is found. However, it is not worked to any great extent, owing to the difficulty in cutting.

Manufacture of Emery Wheels.[1]

The Pittsburg Emery Wheel Company, of Pittsburg, has recently completed and put in operation at Rochester, Penn., a plant for the manufacture of emery and corundum wheels, with a capacity of wheels up to 36 in. diameter. Although the making of a completed wheel requires about three weeks from start to finish, the process is comparatively simple and the machinery required is not extensive.

The raw material for the wheels comprises emery, corundum or adamite mixed with a clay binder. The emery in its raw form contains considerable iron and in use may shrink unless it is burnt. Accordingly both the emery and the binder are burnt in a kiln which is built in the form of a muffle furnace. The clay is thoroughly dried and the emery or corundum burnt in a temperature of about Seger cone 10. This burning removes from the emery all tendency to further shrinkage and it can then be put in the wheels without danger of their cracking or becoming imperfect in use.

Molding.—After this preliminary burning, the first process in the making of the completed wheel is molding. The clay binder and the emery, corundum or adamite, as the case may be, are mixed with water in kettle-shaped vats to a consistency of thick mud. The proportions of emery

[1] *Iron Trade Rev.*, Aug. 13, 1908.

and clay are varied as are the grades of emery used to suit the conditions under which the wheel is to operate. A soft wheel in general possesses a relatively greater amount of binder than a hard wheel. After mixing, the wheels are molded on plaster of paris "bats." These consist of circular plates of plaster of paris properly baked and dried, and on which is mounted a galvanized iron ring of the diameter to which it is desired to mold the wheel. A sheet of paper is placed over the plaster of paris bat and inside of the metal ring to prevent the wheel from adhering when it is dry.

Drying.—The next operation is air-drying of the wheel. After it has been molded, the metal ring is removed and the wheel allowed to stand on the bat until it has attained a certain firmness. It is then removed to the upper floor of the kiln room where the temperature is rather high, and allowed to stand for a few days until it is fairly hard and firm enough to be trued up in the shaving machine. This shaving machine bears a close resemblance to the ordinary potter's wheel. It comprises a vertical spindle carrying a revolving platen on which the wheel to be trimmed is placed. A pair of standards carry a cross rail, extending across the platen, on which trimming knives are carried. These are adjustable to any hight, and by running them across the face of the wheel as it is revolving on the platen, the form is cut down to any thickness desired. It is also trimmed roughly to the proper diameter by feeding the knives down vertically on the edge of the wheel. This shaving machine is also used in forming odd-shaped wheels such as cup wheels, etc.

Final burning.—After shaving, the wheels are transferred to the kilns for final burning. The kilns are four in number, 20 ft. in diameter, and strongly banded with iron. They are coal-fired, each kiln being provided with seven fireplaces. The wheels are placed in the kilns in piles separated from one another by tall plaster of paris bats, and after bricking up the entrance the kiln is lighted. The passage of the gases from the fire at first is into the kiln chamber, up through the wheels and out of a damper at the top. As the burning proceeds, this damper is gradually closed until at the end the passage for the gases is into the chamber at the top, then through holes in the floor, through passages under the floor and up through an annular space in the kiln wall. When this stage of the burning is reached, the kiln is at a white heat. The burning proceeds for three or four days, and then the fires are drawn and the wheels allowed to cool, which process takes about three days.

Trimming.—The final operation in the manufacture of the wheels is the trimming to exact sizes. This is done in four trimming lathes, three of which are operated on the side of the wheel and the third one on the

face. The lathes are strongly built machines with both cross and longitudinal feeds to the carriage, which is provided with a compound rest to allow wheels to be trimmed to any angle. For instance, some wheels are thinner at the circumference than at the hub, and others thicker at the circumference. These latter wheels are used in universal grinders and are made in this manner in order to provide clearance when working in slots or similar situations. After truing, a lead center is poured into the hole in the wheel and bored to the proper diameter to fit on the arbor, and then the wheels are tested for bursting strength.

Testing.—This testing is done in a machine which comprises simply a pair of arbors running in strong steel casings provided with doors. The arbors are provided with flanges and nuts on which the emery wheels may be mounted, and after the doors are closed the wheels are speeded up until they attain a rim velocity of about 1½ miles per min. They are subjected to this speed for about five min. and in that time any defects in their construction will become apparent. If the wheel bursts the pieces are confined within the steel casing. Varying speeds are necessary for varying diameters, and these are obtained by means of a pair of Evans friction cones and suitable cone pulleys on the countershafts.

Steel flanges are used to confine the wheel, these being made in a machine shop adjoining. The flanges are clamped on either side of the wheel, and are steel castings. As the wheel wears down the flanges are removed and smaller ones substituted until the wheel becomes too small for any further use. The use of flanges extending almost to the rim of the wheel tends to strengthen it and prevent undue strains leading to bursting. As the emery worn from the wheels has a deleterious effect on the machinery if allowed to accumulate and work into the bearings, provision has been made for its removal by means of exhaust fans. These are piped directly to the machines and exhaust outdoors, so that the air in the shop is remarkably pure.

CRYOLITE.

The supply of cryolite in the United States is furnished entirely by imports, which amounted in 1908 to 1124 long tons, valued at $16,445. The business is entirely in the hands of the Pennsylvania Salt Manufacturing Company, of Natrona, Penn. The imports of cryolite and the value of the same imported into the United States during recent years are given in the accompanying table.

IMPORTS OF CRYOLITE IN THE UNITED STATES.

Year.	Long Tons.	Value.	Av. per Ton.	Year.	Long Tons.	Value.	Av. per Ton.
1899	5,879	$ 78,676	$13.38	1904	959	$13,708	$14.30
1900	5,437	72,763	13.37	1905	1,600	22,482	14.05
1901	5,383	70,886	13.17	1906	1,505	29,683	19.72
1902	6,188	85,650	13.84	1907	1,284	28,902	22.51
1903	7,708	102,879	13.35	1908	1,124	16,445	14.63

Cryolite is used in the manufacture of sodium salts, the manufacture of certain special kinds of porcelain and glass, and as a flux in aluminum smelting. For the last purpose, however, the demand has lately been greatly diminished by the substitution of a product derived from fluorspar. By far the major portion of the cryolite imported into the United States is employed in the manufacture of sodium salts.

FELDSPAR.

By Albert H. Fay.

There were few events of importance in the feldspar industry in the United States during 1908. The volume of business reported was from 15 to 25 per cent. below that of 1907. At the close of the year, however, mills were running almost at full capacity. In Virginia one new company began production in 1908.

FELDSPAR IN THE UNITED STATES. (a)
(In tons of 2000 lb.)

Year.	Crude.		Ground.		Total.	
	Quantity.	Value.	Quantity.	Value.	Quantity.	Value.
1901	9,960	$21,690	24,781	$198,753	34,741	$220,422
1902	21,870	55,501	23,417	194,923	45,287	250,424
1903	13,432	51,036	28,459	205,697	41,891	256,733
1904	19,413	66,714	25,775	199,612	45,188	266,326
1905	14,517	57,976	20,902	168,181	35,419	226,157
1906	39,976	132,643	32,680	268,888	72,656	401,531
1907	31,080	101,816	53,469	397,253	84,549	499,069
1908	.....	.....	.....	.....	59,180	319,570

(a) Statistics reported by the U. S. Geological Survey, except for 1908.

Prices.—During 1908 the feldspar industry, in common with other mining interests, suffered more or less from the general business depression. The average price fetched by good quality crude feldspar was about $3.50 per long ton, f.o.b. cars at the mine. The price for the ground material was $7@8 per short ton.

Feldspar Mining in the United States.

Maine.—No new properties were opened during 1908. The principal deposits operated were those owned by Golding Sons' Company, at Georgetown, and the Trenton Spar Company, near Brunswick. A small amount of work was done at Poland. The Trenton Spar Company's mines have been worked for at least 30 years. The mining is all open-cut work. All the rock mined is ground by the company, only a limited supply being shipped to a few regular customers. The plant was in operation during all of 1908, but not quite at its full capacity. The Georgetown deposit is on the Kennebec river. Canal barges are loaded

at the mine and towed direct to Trenton, N. J., without transferring the rock. This deposit is also an open-cut mine and has been operated by Golding Sons' Company for 32 years. For a number of years before this company purchased the property it bought the rock from the original owner. Most of the feldspar from Maine is classed as second grade, but by close hand picking a small amount of first-class rock is obtained. The principal impurities are quartz and mica.

New York. (By D. H. Newland.)—The small demand for feldspar among pottery manufacturers in 1908 was reflected in a decreased output of the better grades of the mineral, i.e., the potash feldspars free from iron or other obnoxious impurities. On the other hand, the production of unselected feldspar, such as finds employment for roofing material, poultry grit, abrasive purposes, etc., showed a substantial gain over the quantity reported in 1907.

The occurrence of pegmatite, from which feldspar is obtained, is limited in this State to the Adirondack Precambric area and the southeastern region of crystalline rocks. The pegmatite forms dikes, sheets or irregular masses, usually in association with larger masses of allied intrusive rocks of which it constitutes a coarser phase. Most occurrences have the composition of granite, containing besides feldspar a considerable proportion of quartz and smaller amounts of mica, hornblende or rarely pyroxene. The presence of iron-bearing minerals has an important bearing upon the quality of the product, since for pottery purposes it is absolutely essential that the feldspar contain nothing more than a trace of iron. When in segregated particles their elimination can be effected sometimes by careful cobbing. In addition to those already enumerated, pyrite, magnetite and tourmaline are among the iron-bearing minerals commonly found in pegmatite. For pottery uses the potash feldspars (microcline and orthoclase) are the only ones produced in New York and they seem to have the preference in that trade, though for what reason is not well understood. The plagioclase feldspars are found in the Adirondack pegmatites which are chiefly quarried for roofing material.

The quarries now active in New York are situated in Westchester, Essex and Saratoga counties. Those near Bedford, Westchester county, produce most of the pottery feldspar. They are operated by P. H. Kinkel's Sons who grind a large part of the output before shipment. The quartz obtained in the quarry operations also finds sale as a material for making wood filler. In the Adirondacks, the Claspka Mining Company, with a quarry near Batchellerville, Saratoga county, produces a small quantity of pottery feldspar, which is shipped crude to Trenton, N. J. The Crown Point Spar Company, is a recent enterprise, with quarries and a mill near Crown Point. The property at Rock pond, Essex county,

formerly owned by the International Mineral Company, is now operated by the Barrett Manufacturing Company, which uses the output in the manufacture of "amatite" roofing.

Some attention was given during 1908 to an occurrence of pegmatite near Corinth, Saratoga county. The American Feldspar and Milling Company was organized to develop the property, but no shipments were made, except for experiment. The pegmatite is stated to occur in a dike 60 ft. wide and traceable for 2000 ft. on the line of strike. The outcrop is about three miles distant and 700 ft. above the railroad.

The total production of feldspar in New York in 1908 amounted to 15,312 short tons, valued at $54,550. The prices received at the quarries for the pottery grades ranged from $3 a long ton for crude to $6.50 a short ton for ground feldspar. The common grades of crushed feldspar brought about $3 a short ton.

Pennsylvania.—The Brandywine Summit Feldspar and Kaolin Company owns and operates a mill at Brandywine Summit.. The mining is all open-cut work. All the feldspar is ground before shipment to the potteries. About 20 men were employed per day at the mine and mill during 1908.

Virginia.—The Bedford Spar Company, of Lowry, began shipping in 1908. The feldspar occurs at the surface and is obtained by quarrying. The quarry is seven or eight miles from the railroad and all of the feldspar is hauled by wagon. The material is of a good grade; it is white, free from quartz, and grinds easily. The mining work was conducted during only a portion of 1908, but was resumed toward the end of the year.

Uses for Feldspar.

Practically all of the high-grade feldspar produced in the United States is used in pottery work. A notable quantity is consumed in the manufacture of certain cleansers, and less amounts as a filler in paints. Some experiments have been made at Henryton, Md., and elsewhere, with a view to the use of ground feldspar as a fertilizer, it being contemplated that potash may be secured in this way more cheaply than from the imported potassium salts. Another use of feldspar is for surfacing some of the tar and felt compositions for ready roofing. In this work feldspar takes the place of ordinary gravel, but it is superior to the latter because of its pleasing light color. For this purpose the presence of mica in the feldspar is not objectionable; indeed, it is rather desirable.

In order that the feldspar may be used in the best grade of pottery, china, etc., it must be free from iron. A very small amount of iron will discolor the entire surface, and if the iron is concentrated, as in the case

of mica scales, the discoloration will occur in spots. The formulas used by the potteries vary greatly as to the percentage of feldspar added to the china clay, the range being from 10 to 50 per cent., although sometimes as high as 60 per cent. is used for some of the highest grades of ceramic tiles.

Preparation of Feldspar for Market.

The grinding of feldspar and flint is a simple process. Preliminary crushing is done by chasers and final grinding by ball mills. The Golding Sons' Company plant, at Trenton, N. J., which has been in practically continuous operation for 65 years, typifies the practice in this country. This company's equipment consists of storage platforms for the crude material; bins for the finished product; 140 h.p. steam power plant; 125 h.p. water turbine; three calcining furnaces for French pebbles; 11 ball mills; two sets of chasers, and the necessary buildings to house the equipment. The capacity of the plant is 300 to 350 tons per week.

Chasers.—These are similar to the Chilean mill and consist of two wheels of hard quartz rock 18 in. thick, 7 ft. in diameter, weighing approximately 8000 lb. each, when new. These wheels are mounted on an axle attached to a vertical shaft and driven by bevel gears from a line shaft. The chasers run on a stone about 6 ft. in diameter and 1 ft. thick. The feldspar is shoveled by hand upon this stone and the chasers crush it by their enormous weight and grinding action. The axle is so attached to the vertical shaft that as the wheels wear they are allowed to settle down and always bear their entire weight on the material to be crushed. The chasers make about 20 to 25 r.p.m. The capacity of a pair of these chasers is 28 tons in 24 hours. While steel rollers could be used to crush the spar, the great objection to them is the fact that in their abrasion, a large amount of iron will find its way into the finished product and thus deteriorate it very much. The smallest particle of iron will show as a dark spot in the finished porcelain.

From the chasers the crushed rock is elevated to a 20-mesh trommel. The undersize is collected in 1-ton cars and trammed direct to the ball mills. The oversize goes back to the chasers to be re-crushed.

Grinding.—In the early days the grinding of feldspar by this company was conducted in pans similar to the Mexican arrastre. These pans were 12 ft. in diameter, 3 ft. deep, and the bottom was lined with a hard rock from Ulster county, N. Y., only slightly inferior to the best grade of bur stone. This bottom was about 1 ft. thick. The sides were wooden staves. In the center of the pan was a vertical shaft with gudgeon bearing at the bottom, revolved by means of a bevel gear. To this vertical shaft were attached four wooden arms, which were only about an inch

above the bottom of the pan. In front of these arms and resting on the bottom was placed a stone about 1 ft. square and 4 ft. long. As the arms revolved the stones were pushed forward. The grinding was accomplished by the rubbing of this stone on the bottom. A single charge consisted of 3000 lb. feldspar crushed by the chasers to about 20 mesh. The time required for grinding was 24 hours. Water was used in this process.

Drying.—The pulp from the pan passed through a 140-mesh trommel. All that was fine enough to go through the screen went into a vat, and was allowed to settle and the water drawn off. The pulp was shoveled to the top of a furnace to dry. This furnace was made of brick, and covered with flag-stone, the drying floor having an approximate area of 300 sq.ft. The furnace was fired with coal. The heat from the exhaust steam pipe from the power house was also used in a series of pipes placed under the flagstone top. Since the introduction of the dry grinding this dryer has not been used.

Ball Mills.—The system of wet grinding has been supplanted by a more modern one in which 11 ball mills are used. The ball mills are 6x8 ft. with silex lining and are charged with four tons of French pebbles; one charge of pebbles will last several weeks. About four or five tons of crushed spar passing a 20-mesh screen are charged every six hours. The material is ground to 140 mesh, when it is discharged on the floor under the mill. The ground feldspar as it comes from the ball mill is slightly dampened with water in order to facilitate future handling. It is then shoveled into the storage bins below the mills.

It is seldom that more than five of the mills are in operation at one time. One set of chasers will crush enough spar for the five mills. The horsepower required for the operation of this one unit is approximately 125, which is furnished by water from the Delaware river. The water is led to the mill by a canal extending several miles up the river, and at the mill it furnishes a 15 ft. head for the turbine.

French Pebbles.—About 1500 tons of pebbles are imported and ground each year by this company. This product is also used in the pottery business. The small amount of pebbles required in the ball mills is selected from these. Before grinding the pebbles it is necessary to calcine them; this is done in a shaft furnace in which 15 tons of pebbles are charged with one cord of wood. One-half of the wood is placed at the bottom of the furnace and then a layer of pebbles, then another layer of wood, until there are three layers of wood and three layers of pebbles. The time required for burning is about 15 hours. This softens the pebbles so that they grind quite easily under the chasers, and in the ball mills.

Feldspar.—The crude rock ground at this plant comes from Georgetown, Maine, and recently a new supply has been obtained from Lowry, Va. The ground feldspar that is to be shipped by railroad is either sacked in 160-lb. sacks, or put up in barrels of 370 to 400 lb. each. The larger part of this company's product, however, is delivered by wagon to the potteries in Trenton. In addition to the grinding plant at Trenton, Golding Sons' Company own similar plants of like capacity at Wilmington, Del., and East Liverpool, Ohio.

FLUORSPAR.

By Albert H. Fay.

The fluorspar deposits in the United States occur in Kentucky, Illinois, Colorado, Tennessee and Arizona. The largest operations are in southern Illinois and western Kentucky. A few years ago there were about 200 companies engaged in mining fluorspar. Through reorganization and consolidation there are now only about a dozen operating concerns of importance.

PRODUCTION OF FLUORSPAR IN THE UNITED STATES.
(In short tons.)

Year.	Tons.	Value.	Per Ton.	Year.	Tons.	Value.	Per Ton.
1899	24,030	$152,655	$6.35	1904 (a)	36,452	$234,755	$6.44
1900	21,656	113,430	5.24	1905	39,600	232,452	5.87
1901 (a)	19,586	113,803	5.81	1906	34,683	201,481	5.78
1902 (a)	48,018	271,832	5.19	1907	36,350	202,736	5.58
1903 (a)	42,523	213,617	4.28	1908	39,389	212,647	5.40

(a) Statistics of the U. S. Geological Survey.

Market and Prices.—New York quotations were steady throughout 1908 as follows: Domestic, f.o.b. shipping port, lump, $8@10 per 2240 lb.; ground, $11.50@13.50. Foreign, crude, ex-dock, $8@10 per 2240 lb. Concessions from the above prices were obtainable owing to the competition with English material. The average prices received by the producers in this country during 1908 were as follows: Ground, $9@11; lump, $5@6; unwashed gravel, $4@4.50; washed gravel, $4.50@5.50.

Fluorspar Mining in the United States.

Arizona.—The Castle Dome Mining and Milling Company, of Dome, Yuma county, produced a large amount of fluorspar in 1908. Its shipments, however, were not large. The fluorspar is obtained in connection with lead ore.

Illinois. (By Frank W. DeWolf.)—During 1908 there was but a slight increase in the production of flourspar in the Illinois district, though the region was very active during the last few months, when the Fairview mine is reported to have been shipping at the rate of 4000 tons per month. The general business depression affected the industry much the same as in 1907. It is reported that the Rosiclaire mine will be equipped with new

milling machinery and that other improvements in the district are to be taken up at once. It seems probable that several mines other than those now in operation will be opened up during 1909.

According to figures received by the State Geological Survey, the total production of the district amounted to nearly 27,000 tons, with a value of approximately $156,000. A small amount of the production remained unsold at the end of the year and the spar actually placed on the market was slightly less than in 1907. With the increasing demand for fluorspar in the manufacture of open-hearth steel, especially with the growth of the new plant at Gary, Ind., it seems probable that the Illinois mines will feel a much stronger demand than in the past.

Kentucky.—Foreign competition caused many of the Kentucky operators to suspend work in 1908. The principal producers during the year were the Sunnybrook Lead and Fluorspar Company, the Kentucky Fluorspar Company, the Albany Mining and Investment Company, the Blue Grass Fluorspar Company, the Indiana and Kentucky Fluorspar and Lead Mining Company. The Marion Zinc Company, of Marion, did not operate. John W. Wilson, Marion, shipped some fluorspar, and expects to make a large output in 1909. The Pope Mining Company, Louisville, prosecuted only development work during 1908.

Fluorspar in England.

The increased use of fluorspar as a flux in metallurgical works during the last few years has led to great development in the mining of the mineral. Until 1902 the bulk of the output came from the mines of the Weardale Lead Company, Durham, but since that year Derbyshire has gone rapidly

FLUORSPAR OUTPUT OF THE PRINCIPAL PRODUCING COUNTRIES.
(In metric tons.)

Year.	France.	Germany.	Spain.	United Kingdom.	United States.
1897	2,722	23,232	2	302	3,973
1898	3,077	23,787	5	57	11,021
1899	5,140	24,306	310	796	21,806
1900	3,430	30,310	4	1,471	19,646
1901	3,970	28,741	*Nil.*	4,232	17,768
1902	2,650	(a)14,177	93	6,388	47,190
1903	2,447	(a)13,028	4,000	12,102	38,577
1904	2,047	(a)13,540	(b)	18,451	33,069
1905	2,434	(a)15,019	(b)	38,606	35,209
1906	4 218	(a)15,493	70	36,860	28,657
1907	4,795	(a)16,624	270	40,873	32,969
1908	(b)	(a)14,702	(b)	25,395	35,738

(a) Exports. German statistics no longer report production. (b) Not reported.

ahead, and Durham now takes the second place. This has been brought about chiefly through the working of the old lead mine dumps, in which the fluorspar exists as the waste material from lead mining operations

dating from remote times. At present there are about 20 workings for fluorspar in Derbyshire, and with the exception of one or two at Ashover, in the isolated mining district to the east of the county, these are all in the northern district, Great Longstone, Eyam, and Bradwell being the principal points of activity. The great master vein on Longstone Edge, known in the West as the High Rake and in the East as the Deep Rake, is a large producer of fluorspar, which is now being got from virgin ground as well as extracted from old dumps. In Durham, the whole of the output of fluorspar comes from the neighborhood of Weardale.

Uses of Fluorspar.

Fluorspar is used extensively as a flux in the metallurgy of iron and steel, because of the great fluidity which it gives to open hearth slags. It is chiefly used in the basic process and should not contain more than 3 or 4 per cent. silica. By the use of fluorspar in the manufacture of ferro-silicon an iron alloy containing 10 per cent. silicon can be obtained in an ordinary blast furnace charged with silicious iron ore. It is also claimed that it is better than limestone flux in that it requires less fuel to reduce the charge, and an increase in the percentage of iron recovered. Low-grade fluorspar is also used to some extent in iron and brass foundries, as it renders the metal more fluid. It also has a limited use as a bonding element in the manufacture of emery wheels. This mineral is also used in the glass industry to the extent of probably to 10 or 15 per cent. of the total output.

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FULLER'S EARTH.

The production of fuller's earth in the United States during 1908 showed a slight decrease, due chiefly to the general industrial depression. The bulk of the output came from Florida, although Georgia, Alabama, Arkansas and California contributed small amounts. The principal producers were the Owl Commercial Company, Quincy, Fla.; Atlantic Refining Company, Philadelphia, Penn.; Lester Clay Company, Attapulgus, Ga.; Arkansas Fuller's Earth Company, Chicago, Ill.; and Southern Fuller's Earth Company, Warren, Penn. (mines and mill at Jamieson, Fla.). The production and imports of fuller's earth for a period of years is shown in the accompanying table.

STATISTICS OF FULLER'S EARTH IN THE UNITED STATES.
(In tons of 2000 lb.)

Year.	Production.		Imports.		Year	Production.		Imports.	
	Sh. Tons.	Value.	Sh. Tons.	Value.		Sh. Tons.	Value.	Sh. Tons.	Value
1899.....	13,620	$ 81,900	11,558	$69,460	1904.....	29,480	$168,500	10,221	$ 74,000
1900.....	11,813	70,565	9,154	64,790	1905.....	25,745	157,776	15,181	105,997
1901.....	14,112	96,835	12,061	80,697	1906.....	28,000	237,950	14,827	168,696
1902.....	14,100	109,980	15,135	102,580	1907.....	34,039	323,275	14,648	122,221
1903.....	20,693	190,277	17,100	120,671	1908.....	30,517	270,685	12,279	92,413

By the close of 1908 the American Fuller's Earth Company, of Fort Worth, Texas, had completed its plant and was ready to ship. The Southern Fuller's Earth Company overhauled and enlarged its plant at Jamieson, Fla. Early in 1909 this company was producing double its output for the corresponding months of 1908.

Prices and Commercial Conditions.—Fuller's earth, both powdered and lump, was steady throughout 1908 at 75@85c. per 100 lb. in large lots. At the mines the material produced had a value of between $8 and $9 per ton. Producers in this country are still forced to compete with the English fuller's earth. The English producers have two factors in their favor—comparatively low cost of mining and transportation, and superiority of product. Although the American producers have not the duty of $3 per ton on the finished product and $1.50 per ton on the crude to contend with, high freight rates from the mine to the consumer

present to them a troublesome problem. For many years the English earth was the only one which could be used for bleaching edible oils, the American product imparting a decided taste and odor when used for this purpose. Recently, however, deposits have been uncovered in this country which have been pronounced superior to the English earth, and it is thought that the importation of fuller's earth will now gradually diminish.

FULLER'S EARTH IN FLORIDA.

By E. H. SELLARDS.

Florida fuller's earth occurs irregularly in bedded deposits. The outcrops are found usually along the banks of streams, and in other natural exposures. The thickness of the fuller's earth stratum varies from a few inches to several feet. Those beds which are being worked range from 4 to 12 ft. in thickness. The overburden is variable both in character of material and in thickness. In some hilly sections of central west Florida, fuller's earth may be found overlaid by the Grand Gulf and Lafayette formations to a depth of 100 or 200 ft. The workable deposits, however, are those which have an overburden not exceeding 10 or 15 ft. The overburden is usually either sand or clay or a sandy clay mixture. Occasionally a somewhat indurated sand or marl stratum occurs. The following is a section taken at one of the pits in Gadsden county: (1) Sandy red clay indistinctly stratified, 12 ft.; (2) greenish, more sandy, indurated clay, 4 ft.; (3) fuller's earth stratum, 6 ft.

The overburden is removed by steam shovel, by pick and shovel, or by team and scraper. The fuller's earth is handled by pick and shovel, being first broken up when necessary by explosives. At the pit the clay is usually loaded into dump cars and drawn to the crushing mill. After being broken the clay passes into a drying cylinder and afterward into a mill for grinding. After being ground the material is passed through sieves in order to separate it into the size of grain desired for commercial purposes. The sieves used vary from 10- to 100- or 140-mesh. The prepared product is sacked for shipment and is labeled according to the degree of fineness to which it has been ground.

Judging from reports thus far received the output of fuller's earth in Florida in 1908 exceeded 20,000 tons, but fell somewhat short of the output of the previous year, which was 24,148 tons. This slight falling off is due doubtless to the general business depression of the year, and does not indicate a permanent reduction. A new plant at Attapulgus, Ga., was in operation during the latter part of 1908. This plant is situated just across the Florida line, and the deposit worked is a northward extent of the Florida deposits. Florida fuller's earth is now find-

ing a market in foreign countries, 10 to 15 per cent. of the output being exported.

Technology of Fuller's Earth.

Properties and uses, mining and preparation, and the question of English vs. American earths, were treated in Vol. XVI, of The Mineral Industry.

Composition.—The composition of fuller's earth from different localities varies widely; the term may be applied to a variety of clayey substances that possess strong absorbent properties. It is believed to be a decomposition product of hornblendes and augites, rather than of feldspars, as is the case with ordinary clays. The following range of composition is shown in a table of 26 analyses published by Porter: SiO_2, 39.66 to 72 per cent.; Al_2O_3, 5 to 33.38; H_2O, 4.30 to 24.95; Fe_2O_3, 1.26 to 14.87; CaO, 0.33 to 7.40; MgO, 0.43 to 4.36; alkalies, 0.45 to 1.84; other substances occasionally found associated with fuller's earth are P_2O_3, TiO_3, P_2O_5, SO_3, NaCl, organic matter and volatile matter. No definite formula can be stated for fuller's earth owing to its wide range of composition.

Preparation in Scotland.—In Scotland the only known deposit of fuller's earth is at Keltie, Dunning, and Perthshire. The preparation of the raw material is carried on in various ways according to the variety of the material and the purpose for which it is required. The simplest process is that in which the crude earth is dug and kiln dried, while it may be further treated by being ground to a fine powder, say 120 mesh, in which latter form it is largely used by manufacturers for refining purposes. In other cases the crude earth, after being mined or dug, is ground and washed, sand and other impurities being taken out by specially constructed sand traps and revolving sieves, then deposited in large tanks and afterward kiln-dried, when it takes the form of lumps and is used for cleansing and fulling cloth, etc. The lump earth is also reground, purified and reduced to different grades and classes of powder according to the purposes for which it is required.

Treatment of Oils.—In treating mineral oils the earth is usually dried carefully after it has been ground to 60-mesh; it is then run into long cylinders, through which the crude black mineral oils are allowed to percolate slowly. The issuing oil is perfectly white in color, that which comes out first being markedly thinner than that which follows. The oil is allowed to continue percolating through the earth until the color reaches a certain maximum shade; then the process is stopped, to be continued with a new portion of earth. The oil is recovered from the spent earth.

In treating vegetable oils the process differs radically. The oil is heated in large tanks above the boiling point of water; from 5 to 10 per cent. of its weight of fuller's earth is then added, and the mixture stirred vigorously for about 10 minutes. It is then filtered off through bag filters. The coloring matter remains with the earth, leaving the oil of a very pale straw color, provided the original cottonseed oil has been sufficiently well refined by the ordinary process to admit of this, and provided the operation has been conducted with sufficient care. A remarkable feature of the filtration is the different rates of speed at which oils of different density will percolate through, with the result that the first oil which makes its appearance is not only much lighter in color, but markedly lower in specific gravity. In fact, by this process, separations can be made which are comparable with the results of fractional distillation.

Clarifying Action.—John T. Porter offers a new theory to explain the clarifying action of fuller's earth. It is based on the presence of a group of aluminum hydrosilicates existing in the form of pectoids. Mr. Porter writes as follows: "The theory which I have devised to explain the peculiar properties of fuller's earth may be stated as follows: (1) Fuller's earth has for its base a series of hydrous aluminum silicates. (2) These silicates differ in chemical composition. (3) They are, however, similar in that they all possess an amorphous colloidal structure. (4) The colloidal structure is of a rather persistent form and is not lost on drying at a temperature of 130 deg. C., or possibly higher. (5) These colloidal silicates possess the power of absorbing and retaining organic coloring matter, thus bleaching oils and fats.

"I have used the word colloidal in this statement in its broadest sense—to cover the whole range of conditions expressed by the words colloid, pectoid, and hydrogel. It is my opinion that the word pectoid would most properly express the condition of the active constituents of fuller's earth, but it is not impossible that these may go into partial solution in oil and thus become true colloids. Colloids are known to possess the power of taking up organic colors from solution, and the analogy with fuller's earth is so apparent as to excite surprise that it has not been studied from this point of view before. It has even been known that pectoids could extract colors from oils and resins as well as aqueous solutions. This I regard as a strong point in favor of the colloidal theory, although by itself it does not, of course, constitute a proof. The power of colloids to absorb certain salts, or at least the basic iron of these salts, has been known a long time. It is also possessed to a degree by certain amorphous substances having a fine porous structure, such as charcoal and boneblack. Fuller's earth has this property to a

marked degree. In fact, its use has been proposed to remove the lime from boiler water. I have found that after it has absorbed ammonia or salts it loses a great part of its efficiency in bleaching oils. From this I infer that the bases are absorbed in a similar manner to the coloring matter of oil, and occupy the pores which otherwise would hold the color.

"It has been pointed out that these absorbent materials have a selective action on the salts, absorbing the unlike ions and discarding the ions of like chemical nature. If colors in oil solution are absorbed in the same manner as salts in aqueous solution, analogy would lead to the supposition that fuller's earth would exert a selective action for certain classes of coloring matter and, moreover, that the earths themselves would differ in their selection of colors according as they are more or less acidic in composition. We find this entirely in accordance with the facts. I have noticed in the course of my oil tests that on the same sample of oil different materials give products varying greatly in shade, the main color being in some tests of a yellow and in others of a green tint. Is it not likely that the oil contains several compounds of varying chemical nature, and that the earths or other materials used in bleaching extract them in ratios proportional to their own basicity or acidity? I have been unable to obtain any exact data on this point from my results, owing to the difficulty of following these slight changes in tint with the unaided eye. A tintometer would be needed if this line of investigation were to be followed out."

GARNET.

New York is still the chief source of the abrasive garnet supply of the United States. The deposits of Pennsylvania and North Carolina furnish but a comparatively small proportion of the total production. The industry in New York is fully reviewed in the following paragraphs. The amount and value of the garnet produced in the United States for a period of years is shown in the accompanying table.

PRODUCTION OF GARNET IN THE UNITED STATES.
(In tons of 2000 lb.)

Year.	Short Tons.	Value.	Value per Ton.	Year.	Short Tons.	Value.	Value per Ton.
1899	2,565	$ 72,672	$28.33	1904	2,952	$ 89,636	$30.36
1900	3,285	92,801	28.25	1905	3,694	114,625	31.01
1901	4,444	158,100	35.51	1906	5,404	179,548	33.22
1902	3 722	122,826	33.00	1907	6,723	201,995	31.22
1903	4,413	146,955	33.30	1908	2,530	78,090	30.86

New York. (By D. H. Newland.)—The abrasive garnet trade afforded no exception to the general course of business conditions during 1908. The Adirondack mines supplied, as heretofore, the greater part of the production of garnet for abrasive uses, but owing to the poor demand their output was reduced to less than one-half the total of 1907. The amount, as reported by the three active companies, was 2,480 short tons, valued at $79,890; in 1907 the output was 5709 short tons, valued at $174,800. There was no corresponding decline in the prices obtained for the garnet, since the margin over the cost of mining is not large enough to admit any material reduction. Most of the output, also, was sold on contracts covering the annual requirements of the consumer.

No new companies were engaged in mining in the Adirondacks during 1908. The North River Garnet Company, with mines and a mill on Thirteenth lake, was the main producer. Since October, 1907, this company worked a single shift only and suspended operations entirely for a period of three months last spring. On Gore mountain the mines of H. H. Barton & Son were run on a reduced scale. The Crehore mine near North river was inoperative throughout the year. A small output of massive garnet was reported from the new locality near Keeseville in

the northern part of Essex county. This grade of garnet is mostly sold abroad.

A recent feature of the trade has been the introduction of foreign garnet which is finding a limited market in this country. It comes from Spain, where it is said to occur in river sands, doubtless a product of the disintegration of crystalline rocks. The imports of abrasive garnet amounted to a value of $6432 in 1907, but fell off in 1908 to a value of $2095. They were entered at the ports of New York and New Orleans, and were given a valuation of about $16.40 per long ton, which is less than one-half the average selling price of the domestic product. The material cannot displace the domestic garnet to any great extent, as it is useful only for the finer sizes of abrasive papers. Garnet is a common ingredient of the sands found along the streams and lakes of the Adirondacks, but it seldom if ever forms deposits of any extent and is also always in finely comminuted condition.

GLASS.

The year 1908 was extremely unsatisfactory in the window-glass industry. At all times demand was abnormally light with the natural result that prices were cut. Even the prevailing low prices failed to make buyers anticipate their needs; large supplies, accumulated before the panic of October, 1907, being still on their hands, and, owing to the curtailment of building operations and general business retrenchment, there was no outlet for this surplus product.

Notwithstanding labor troubles and a reduction in the price of machine-made glass, 900 additional pots were started in the latter part of March and early part of April, making a total of 1500 working in the United States. By increasing their output these factories were able to continue in operation. In October, the failure of an attempt to organize the independent manufacturers into a selling agency to do away with injurious price cutting resulted in a further slump in prices, but toward the end of December the market began to show strength. This was due in part to a curtailment of output, brought about by the closing of some plants on account of strikes.

As outlined above there was a continual fight to keep the market price of glass up to such a figure as would allow its manufacture by the hand operating plants. The best general quotations were on the basis of 90 and 20 per cent. off single and double with the machine interests holding to the quotable basis of 90 and 25 per cent. off single and 90 and 30 per cent. off double. Recent investigations of the glass industry in Germany show that there are 374 works in operation, employing in all 62,159 persons. Bottle making is an important branch of the industry. Belgium produces a large quantity of glass of an inferior grade, which fetches only about half the price of the American product. On account of the low prices the Belgium glass competes to some extent with American glass.

GOLD AND SILVER.

The gold mines of the world in 1908 not only maintained the great production of recent years, but also increased it by a considerable amount. Australasia showed a small decrease, but in the United States, the Transvaal, Russia, Mexico, and Rhodesia there were large increases. The statistics are given in the accompanying table. For all of the principal countries and for many of those of less importance, the reports are official, even those for 1908. In the case of the United States, however, there is likely to be more or less revision, inasmuch as the figures used are the preliminary estimates by Frank A. Leach, director of the United

PRODUCTION OF GOLD IN THE UNITED STATES. (a)

States.	1905		1906		1907		1908	
	Fine Ounces.	Value.	Fine Ounces.	Value.	Fine Ounces.	Value	Fine Ounces.	Value.
Alabama	2,008	$41,500	1,137	$23,500	1,325	$27,400	2,111	$43,686
Alaska	722,090	14,925,600	1,033,537	21,365,100	894,424	18,489,400	1,012,549	20,930,784
Arizona	130,203	2,691,300	132,891	2,747,100	128,871	2,664,000	113,565	2,345,308
California	928,742	19,197,100	911,041	18,832,900	815,288	16,853,500	947,278	19,581,570
Colorado	1,243,401	25,701,100	1,109,452	22,934,400	1,010,921	20,897,600	1,103,530	22,811,784
Georgia	4,586	94,800	1,146	23,700	3,135	64,800	2,367	48,918
Idaho	52,085	1,075,600	50,102	1,035,700	60,754	1,255,900	70,185	1,450,830
Maryland	818	16,900	.....	.....	(b)	.....	(b)	.....
Montana	236,541	4,889,300	218,752	4,522,000	167,987	3,472,600	160,732	3,322,551
Nevada	259,269	5,359,100	448,852	9,278,600	745,507	15,411,000	584,872	12,090,218
New Mexico	12,859	265,800	12,877	266,200	15,964	330,000	11,646	240,756
N. Carolina	5,990	123,900	4,397	90,900	3,807	78,700	4,122	85,314
Oregon	60,227	1,244,900	63,860	1,320,100	59,124	1,222,200	54,158	1,119,528
S. Carolina	4,601	95,100	3,609	74,600	2,811	58,100	2,819	58,336
S. Dakota	334,490	6,913,900	319,512	6,604,900	200,185	4,138,200	372,026	7,690,294
Tennessee	160	3,300	39	800	184	3,800	154	3,186
Texas	92	1,900	164	3,400	48	1,000	26	545
Utah	248,713	5,140,900	248,208	5,130,900	247,758	5,121,600	190,132	3,930,290
Virginia	242	5,000	498	10,300	402	8,300	209	4,321
Washington	17,900	370,000	4,983	103,000	12,689	262,300	10,734	222,189
Wyoming	1,147	23,700	276	5,700	455	9,400	86	1,773
Other States	.....	.....	.....	.....	.....	.....	1,147	23,725
Total	4,266,120	$88,180,700	4,565,333	$94,373,800	4,371,639	$90,369,800	4,644,493	$96,005,906
Porto Rico	.....	.....	.....	.....	58	1,200	31	642
Philippine Islands	.....	.....	.....	.....	3,130	64,700	14,836	306,708
Total	4,266,120	$88,180,700	4,565,333	$94,373,800	4,374,827	$90,435,700	4,659,360	$96,313,256

(a) The statistics in this table are as reported by the Director of the Mint, those for 1908 being the preliminary figures (subject to revision). (b) Included in other States.

States Mint, based on the returns of the various branch mints and assay offices. His final figures were not ready at the time when this volume was put upon the press.

The production of silver in the world in 1908 showed an increase, but that of the United States was the smallest reported in nearly 20 years, being less than that of 1907 by 8.3 per cent. As usual, the production of

PRODUCTION OF SILVER IN THE UNITED STATES. (*a*)

States.	1905		1906		1907		1908	
	Fine Ounces.	Commercial Value.	Fine Ounces.	Commercial Value.	Fine Ounces.	Commercial Value.	Fine Ounces.	Commercial Value.
Alabama	300	$181	100	$67	(*c*)	.....	(*c*)	.....
Alaska	169,200	102,116	203,500	135,920	179,250	$118,300	201,988	$106,777
Arizona	2,005,700	1,672,592	2,969,200	1,983,158	2,903,050	1,916,000	3,046,137	1,610,309
California	1,082,000	653,009	1,517,500	1,013,553	1,590,000	1,049,400	1,823,588	964,019
Colorado	12,942,900	7,811,239	12,447,400	8,313,743	11,494,500	7,587,000	10,161,318	5,371,715
Georgia	900	543	300	200	(*c*)	.....	(*c*)	.....
Idaho	8,125,600	4,903,962	8,836,200	5,901,786	7,888,400	5,206,300	6,289,585	3,325,027
Michigan	253,000	152,690	186,100	124,298	331,350	218,700	232,184	122,742
Missouri	(*c*)	.....	(*c*)	.....	25,150	16,700	25,000	13,216
Montana	13,454,700	8,120,181	12,540,300	8,375,792	11,129,600	7,345,500	11,518,913	6,089,352
Nevada	5,863,500	3,538,740	5,207,600	3,478,208	8,250,450	5,465,100	9,322,828	4,928,422
N. Mexico	354,900	214,189	453,400	302,830	599,550	395,700	372,950	197,152
N. Carolina	13,200	7,966	24,700	16,497	25,150	16,600	16,090	8,505
Oregon	88,900	53,653	90,700	60,579	96,050	63,400	109,640	57,961
S. Dakota	179,000	108,030	155,200	103,658	106,600	70,400	197,996	104,668
Tennessee	95,400	57,576	25,600	17,098	58,350	38,500	59,876	31,653
Texas	417,200	251,788	277,400	185,278	305,300	201,500	461,715	244,081
Utah	10,319,800	6,228,205	11,506,000	7,686,298	11,406,800	7,528,500	7,718,872	4,080,505
Virginia	200	121	100	67	(*c*)	.....	(*c*)	.....
Washington	119,400	72,060	42,100	28,119	83,950	55,400	68,175	36,040
Wyoming	2,700	1,630	1,100	735	(*c*)	.....	(*c*)	.....
Other States	(*b*)13,000	7,846	31,300	20,906	10,000	6,600	167,495	88,539
Total	50,101,600	$33,858,438	56,517,900	$37,748,757	56,514,550	$37,299,600	51,794,350	$27,380,683
Porto Rico	.....	.....	.....	.....	.....	.....	3	1
Philippine Islands	.....	.....	.....	.....	150	99	3,700	1,965
Total	56,101,600	$33,858,438	56,517,900	$37,748,757	56,514,700	$37,299,699	51,798,053	$27,382,640

(*a*) The statistics in this table are reported by the Director of the Mint, those for 1908 being the preliminary figures (subject to revision). (*b*) Includes Maryland, 100 oz.; Missouri, 2,900 oz. (*c*) Included in other States.

TOTAL PRODUCTION OF GOLD AND SILVER IN THE UNITED STATES.

Years.	Gold.	Silver.	Years.	Gold.	Silver.	Years.	Gold.	Silver.
	Dollars.	*Ounces.*		*Dollars.*	*Ounces.*		*Dollars.*	*Ounces.*
1792-1834	14,000,000	*Nil*	1884.	30,800,000	37,800,000	1898.	64,463,000	54,438,000
1835-1844	7,500,000	193,365	1885.	31,800,000	39,910,000	1899.	71,053,000	54,764,000
1845-1854	343,036,769	386,730	1886.	35,000,000	39,685,513	1900.	79,171,000	57,647,000
1855-1864	479,300,000	20,806,518	1887.	33,000,000	41,721,592	1901.	78,666,700	55,214,000
1865-1874	454,950,000	154,390,609	1888.	33,175,000	45,792,682	1902.	80,000,000	55,500,000
1875	33,400,000	24,533,993	1889.	32,800,000	50,000,773	1903.	73,591,700	54,300,000
1876	39,900,000	30,010,054	1890.	32,845,000	54,516,300	1904.	80,723,200	57,786,100
1877	46,900,000	30,783,509	1891.	33,175,000	58,330,000	1905.	88,180,700	56,101,600
1878	51,200,000	34,960,000	1892.	33,000,000	64,900,000	1906.	94,373,800	56,517,900
1879	38,900,000	31,550,000	1893.	35,955,000	60,000,000	1907.	90,435,700	56,514,700
1880	36,000,000	30,320,000	1894.	39,500,000	49,600,000	1908.	96,313,256	51,798,053
1881	34,700,000	33,260,000	1895.	46,610,000	55,727,000			
1882	32,500,000	36,200,000	1896.	53,088,000	58,835,000	Total	3,067,369,825	1,782,130,575
1883	35,000,000	35,730,000	1897.	57,363,000	53,860,000			

Note.—To the end of 1872, the statistics are those of R. W. Raymond, United States Mining Commissioner; subsequent statistics are those reported by the Director of the Mint. 1908 figures provisional.

silver was affected by that of lead and copper, in connection with which metals a large part of the silver is obtained. Three of the important silver producers of the United States—Montana, Nevada and Arizona—

showed gains, only that of Nevada being large. The increases in Montana and Arizona were due to the greater output of copper carrying some silver. On the other hand there were heavy decreases in Colorado, Idaho and Utah, where the silver is obtained chiefly in connection with lead.

GOLD AND SILVER PRODUCTION OF THE WORLD, 1493-1850.
According to Dr. Adolph Soetbeer.

Period.	Estimated Production In Kilograms.		Ratio of Silver to Gold. Weight.	Ratio of Gold to Silver. Value.	Period.	Estimated Production In Kilograms.		Ratio of Silver to Gold. Weight.	Ratio of Gold to Silver. Value.
	Gold.	Silver.				Gold.	Silver.		
1493-1520	162,400	1,316,000	8.1	10.75	1701-1720	256,400	7,112,000	27.7	15.21
1521-1544	171,840	2,164,800	12.6	11.25	1721-1740	381,600	8,624,000	22.6	15.08
1545-1560	136,160	4,985,600	36.6	11.30	1741-1760	492,200	10,662,900	21.7	14.75
1561-1580	136,800	5,990,000	43.8	11.50	1761-1780	414,100	13,054,800	31.5	14.73
1581-1600	147,600	8,378,000	56.8	11.80	1781-1800	355,800	17,581,200	49.4	15.09
1601-1620	170,400	8,458,000	49.6	12.25	1801-1810	177,780	8,941,500	50.3	15.61
1621-1640	166,000	7,872,000	47.4	14.00	1811-1820	114,450	5,407,700	47.2	15.51
1641-1660	175,400	7,326,000	41.8	14.50	1821-1830	142,160	4,605,600	32.4	15.80
1661-1680	185,200	6,740,000	36.4	15.00	1831-1840	202,890	5,964,500	29.4	15.75
1681-1700	215,300	6,838,000	31.8	14.97	1841-1850	547,590	7,804,150	14.3	15.83

GOLD PRODUCTION OF THE WORLD, 1851-1908.

Year.	Value.	Year.	Value.	Year.	Value.	Year.	Value.
1851.....	$67,600,000	1866.....	$121,000,000	1881.....	$103,102,000	1896.....	$211,242,081
1852.....	132,800,000	1867.....	104,000,000	1882.....	102,000,000	1897.....	237,833,984
1853.....	155,500,000	1868.....	109,700,000	1883.....	95,400,000	1898.....	287,327,833
1854.....	127,500,000	1869.....	106,200,000	1884.....	101,700,000	1899.....	311,505,947
1855.....	135,100,000	1870.....	106,900,000	1885.....	108,400,000	1900.....	258,829,703
1856.....	147,600,000	1871.....	107,000,000	1886.....	106,000,000	1901.....	260,877,429
1857.....	133,300,000	1872.....	99,600,000	1887.....	105,775,000	1902.....	298,812,493
1858.....	124,700,000	1873.....	96,200,000	1888.....	110,197,000	1903.....	329,475,401
1859.....	124,900,000	1874.....	90,800,000	1889.....	123,489,000	1904.....	349,068,293
1860.....	119,300,000	1875.....	97,500,000	1890.....	118,848,700	1905.....	378,411,754
1861.....	113,800,000	1876.....	103,700,000	1891.....	130,650,000	1906.....	405,551,022
1862.....	107,800,000	1877.....	114,000,000	1892.....	146,292,600	1907.....	411,294,458
1863.....	107,000,000	1878.....	119,000,000	1893.....	158,437,551	1908.....	444,382,312
1864.....	113,000,000	1879.....	109,000,000	1894.....	182,509,283	.....	.....
1865.....	120,200,000	1880.....	106,600,000	1895.....	198,995,741	.....	.....

SILVER PRODUCTION OF THE WORLD, 1851-1908.

Year.	Kilograms.	Year.	Kilograms.	Year.	Kilograms.	Year.	Kilograms.
1851-1855..	4,430,575	1881.....	2,592,639	1891.....	4,479,649	1901.....	5,438,443
1856-1860..	4,534,950	1882.....	2,769,065	1892.....	4,985,855	1902.....	5,121,469
1861-1865..	5,505,575	1883.....	2,746,123	1893.....	5,339,746	1903.....	5,386,044
1866-1870..	6,695,425	1884.....	2,788,727	1894.....	5,205,065	1904.....	5,669,124
1871-1875..	9,847,125	1885.....	2,993,805	1895.....	5,667,691	1905.....	5,638,183
1876.....	2,323,729	1886.....	2,902,471	1896.....	5,496,178	1906.....	5,683,947
1877.....	2,388,612	1887.....	2,990,398	1897.....	5,663,304	1907.....	5,704,083
1878.....	2,551,364	1888.....	3,385,606	1898.....	5,575,336	1908.....	6,241,225
1879.....	2,507,507	1889.....	3,901,809	1899.....	5,529,024	.....	.....
1880.....	2,499,998	1890.....	4,180,532	1900.....	5,599,216	.....	.....

GOLD AND SILVER MINING IN THE UNITED STATES.

A large part of the gold and silver produced in the United States, as well as elsewhere in the world, is derived from the ores of copper and lead. It has been the custom in THE MINERAL INDUSTRY to review such

districts under the caption of the principal base metal, which custom is conformed to in this volume. The following review relates only to the mines of gold and silver proper, i.e., those in which the precious metals are the chief element of value.

GOLD PRODUCTION OF THE WORLD.

Countries.	1906			1907			1908		
	Oz. Fine.	Kilo-grams.	Value.	Oz. Fine.	Kilo-grams.	Value.	Oz. Fine.	Kilo-grams.	Value.
America, North:									
United States	(a)4,565,333	142,001	$94,373,800	(a)4,374,827	136,075	$90,435,700	(e)4,659,360	144,933	$96,313,256
Canada	(b)581,709	18,093	12,023,932	(b)399,844	12,437	8,264,765	(b)462,467	14,384	9,559,274
Newfoundland	4,475	139	92,500	4,315	134	89,191	(e) 4,400	137	90,948
Mexico	896,615	27,889	18,534,700	(a)903,672	28,108	18,739,181	(a)1,182,445	36,779	24,518,548
Central America	(b) 92,431	2,875	1,910,725	(b)101,980	3,172	2,096,911	(e)110,000	3,422	2,273,875
America, South:									
Argentina	(b) 257	8	5,317	(b) 4,983	155	103,013	(e) 5,000	155	413,428
Bolivia	890	28	17,403	1,209	38	25,000	(e) 1,300	40	26,873
Brazil	(b)116,254	3,616	2,403,194	146,218	4,548	3,022,326	(e)150,000	4,670	3,100,650
Chile	(a) 36,490	1,135	754,321	(a) 61,310	1,907	1,267,278	(e) 60,000	1,871	1,240,200
Columbia	(b)105,966	3,296	2,190,522	(b)157,471	4,898	3,255,311	(e)170,000	5,289	3,514,073
Ecuador	(b) 14,242	443	294,417	(b) 12,924	402	267,169	(e) 12,000	374	248,052
Guiana, British	79,682	2,478	1,647,031	59,796	1,859	1,234,988	62,406	1,941	1,289,948
Guiana, Dutch	38,162	1,187	788,820	35,494	1,104	733,718	38,790	1,210	810,829
Guiana, French	108,924	3,388	1,765,312	(b) 89,923	2,797	1,858,886	(e) 90,000	2,800	1,860,300
Peru	(a) 40,091	1,247	828,756	(a) 24,981	777	516,394	(e) 25,000	778	516,500
Uruguay	(b) 1,542	48	31,901	(b) 2,508	78	51,839	(e) 2,500	78	51,650
Venezuela	(b) 1,222	38	25,255	(b) 1,093	34	22,596	(e) 1,100	34	22,737
Europe:									
Austria	(a) 4,035	125	83,401	(a) 4,437	138	91,719	(e) 4,400	137	90,948
France	(a) 24,754	769	511,665	(a) 40,991	1,275	847,290	(e) 35,000	1,088	723,450
Hungary	(a)120,176	3,738	2,484,275	(a)112,557	3,501	2,326,765	(e)110,000	3,422	2,273,815
Germany	3,890	121	80,400	3,220	100	66,460	3,220	100	66,460
Italy	(b) 1,990	62	41,205	(b) 1,928	60	39,876	(e) 1,900	59	39,273
Norway	(a) 261	8	5,400	.....	.....	.....	(e) 250	8	5,165
Portugal	48	1	1,000	48	1	1,000	(e) 40	1	827
Russia	1,087,056	33,812	22,469,432	1,282,635	39,895	26,512,065	1,497,076	46,565	30,944,561
Spain	226	8	5,500	322	10	6,645	(e) 300	9	6,201
Sweden	(a) 643	20	13,292	(a) 900	28	18,590	(e) 900	28	18,603
Turkey	(b) 289	9	5,981	(b) 225	7	4,652	(e) 250	8	5,165
United Kingdom	(b) 1,415	44	29,242	(b) 1,415	44	29,242	(e) 1,400	43	28,938
Africa:									
Madagascar	56,585	1,760	1,169,608	54,012	1,680	1,116,428	(e) 55,000	1,712	1,136,850
Rhodesia	(a)457,115	14,218	9,647,581	(a)512,791	15,959	10,589,385	(a)594,407	18,488	12,276,394
Transvaal	5,786,617	179,988	119,609,373	6,431,384	200,041	132,936,707	7,054,621	219,429	145,819,016
West Coast	199,432	6,203	4,122,260	272,277	8,468	5,627,970	279,320	8,688	5,773,544
Asia:									
Borneo, British	(b) 70,087	2,180	1,448,828	(b) 75,520	2,349	1,561,145	(e) 80,000	2,488	1,653,600
China	(e)217,688	6,771	4,500,000	(e)217,688	6,771	4,500,000	(e)250,000	7,778	5,165,000
East Indies, Dutch	(b) 60,699	1,888	1,254,765	(b) 79,636	2,477	1,646,214	108,641	3,379	2,245,609
India, British	(a)525,527	16,346	10,852,546	(a)495,965	15,426	10,251,494	504,309	15,686	10,424,067
Japan (d)	132,979	4,135	2,748,920	164,753	5,090	3,403,378	(e)155,000	4,824	3,203,850
Korea	(b)110,434	3,435	2,282,901	(b)105,002	3,266	2,170,584	108,502	3,385	2,250,000
Malay Peninsula	16,933	527	350,000	15,627	485	325,000	(e) 15,000	467	320,385
Australasia (c)	3,984,538	123,927	82,358,207	3,659,693	114,132	75,849,349	3,557,705	110,632	72,509,200
Other Countries	(e) 72,570	2,257	1,500,000	72,570	2,257	1,500,000	(e) 75,000	2,332	1,550,250
Total	19,620,272	610,261	$405551022	19,988,144	618,898	$411,294,458	21,529,300	669,651	$444,382,312

(a) Official statistics of the country. (b) U. S. Mint report. (c) Six states and New Zealand. (d) Exclusive of Formosa. (e) Estimated.

Note—Fine gold is valued at $20.67 per oz.; which is equivalent to $664.60 per kilogram.

Alaska.—Twelve gold and silver lode mines produced in 1908, as compared to 13 in 1907. The extraction of ore was about 1,700,000 tons, averaging about $2.30 per ton, as compared to 1,300,000 tons in 1907. Fewer placer mines were operated in 1908 than in 1907. Their output was in the neighborhood of $15,000,000 as compared to $16,491,000 in

1907. The decrease was due to shortage of water supply; also to the approaching exhaustion of some of the bonanzas. However, the introduction of gold dredges is likely to play an important part in mining in Alaska in the future. In 1908 nearly 20 dredges were successfully operated.

The gold production of southeastern Alaska in 1908 was about $3,500,000, nearly all from lode mining. At the Treadwell mine develop-

SILVER PRODUCTION OF THE WORLD.

Country.	1907			1908		
	Oz. Fine.	Kilograms.	Value.	Oz. Fine.	Kilograms.	Value.
North America:						
United States	(b)56,514,700	1,757,944	$37,299,699	51,798,053	1,611,137	$27,382,640
Canada	(a)12,779,799	397,500	8,348,659	(d)22,070,212	686,476	11,667,197
Mexico	(a)61,147,178	1,901,934	41,522,453	(a)72,596,772	2,258,080	39,949,126
Central America	(b)1,892,896	58,867	1,249,300	(e)1,929,000	60,000	1,018,512
South America:						
Argentina	(b)25,178	783	16,600	(e)48,225	1,500	25,563
Bolivia	(e)6,750,000	209,974	4,409,573	(e)6,912,250	215,000	3,649,668
Chile	909,202	28,280	593,709	964,500	30,000	509,256
Colombia	(b)1,048,719	32,609	692,200	(e)1,028,800	32,000	543,206
Ecuador	(b)2,456	76	1,600	(e)3,215	100	1,697
Peru	(a)6,681,091	207,810	4,362,770	(e)7,233,750	225,000	3,819,420
Uruguay	(e)1,000	31	653	(e)1,000	31	528
Europe:						
Austria	(a)1,245,555	38,742	813,347	(e)1,350,300	42,000	712,958
Hungary	(a)407,019	12,660	265,785	(e)401,875	12,500	212,190
France	(b)719,453	22,358	474,800	(e)707,300	22,000	373,454
Germany	5,088,086	158,261	3,358,100	(e)5,200,000	161,968	2,745,600
Greece	(b)829,025	25,786	547,200	(e)803,750	25,000	424,380
Italy	(a)659,139	20,502	430,418	(e)643,000	20,000	339,504
Norway	(a)215,405	6,700	140,659	(e)218,620	6,800	115,431
Russia	(b)132,122	4,110	87,200	(e)135,030	4,200	71,296
Spain	(b)4,097,035	127,415	2,704,000	(e)4,099,125	127,500	2,164,434
Sweden	(a)29,761	929	19,600	(e)32,150	1,000	16,975
Turkey	522,438	16,250	341,293	(e)546,550	17,000	288,578
United Kingdom	138,245	4,300	90,311	(e)128,600	4,000	67,901
Asia:						
Dutch East Indies	190,000	5,900	124,121	500,000	15,521	264,000
Japan	2,856,560	88,851	1,865,334	(e)2,893,500	90,000	1,527,768
Australasia	17,516,433	544,803	11,442,960	(f)17,308,281	538,396	9,034,965
Africa	(b)790,431	24,586	521,700	(e)803,750	25,000	424,380
Rhodesia	147,324	4,582	96,203	238,425	7,416	125,888
Other Countries	50,000	1,555	32,663	(e)51,440	1,600	27,160
Total	183,386,250	5,704,098	$119,751,221	200,655,383	6,241,225	$107,503,675

(a) Official statistics. (b) U. S. Mint figures. (d) Preliminary official figures. (e)Estimated. (f) In part estimated.

Note:—Value of silver for 1907 is reckoned as $0.653 per oz. and as $0.528 for 1908, unless value is given by country officially reporting.

ments were made on the 1450-ft. level, and at the Ready Bullion mine on the 1500-ft. level. The stamp mills on Douglas island were operated steadily in 1908. The use of oil instead of coal at these mines has reduced expenses materially. At the Perseverance mine the 100-stamp mill was operated during the second half of the year. Encouraging results were reported from the Juneau gold belt.

The gold production of the Alaska-Yukon was about $9,500,000, or about the same as in 1907. Most of the output came from the Fairbanks

district, which reported upward of $8,000,000. The discovery of valuable lodes caused excitement in the summer. Many of the reports were greatly exaggerated, but some of the finds are probably worth prospecting.

(By John Power Hutchins.)—The gold production of the Seward peninsula, of which Nome is the headquarters, was about $5,000,000 in 1908, against $6,500,000 in 1907. The output in previous years was as follows: 1900, $4,000,000; 1901, $4,500,000; 1902, $5,050,000; 1903, $4,437,000; 1904, $4,878,000; 1905, $4,600,000; 1906, $7,000,000. The summer of 1908 was very dry, which caused the reduction in output. The freeze-up came about Sept 20 and was earlier than usual. Discoveries of importance were made on the old beach lines, back of Nome, and it is probable that these will continue for a number of years. Considerable gravel containing as much as $100 per cubic yard was extracted. There were no labor difficulties in 1908, and the supply of men was ample at all times. Wages were 40c. per hour, plus board, and 30c. per hour, plus board, for skilled and unskilled labor respectively. The production of gold by drift mining was proportionately larger than in normal years, this kind of mining being not seriously affected by droughts. Probably more than two-thirds of the gold in 1908 was produced by this method, which is applied almost exclusively for mining the old beach lines back of Nome. These old beaches are interesting, both economically and geologically. They are probably areas where gold-bearing material transported by streams were concentrated by wave action, the region being successively elevated. It is believed locally that the old beaches and the intervening areas will be worked by floating dredges. Much of the ground is frozen, and as yet it is uncertain whether it can be thawed cheaply enough to permit of profitable extraction in that way.

In 1909 gold dredging was carried on in the vicinity of Nome about as usual. A new dredge with 9-cu.ft. buckets began operations; after a series of accidents it sank and is now a wreck, the design, construction and management, combining with difficult conditions to produce this result. Similar experience has attended almost every other dredge that has been installed on the Seward peninsula. Hydraulic mining suffered most in 1908 because of the dry season. There was not much open-cut mining with hand labor.

Arizona.—The production of gold decreased a little in 1908. The production of silver was about the same as in 1907. The market for smelting ores was reduced by the idleness of the Humboldt smelter, but on the other hand, the inauguration of smelting at Sasco increased the capacity in the southern portion of the Territory. The industrial depres-

sion had generally a grave effect, all but the well established mining enterprises being closed down.

(By William P. Blake.)—A new producer in 1908 was the Mohawk mine in the Old Hat district, near Schultz, Pinal county, which previous to 1907 had been idle for 10 years or more. At the contact between granite and an eruptive rock on the 500-ft. level new bodies of free milling ore have been opened, and a 30-stamp mill has been supplied. The ore contains a noteworthy percentage of wulfenite, but insufficient to pay for its extraction by concentration. The Fortuna mine, in the extreme southwest of the Territory, has ceased to be an important producer. In the Quijotoa district, the Weldon Gold and Copper Company continued operations, increased the capacity of its mill and added a cyanide plant. The Congress and other gold mines of the Piedmont district of the Bradshaw mountains were operated. The Octave mill was started late in 1908. Near Prescott, the Bullwhacker mine was worked, its ore being treated in the Oriental mill, at Poland. The Arizona Copper-Gold Mines prosecuted development work up to Sept. 20, making ready for the installation of a mill in 1909. In northern Pinal county development work on the gold-bearing veins in the Saddle mountain district, south of the Gila river, was resumed in the latter part of 1908, particularly upon the property of the Ash Creek Mining Company. In this property the vein may be said to be double, being formed on both sides of an intrusive dike. A cross-cut shows the south vein to be 20 in. wide, assaying 5 to 20 per cent. copper, and $20 to 70 in gold; and the north vein about 4 ft. wide assaying 4 per cent. copper. In the Bunker Hill district, east of Mammoth, prospecting gave satisfactory results and a 200-ton concentrating mill is to be erected there. The district within which there are good evidences of copper and gold ore is described as about five miles wide and 12 miles long. Near Kingman some promising orebodies were developed. A large body of low-grade auriferous copper ore was opened in the Signal mine in northern Yuma county. The King of Arizona mine at Kofa was milling about 200 tons of ore per day at the end of 1908. A new mine in the same district known as the North Star has been rapidly developed and is full of promise. The vein is described as occurring in brecciated andesite. The Desert mines, two miles north of Vicksburg, were reopened and the mill rebuilt. At Salome, Yuma county, north of the Harqua Hala mountains, there was increased activity. At the Harqua Hala mines the efforts to work the fillings of the old stopes were abandoned and attention is now being given to unwatering and sinking for new orebodies below the old workings.

Most of the silver production of Arizona is derived from the copper mines. The Territory was formerly celebrated as the source of many

ponderous masses of nearly pure silver from the gravelly placer deposits near Globe. In 1908 the old Stonewall Jackson mine, supposed to be the source of the placer silver, was reopened; at the depth of 280 ft. a rich vein of silver glance is reported to have been found. Efforts were made to resuscitate the Silver King mine in Pinal county, but up to the end of 1908 active work had not been resumed.

California.—There were no specially important developments in the gold-mining industry of this State in 1908. Numerous changes were made in ownership of mines; some new ones were opened and some old ones abandoned. The reopening of long-abandoned properties continued in the principal counties. Many of the old mines of the State were given up years ago because, under the circumstances then existing, they did not pay; but under modern conditions they can be made profitable when sufficient capital is invested for proper development and suitable machinery. In general California did better in 1908 than in 1907 because there were no general labor strikes and no floods, although the dryness in the fall affected the yield to some extent.

In the central and northern portions of the State there were no developments of note aside from a few instances. In the southern portion new railroads brought new sections into notice and gave prospectors a chance to enter regions heretofore neglected, especially in Inyo and San Bernardino counties, where numerous discoveries were made and many new mines were opened. In the desert regions, particularly those bordering on Nevada, there was more or less activity due to the fact that, of late, transportation facilities have been provided and prospects, hitherto worthless, have become of value because ores can now be shipped to reduction works. In the Mother Lode section and in other older portions there were few changes. Development work continued as usual. In Siskiyou and Trinity counties there was much activity in gravel mining and some new quartz enterprises were begun.

The increased output of gold for California for 1908 came mainly from dredging. This industry is fast increasing in output. The machines recently built are far more powerful and of greater capacity than those first in use, are more cheaply operated and are better gold savers. The use of double banks of gold-saving tables on the larger machines gives much more amalgamating surface than was possible by the old plan. The adoption of hydraulic nozzles throwing water under pressure on the hard ground in certain localities, breaking it down into the buckets, relieves the dredges of much work, and increases their capacity for digging. The rotary screens are also excellent, especially where the gravel is hard to wash. Twelve new dredges were set in operation in 1908 and about a dozen more are being built or are planned for. There

are now 69 gold-mining dredges in operation in California. Of these 37 are in Butte county; two in Calaveras county; one each in the counties of Merced, Stanislaus, Shasta, Placer and Siskiyou; 10 in Sacramento, and 15 in Yuba county. The principal fields continue to be at Oroville, Butte county; Marysville, Yuba county and Folsom, Sacramento county, where there are large tracts of available dredging ground at points where the Feather, Yuba and American rivers cease to be torrential streams and widen out as they debouch from the mountains or foothills into the valley sections.

At such points, though the gold is fine, it is evenly distributed through the gravel and the average per cubic yard is subject to little change. At other places in the State small areas of ground available for dredging have been found in isolated spots, generally but a few hundred acres in extent. At such places one or two dredges can be profitably installed. Along the Klamath river in Trinity and Siskiyou counties several dredges are being built, but not more than one at any particular place, the area where dredging may be carried on being usually small.

The most profitable dredging appears to be carried on near to, but away from the rivers themselves; but little is being done in the river beds proper. Those set at work directly in the river beds in most of the counties of the State are liable to be enjoined unless they are "land-locked," that is, throw a rock wall around the pits where operations are carried on, so that the material disturbed in the digging may be confined and not allowed to pass into the running stream. Where this latter occurs, there is liability of damage to the lands adjacent to the streams, and litigation is apt to ensue. The dredge-men perfectly understand this feature and are usually careful to avoid any chance of harming other interests and involving themselves in lawsuits. Moreover, when dredges are operated in the main rivers there is danger of wreck and damage to the machines in case of floods. This was demonstrated at Oroville in 1907, for the spring floods in the Feather river destroyed several dredges and damaged several others. Some of the large new dredges are now mining in soft ground at a cost of less than 2c. per cubic yard for operating expenses.

Colorado. (By George E. Collins.)—Mining in Colorado during 1908 was on the whole in a distinctly depressed condition. With the exception of Cripple Creek and Telluride, both of which are essentially gold-producing camps, every district in the State was materially affected not only by the low metal prices, but also by the fact that the financial conditions in the Eastern States necessitated the postponement of new outlay both for equipment and for prospecting. The ore received at the various plants of the American Smelting and Refining Company, which

may be taken as a fair index of the condition of the mining industry, outside of that from purely gold-producing mines, was about 550,000 tons as compared with 796,000 tons in 1907. Owing to the lower prices received for the product, the falling-off in value was still more serious.

As to the Cripple Creek district, a reduction in the cost of treatment at the custom chlorination plants was in force during the greater part of 1908, and resulted in a largely increased tonnage. This increase, however, was mainly in shipments of very low-grade material, so that the total output, while somewhat greater than for 1907, was less than has been generally supposed. The minimum price for freight and treatment was $3.50 per ton; this rate enabled the shipment of great quantities of dump material.[1] The shipments of ore of the better grades were, however, smaller than during 1907, and the average value of the output was reduced. The principal producing properties were the Portland, Elkton, Golden Cycle, Strong and Cresson; an important output was maintained by leasers working in the Vindicator and El Paso. Of the outside leases, the most profitable and productive was probably that on the Lucky Gus mine; the Dante leases hardly came up to what was expected. The drainage tunnel made good progress, the face at the end of 1908 being about 4500 ft. from the portal. The footage reported during the last few months of the year was particularly gratifying. The new mill of Stratton's Independence was completed early in 1908, but owing to a variety of causes, principally financial, it did not go into steady operation. It is understood that these difficulties have now been straightened out; so that, unless a serious error has been made in sampling the dump, which it was designed primarily to treat, the mill should add largely to the production next year. The future of Cripple Creek depends much upon the cheap treatment of the low-grade ores. Discoveries of high-grade orebodies seem to be fewer in number each year, and it is improbable that the existing rate of production can be maintained much longer from ore of the present grade; still, the completion of the drainage tunnel and the introduction of cheaper milling methods will doubtless result in maintaining Cripple Creek as the chief gold-producing district of Colorado for many years to come.

The Ouray district was comparatively inactive, with the exception of the great Camp Bird mine, which maintained a large production throughout 1908. It is, however, generally understood that the orebodies in that mine are approaching exhaustion. At Telluride the Tom Boy, the Smuggler-Union and the Liberty Bell maintained the same output as

[1] Immediately after the Golden Cycle mill went into active operation, the Mill Trust established an open rate, effective also on all existing contracts, under which the treatment was reduced $1@2 per ton, and $3.50 was named for ores below $5 in value. On Oct. 15 the old rates were restored, but on Dec. 26 a revised schedule was published as follows: On ore up to 0.5 oz., freight and treatment, $4.50—old rate, $5.25; to 0.75 oz., $5.25—old, $6.50; 1 oz., $6—old, $6.75 per ton.

for some years past, but the Smuggler-Union was considerably affected by the low price of silver. At Silverton, the Gold King was shut down for several months, on account of a disastrous fire. The Silver Lake was compelled to curtail its production on account of the low values of lead, silver and copper, and probably also the partial exhaustion of the best orebodies, and at present the mine is being worked only by lessees.

Leadville and Aspen were both seriously affected by the drop in silver. Previous to that the Crescentia and Reindeer shipped largely. A considerable production of gold-bearing material was made from the territory contiguous to the Ibex properties. The Dinero tunnel was completed to the vein during the fall, and development work is now in progress. At Aspen the production, as to quantity, was not much less than in 1907. The Della S. and Durant shipped a large tonnage; in the latter case low-grade lead-silver ore, that was mined by lessees. The Percy LaSalle was shut down, but the Smuggler mill ran continuously throughout the year. At Creede also the tonnage handled was about the same as in 1907, but here again the low prices of metals must have seriously curtailed the profit margin.

From Gilpin county the output was perhaps a little better than in 1907, but as a whole the district was in a depressed condition. The opening-up of important orebodies in the Bobtail vein on the lowest level promises an increased production in 1909, while the resumption of work in the Newhouse tunnel is certain to lead to a revival in the district around Nevadaville. The Georgetown district was hard hit, on account of the low metal prices. Although the high-grade silver ores cannot fairly be described as approaching exhaustion, it is unfortunately true that the discoveries made of late years have been of small extent. The amount of development work done, however, is altogether inadequate, and is likely to remain so unless capital is invested from outside. An important orebody, the value in which is largely gold, has been opened in the Capital tunnel, and a mill has been built at the entrance of the tunnel to handle this ore. At Idaho Springs, the Stanley and Lamartine mines produced very little ore. The Gem and Little Mattie mines, both operated on the leasing system, were probably the largest contributors to the output, together with the Specie Payment, worked through the Two Brothers tunnel. The Sun and Moon mine, on the boundary line between Gilpin and Clear Creek counties, has been opened up with a raise, from the level of the Newhouse tunnel, that connects with the lowest workings from surface. Several levels from this raise have been commenced in ore, and probably in 1909 this mine will be the largest producer in the district. The resumption of work in the heading of the Newhouse tunnel,

which took place on Dec. 16, is likely to prove of great permanent importance to this region.

The well-known Alice orebody, at the head of Fall river, is again being tested; the result of several careful mill-runs indicates that it is susceptible of profitable working on a large scale. There is very little of importance to report with reference to precious-metal developments in Boulder and Summit counties. The large dredging operations planned for the Breckenridge district have been delayed, as a result of the financial stringency in the East.

Idaho. (By Robert N. Bell.)—The Golden Sunbeam mine in the Yankee Fork district, Custer county, operated its 50-ton mill throughout 1908 and made a large output of gold bullion. Developments in the mine were satisfactory. At Indian creek, Lemhi county, the Ulysses mine resumed operation and made an important yield. In the Silver City district, Owyhee county, about 600 men were employed. The Delamar mine increased its ore reserves and added to its milling capacity. Operations at the Trade Dollar were somewhat interrupted by insufficient electric power. The most successful new enterprise near Silver City is the Potosi mine, now developed to depth of 300 ft.; a 20-ton mill was built in 1908 and the mine promises to become a dividend-payer. The Pettit mine (Bagdad-Chase Gold Mining Company), Atlanta district, Elmore county, supplied a 20-stamp mill in 1908; an addition of 20 stamps is being made, which will give a total milling capacity of 150 tons of ore per day. The new mill of the Atlanta mines company proved a failure and will have to be remodeled. The orebodies of this district are large, but proper methods of milling must be employed in order to extract a good proportion of their silver. The Elk City district, Idaho county, furnished employment to a good many men. The Buster mine has been developed to depth of 400 ft. on a vein 10 ft. wide, which supplies a 10-stamp cyanide mill. In the Elk City district there are many promising fissure veins, so that further work is likely to develop other producers. The district also contains some large deposits of low-grade gold ore that are likely to become mines of the Treadwell type, with proper development and equipment. The district is also noted for extensive old-channel, placer deposits; some of the placers are favorable for dredging.

Montana. (By H. K. Welch.)—From a mining standpoint in Montana, gold and silver are secondary to the copper industry. There are, however, several districts which operated continuously for gold in 1908. In the Garnet district in Granite county the Mitchell and Mussigbroad properties made several rich shipments. The Anderson and Magone properties in the same district were turned over to a new corporation,

the Anderson & Magone Mining Company, which will carry on the operations. In the Cable district, in Deer Lodge county, the Cable Consolidated Company operated practically all the year with a small force of men. The operations were confined principally to development work. At the Southern Cross mine operations were continued throughout the year and consisted principally of development work. A new mill was erected. The Gold Coin mill and mine ran for the last few months. At the properties of the Milwaukee Gold Extraction Company the work was uninterrupted. In the Radersburg district in Broadwater county the Keating mine made steady shipments to the Pittsmont and Washoe smelteries. In the Granite district in Granite county the Granite Bi-Metallic Company operated in a small way and several shipments were made to East Helena. In the Ruby district, in Madison county, a new gold dredge was built, making three in all, and operations were carried on all the year. In Silver Bow county the British-Butte company finished the construction of a Risdon gold dredge, but the machinery has not as yet been put in operation.

Nevada. (By William H. Shockley.)—The production of this State in 1908 was less than was anticipated, the shortage being due in part to the fall in the price of silver and in part to the fact that the Goldfield Consolidated husbanded its ore. This company produced about $700,000 in 1908 and should make a large production in 1909 with its new 100-stamp mill. I have learned of the following dividends paid by Nevada mines in 1908: Goldfield: Engineer's Lease, $555,000; Florence Annex, $50,000; Florence Mining Co., $420,000; Little Florence Lease, $750,000; Mohawk-Jumbo, $200,000; Red Top Consolidated, $20,000; Rogers Syndicate Lease, $215,00. Comstock: Ophir Mining Co., $32,400. Round Mountain: Round Mountain Mining Co., $64,000. Tonopah: Tonopah Mining Co., $500,000. Total, $2,806,400. Other dividends were probably paid about which I have no information.

At the end of 1907 a bitter strike was in progress in Goldfield and nothing but the presence of Federal troops prevented mob rule. In January, 1908, under the protection of these troops, non-union miners began work in Goldfield. The strike was called off on April 3, and Goldfield is now an open and peaceful camp. The establishment of a State police at a special session of the legislature in January preserved the peace of the State. This action of the legislature was approved by the voters at the election of Nov. 3, 1908, and this vote makes it certain that future strikes will be conducted peaceably. Though wages were reduced to the Tonopah scale—$4 per diem for miners—in a number of the camps there were no strikes during 1908.

"High Grading," though not so prominent a business as in former years was still carried on. The Western Pacific Railway steadily constructed its line and in a year should have trains running across Nevada into San Francisco. The Bullfrog & Goldfield road was sold to the Tonopah & Tidewater in the spring of 1908. Many new railroads are projected, but no work was done on them in 1908. A branch line to the copper mines at Yerington is likely to be the first one built. Important reductions were made about the end of 1908 by the Clark road in order to attract freight to the Jerome smelter, and by the Tonopah & Tidewater road in favor of the Needles smelter. The change made by the latter was from $6 (old rate) to $3.50 on $20 ore, and from $12 to $9 on $100 ore.

The tendency in all the Nevada gold and silver mills is toward fine grinding. The new mill of the Goldfield Consolidated employs this process; a novelty in this mill is the treatment of the concentrate by sulphuric acid followed by cyaniding. The Greenawalt chlorination process is used in the mill of the Goldfield Chlorination Company. In southern Nevada, the demand for electric power has steadily increased; 30,000 h.p. is now being developed at Bishop's Creek, 113 miles west of Goldfield. Dry washers of two kinds were tried at Manhattan and Round Mountain, but proved failures. Air hammer drills, especially good in raises, are now extensively employed and give satisfaction. Shattered rock, which cementing failed to solidify, defeated an attempt to prospect below the Red Plume shaft of the Tonopah Mining Company with a calyx drill. Hydraulic mining was carried on at Round Mountain. At Rabbit Hole, near Lovelock, high pressure pumps are to be used for hydraulicking.

The yield of Esmeralda county was increased in 1908 by the 100-stamp mill of the Pittsburg Silver Peak Mining Company at Blair; this mill produced $750,000 during the 10 months it ran, and should turn out more than $900,000 in 1909. Goldfield's output was $7,500,000 from 75,000 tons of ore. In 1909 Goldfield's product should be even larger, since the 100-stamp mill of the Goldfield Consolidated is expected to treat 600 tons daily and ship $6,000,000 in the year. With the completion of this mill Goldfield has a daily milling capacity of over 1000 tons.

A well-traveled road does not seem a promising place for prospecting, but several roadside mines have been found in Nevada during the last few years. The croppings of the first mine found at Manhattan had been worn by wagon wheels for more than 40 years. In 1908 the Lucky Boy mine near Hawthorne, and the Great Western south of Goldfield, both found close to old wagon roads, came into prominence through their rich ores. The Great Western is shipping its silver ore to the Belmont mill

at Millers. A number of mines have been found near it; one of these, the Nevada Empress, has built a 35-ton Hathaway mill.

The Rawhide boom was faint early in 1908. A new rush began later, but was interrupted by the burning of the business portion of the town on Sept. 4. More than 50 hoists are working in Rawhide, and it will doubtless prove a good low-grade camp. The Holmes mine at Candelaria has lately been sold at sheriff's sale. A tramway has been built and ore shipments begun from the Mt. Potosi mine in this old camp.

In 1908 the Tonopah mines, in Nye county, produced about $2,000,000 in gold and $4,000,000 in silver. These mines have been seriously hampered by the fall in the price of silver; with silver at less than 50c. per oz., the Tonopah Mining Company is probably the only one that can make a substantial profit. The 100-stamp mill of the Tonopah Mining Company, at Millers, and the 40-stamp mill of the Montana Tonopah, at Tonopah, are both working ore for $3.50 per ton. The Montana, however, is the better mill and has proved that fine grinding is superior to separate treatment of sands and slimes for the Tonopah ore.

Round Mountain has five small mills and one of the few hydraulic mines of Nevada. This is a prosperous camp. The Round Mountain Mining Company paid two dividends during 1908, and reported a profit of $55,000 from working 14,000 tons of ore for the six months ending June 30, 1908. In spite of its facilities for milling 160 tons of ore daily —present charges $5 per ton—Manhattan made very little profit from its mines in 1908. This town will soon have electric power. The town of Helena, 40 miles east of Tonopah, was started in November, 1908, on the strength of the Clifford mines, from which $300-ore was shipped. The Berlin mines formerly owned by J. Phelps Stokes, of New York, were bought by Goldfield operators. The mines in the southern part of Nye county proved disappointing. The Montgomery-Shoshone, heralded as the wonder of the world in 1906, is comparatively a failure; 17,798 tons of ore worked from January to June, 1908, yielding $412,231, gave but little profit.

The mines of Wonder, in Churchill county, shipped a little ore in 1908. The Nevada Hills mine at Fairview was not very prosperous; its ore was cut off by a fault. Gold Circle, in Elko county, is improving and is to have a 40-ton mill. In Humboldt county the new camp of Jessup was prominently mentioned by the newspapers. In the same county the old Hendra mine at Dun Glen was resuscitated by E. S. Chafey, after whom the new town of Chafey is named. He found a body of good ore from which he was able to pay $30,000 for the mine by his ore shipments. These shipments stopped in November; and it is said that the mine is not looking so well. Seven Troughs, present population 1000, is likely

to last for some years; for, though the fabulous riches reported a year ago have not materialized, much good ore has been developed. Twenty hoists are now running in this district, and the Mazuma Hills and the Kindergarten have each built a 10-stamp mill. Rosebud, from which much was expected, has proved a "dead one." Lincoln county is one of the large producers. The Bamberger De Lamar ran steadily in 1908, crushing 130,000 tons of $4.75 ore. There is much activity in Searchlight; here the Quartette has just finished a 150-ton cyanide plant. The famous Comstock mines in Storey county have had much newspaper notoriety lately. Pennsylvania capitalists have obtained control of some of these mines. Whether the stockholders will receive larger profits, or the mill men secure most of the product, as in former years, remains to be seen.

New Mexico. (By Reinold V. Smith.)—Attention was given in 1908 to the old silver mines, many of which were reopened and equipped with new or renovated mills. The placer gold production remained about the same as formerly in the northern counties. The new placers recently discovered at Hachita have not had time thus far to add materially to the production. Operations at Hachita were rather extensive, so far as prospecting was concerned, but the regular development proceeded with caution. Dry washing was tried in this section and also on some of the older placer ground in Santa Fé county, but so far with no marked success.

The shipment of telluride ores from Sylvanite will, no doubt, add materially to the production of gold during 1909; shippers had barely commenced to send out their ores regularly before the end of 1908. The great source of gold in New Mexico is the quartz ore of Socorro county, and the gold-copper ores of Grant and Socorro counties. The Last Chance mine in the Mogollons, the largest individual producer in the Territory, increased its output almost 50 per cent. during 1908. The Rosedale cyanide mill was remodeled and enlarged and is working at full capacity again. The gold ores of the Mogollon mountains were extensively developed, and several companies were organized for exploiting that section. Six mills are now being put into commission in the Territory, three of them being cyanide mills.

Owing to preparations now under way in the southwestern and central western parts of the Territory, together with those of the counties in the north central part, the production of gold should be more than doubled in 1909. A large amount of exploration and development work is being conducted in the Mogollons, Hillsboro, Jicarillo, Lordsburg, and Central districts, in the Organ mountains, and in the Sangre de Cristo mountains and Colfax and Taos counties. Most of the producing sections are in

post-Cretaceous formations, but several promising prospects are in the pre-Cambrian, as, for instance, those in the Sandias, but in this formation the ore seems to be of low grade, averaging, perhaps, $3 and $4 per ton.

The districts of most importance in the production of silver were the Mogollons, the Organs, Pinos Altos, Steeple-rock, and the camps of the central part of the Territory. Among the important undertakings based largely upon the silver content of the ores should be mentioned the Boston Cerrillos Mines Company, an incorporation now commencing extensive improvements on the old Cash Entry mine near Cerrillos. The Last Chance mine, in the Mogollons, is by far the largest producer of silver in New Mexico.

Oklahoma.—According to the report of the Chief Mine Inspector, this State in 1908 produced gold ore to the value of $43,400.

Oregon.—As in previous years, the bulk of the gold production of this State was derived from the quartz mines in Baker county. Hydraulic mining was conducted in Josephine county. No important new districts became productive in 1908, the yield continuing to come from the older districts. A few concentrating and cyanide mills have lately been built. Several dredges have also been installed, and more are projected.

Philippines.—A small amount of gold was produced in 1908 in these islands, and several small stamp mills were installed. There are some promising prospects, but progress in development is slow.

South Dakota. (By John V. N. Dorr.)—Mining in South Dakota was fairly prosperous in 1908. No new properties were developed but the principal mines were in steady operation. As the production is mainly gold, the drop in the price of silver and other metals made no difference, and the principal difficulties were the gradual decrease in value of the ore at many of the old properties, together with the somewhat increased cost of labor due to the adoption of the 8-hour day. The use of electric power was extended, so that the only steam power now in use in the northern Black Hills is at the Homestake properties; indeed there also electric power is now in use at the three stamp-mills, the slime plant and the regrinding plant. Both the Consolidated Power and Light Company, with a steam-electric plant, and the Black Hills Traction Company, with a hydro-electric plant, are loaded to their capacity, and are planning to increase the latter. It is probable that in the summer of 1909 a hydro-electric plant generating from 5000 to 7000 h.p. will be built on Spearfish creek. The saving made by the use of electric power has been a material factor in maintaining the production of the Black Hills.

The Homestake company operated all its stamps continuously during 1908 and made a production, as given by the State mine inspector's report, of $6,000,000, which is considerably larger than that of any

previous year. The company increased the November dividend from 50c. to 75c. per share, the largest amount ever paid, but in December it was returned to the former figure. Among the improvements made in 1908 were the constructing of a large reservoir, the rebuilding of the foundry and the equipment of it with all modern improvements, and the installation of a regrinding plant to handle the coarsest sands from the batteries, the reground pulp passing over plates before going to the cyanide mills. The equipment consists of six grinding pans and a 5x14-ft. tube mill. The operation of the plant should give some interesting data as to the comparative efficiency of the two mills for regrinding under conditions obtaining here. The plant has proved very successful.

Of the silicious ore mines, the Mogul, the Golden Reward, the Imperial, the Lundberg, Dorr & Wilson, the Reliance and the Wasp No. 2 properties operated their mills steadily during 1908. The Imperial company reached the lower quartzite in the Dakota shaft during the summer, drifted several hundred feet in it and found some ore. If the further prospecting of the lower quartzite should disclose orebodies as strong and rich as these found in the shale above, it will mean a much longer life to the district. The Mogul company, operating its remodeled mill at Pluma, averaged over 320 tons per day, reaching a maximum of 388 tons in October. The Golden Reward company, which installed a Chamberlain press for handling its slimes early in 1908, has recently let a contract for a Moore-filter installation and expects to increase greatly the tonnage handled. The American Eagle Mining Company started a 200-ton mill during the summer, but found the mine developments were not adequate to operate this at full capacity and so closed down awaiting further development. In the Garden City district the North Homestake company operated its 40-stamp mill steadily during the year, and the Minnesota Mines Company, with a mill equipped with slow-speed Chilean mills, has commenced operations.

In the Galena district, the Gilt Edge-Maid company, working a very large body of low-grade porphyry and crushing dry to $\frac{3}{16}$-in. size, ran regularly, milling about 200 tons daily. This company is believed to have established a record for the Black Hills for low cost of mining and milling, which is reported to be approximately $1 a ton. The Golden Crest company, also mining in porphyry near the Gilt Edge-Maid, is building a 200-ton mill to crush with stamps, regrind with tube mill and treat the whole product by the Moore process. This is the first company in the Black Hills to erect an all-sliming mill. As the ore is very soft, the cost of complete slime treatment should not exceed that of a sand-and-slime treatment, and the recovery will undoubtedly be higher. The Branch Mint company started its 120-stamp mill, but found, as so often

happens, that it had gone ahead without sufficient mine development, so the mill was shut down after a short run. The property is now leased to a company which is running part of the mill on an old dump, while the mine is being developed.

Considerable work was done in the southern Black Hills during 1908, and the Standly company, at Rochford, ran its mill while doing development work that has extended the limits of its orebody. It promises to become one of the big mines of the Black Hills.

The problem of the successful treatment of the "blue" ores of the district, worth only $5 to $7 per ton, which are too low in grade to stand shipment to outside smelters, has received considerable attention. Renewed efforts have been made to obtain a better extraction by the use of other chemicals with cyanide. Some promising results have been obtained, but nothing has yet reached a commercial basis. Roasting will give a good extraction on much of this ore, but not on all, and it is considered too costly for the grade of the ore. The Black Hills will shortly be the place *par excellence* for the inspection of slime processes, for with the mills now building, there will be in operation four Moore plants, one Merrill sluicing filter-press plant, one closed-type filter-press plant, two decantation plants, one Burt press, one Chamberlain press and a Butters plant. It is of interest to note the influence of Black Hills practice on slime treatment. The Merrill sluicing press was first operated at the Homestake mill and the first successful installation using a submerged filter plant (a Moore process) has been in continuous operation at the Lundberg, Dorr & Wilson mill since February, 1904.

Utah.—The largest part of the gold and silver produced in this State are derived from the lead and copper mines, which are treated under the respective captions of "Lead" and "Copper" elsewhere in the volume. Of the gold producers, the mines in the Camp Floyd district increased their output, a new slime plant having been installed at the Mercur mine. The total of dividends paid by this mine up to the end of 1908 was $3,385,312. The Boston Sunshine Gold Mining Company reconstructed the mill of the old Sunshine mine and added new equipment for cyaniding the refractory, argillaceous ore of this once famous mine.

Washington.—Most of the mining activity centered in Stevens, Ferry and Okanogan counties. The First Thought mine in the Pierre Lake district, Stevens county, continued to be the most important producer of gold ore.

Wyoming. (By H. C. Beeler.)—Almost all the work on gold properties in 1908 was development and preparation. The Miners Delight mine in Fremont county shut down, owing to expense of fuel. The Popo-Agie Oil and Power Company is installing a plant to supply electric power to

this and the other mines of the South Pass gold district, using the oil of the Lander fields as fuel, the plant being situated at the old Murphy wells. At Centennial, Albany county, Copper mountain, the Wind River ranges, South Pass and Atlantic, in Fremont county, all reports showed activity in the gold properties but production during 1908 was small. Active placers were reported from the Little Big Horn, Sheridan county, Willow Rock and Twin Creeks, in Fremont county, Green and Snake rivers in Uinta county, Snake river in Carbon county, Sundance in Crook county and Douglas and Lake creeks in Albany county. The American Gold Placer Mining Company installed a steam shovel, prepared its pits and worked up to the last workable day of the season, but operations were frozen up before a cleanup was complete. Preliminary tests were satisfactory.

GOLD AND SILVER IN FOREIGN COUNTRIES.

Annam.—A correspondent of the *Echo des Mines* reports that the Bong-Miu Mining Company is now in a fairly flourishing condition. The production of gold is 25,000 to 30,000 francs per month, and may attain 500,000 francs a year. The mountains of the Annamite chain are full of similar deposits to that of Bong Miu. The deposit mined is a pyrite lode in gneiss, and was formerly worked by the natives, who, after washing the alluvials of the plain, naturally ascended to the source. They did not, probably, extract much gold from this lode, except, perhaps, in the oxidized outcropping quartz. There are a score of these old Annamite mines. In some places there are millions of cubic meters of alluvials, rather poor, it is true, but with abundance of water near. Prospecting these localities is not easy.

Australasia. (By F. S. Mance.)—The gold production of Australia has for some years past shown a steady decrease, although the falling off during the year 1908 was not so pronounced as in previous years.

PRODUCTION OF GOLD IN AUSTRALASIA.
(In ounces fine.)

State.	1902	1903	1904	1905	1906	1907	1908
Western Australia	1,819,308	2,064,801	1,983,230	1,955,316	1,794,547	1,697,554	1,648,505
Eastern States:							
Victoria	720,866	767,351	765,596	747,166	772,290	695,576	676,001
Queensland	640,463	668,546	639,151	592,620	544,636	465,882	461,359
New South Wales	254,435	254,260	269,817	274,267	253,967	247,363	224,792
Tasmania	70,996	59,891	65,821	73,541	60,023	65,355	60,453
South Australia(a)	24,082	21,195	29,177	20,330	24,439	10,651	14,500
Total Commonwealth	3,530,150	3,836,044	3,752,792	3,663,240	3,449,922	3,182,381	3,085,610
New Zealand	458,933	479,715	467,898	492,954	534,616	477,312	472,095
Total Australasia	3,989,083	4,315,759	4,220,690	4,156,194	3,984,538	3,659,693	3,557,705

(a) Northern Territory is included with South Australia.

PRODUCTION OF SILVER IN AUSTRALASIA.

State	1907			1908		
	Ounces.	Kilograms.	Commercial Value.	Ounces.	Kilograms.	Commercial Value.
New South Wales	12,149,672	377,906	$7,937,016	11,983,669	372,746	$6,232,657
Queensland	921,497	28,662	601,998	1,162,276	36,200	578,648
South Australia	1,000	31	653	(e)1,000	31	653
Tasmania	2,850,000	88,647	1,861,820	(e)2,400,000	74,650	1,260,000
Victoria	31,661	954	20,683	(e)30,000	934	15,870
Commonwealth	15,953,830	496,200	$10,422,158	.....	.....	.....
New Zealand	1,562,603	48,603	1,020,802	1,731,336	53,835	852,137
Total	17,516,433	544,803	$11,442,960	17,308,281	538,396	$9,034,965

Western Australia contributed 53 per cent. of the production of the Commonwealth during 1908, and the indications favor the opinion that the output has reached a normal level, and that there is not likely to be a further continuation of the decline which has characterized tne yield for some years past. Some of the outside fields certainly give evidence of partial exhaustion, but as against this the developments at the deep levels in the mines at Kalgoorlie, and the systematic manner in which operations are proceeding, strongly support the view to which expression has just been given. The dividends paid by mining companies in this State during the first 10 months of 1908 totaled $6,121,000.

The industry in Victoria cannot be viewed with equal complacency; the gold yield for 1908 was the lowest for 15 years. The falling off was principally due to the closing down of one after another of the deep alluvial mines, owing to the exhaustion of the deposits. The efforts made to prove the extension of these leads have involved the expenditure of large sums during recent years, but so far without success. The lower grade of the ore which is being won as depth is attained on the Ballarat and Bendigo fields, the increased cost of working, and the consequent lessening of the margin between profit and loss, are all adverse factors. As illustrating the depth to which operations have extended, it may be mentioned that the Victoria quartz mine at Bendigo has opened up a reef 4 ft. wide at a depth of 4504 ft. from the surface.

In Queensland the depression of 1907 has not in any way lifted. The government has endeavored to stimulate operations by voting a substantial sum for the purpose of carrying out deeper sinking and prospecting, and has also undertaken the construction of railways to connect the outlying mines. However, the success which attended the operations of the Mount Morgan company (referred to under the heading of "Copper") is a particularly pleasing feature of an otherwise disappointing year.

In New South Wales there were no developments of significance during 1908, and the yield was mainly contributed by the established mines

on the Cobar and Wyalong fields, and by the dredging plants. Here, as in the other eastern states, the waning productiveness of the gold mines and the absence of any developments that are likely to arrest the decadence are matters for much concern.

The production of Tasmania and South Australia was unfavorably influenced by the lessened output of those metals with which the gold in small quantity is found associated.

The yield of New Zealand showed a substantial gain. This was due to the increased output by the mines in the Ohinemuri district, which include the famous Waihi mine. This mine has been creating fresh records with each succeeding crushing, and the results have given a gratifying impetus to operations on the Thames field. The production of the Waihi mine in 1908 was £923,172 (from 393,214 tons of ore), an increase of £44,667 over 1907. The dredges at Otago and the West Coast, which have been important contributors in the past, furnished a reduced output during 1908.

Borneo.—The production of gold in Sarawak, North Borneo, from the beginning of 1898 to the end of 1907, was 308,629 oz. fine, or $6,379,361. The production for the 10 years, by years, was as follows, in ounces of fine gold: 1898, 1889; 1899, 12,499; 1900, 20,851; 1901, 32,220; 1902, 34,266; 1903, 40,329; 1904, 41,341; 1905, 44,299; 1906, 39,186; 1907, 41,751. The whole of this gold was produced by the two mines in Upper Sarawak belonging to the Borneo Company, Ltd., which has its offices in London. The Chinese do a limited amount of gold washing, but the government keeps no record of the output. At the best it is insignificant, and cannot amount to more than 100 oz. a year. At the two mines mentioned about 18,000 tons of ore are handled per month, all of which is cyanided directly. The bulk of the gold is obtained by cyaniding the clays, which are decomposition products of shales. This is done at one mill by treatment in lumps in very shallow vats—2 ft. deep—and at the other mill by puddling the clays, fine grinding the ore and sands and filter pressing. A large installation of Ridgway's vacuum filters has been made and the cyanide practice at these works is reported to be of a high order. Borneo is one of the countries which have not been thoroughly prospected, and it is not improbable that other gold deposits may be found.

Brazil.—The production of gold in this country has been dwindling, owing to the less favorable condition of the two principal mines, viz., the St. John del Rey and the Ouro Preto. The former has sunk to a depth of 4400 ft., where mining is rendered difficult by high rock temperature, which is said to be as high as 98 deg. F. The average extraction from the ore during the last six years was 38s. 2d., while the cost

averaged 33s. 2d. Since 1905 the rise in exchange has been severely felt. Although it reduced the purchasing power of the company in the country, it did not result in the cheapening of provisions and articles of primary necessity; thus, while the company was forced to reduce wages by 20 per cent., the cost of living to the workman did not diminish. The shortness of labor, especially skilled labor, has consequently been accentuated. The value of the ore treated by the Ouro Preto company in the last two years was 27s. 7½d. and 26s. 11d., respectively, while the costs were 22s. 11½d. and 22s. 7½d. respectively.

In the six months ending Aug. 31, 1908, the St. John del Rey crushed 80,200 tons of ore, which yielded £158,142, or 39.6s. per ton, a decrease of 2s. 7d. as compared with the corresponding period in 1907. The working costs in Brazil were 29s. 8d., mine developments, 10½ d., London general expense, 7¾d., making a total cost of 33s. 1½d. The ore reserves are estimated at 1,000,000 tons, but the grade is lower than what has been obtained previously. Undoubtedly there are many deposits of low-grade ore in Brazil which could be exploited profitably, if undertaken on a large scale, and if working costs were reduced. Economic exploitation of many of these deposits is, however, impossible under present conditions. The export duty of 3½ per cent. on the gold produced bears heavily upon the mining industry. Other onerous taxes augment the burden. In the meanwhile, American and European companies are doing a good deal of prospecting in Brazil for ground that is suitable for dredging, and several dredges have already been installed. In some cases, these dredges obtain diamonds as well as gold. Great hopes are based upon this industry, but its value has not yet been determined. The auriferous alluvials of the Upper Amazon valley have been described by Sir W. Martin Conway, in a paper reproduced in full in the London *Mining Journal* of Dec. 5 and 12, 1908.

British Guiana.—The statistics of gold production in the calendar years are given in the large table at the beginning of this article. In the fiscal year 1907-8, the gold won amounted to 67,209 oz., or about 18,295 less than for the previous year. The decrease was due to the smaller returns from the alluvial workings. The fields originally worked are gradually being exhausted, and no new ones have been found to take their place. A large number of men previously engaged in the goldfields have now turned their attention to other pursuits, such as rice planting, and prospecting has been practically at a standstill. Quartz mining was continued during the greater part of the year at the Peters mine on the Puruni river. About 8916 tons of ore were crushed, from which 5815 oz. of gold were won. Development work has been steadily carried on at the Barima mine. Dredging operations have continued on the Konawaruk

river, and about 2652 oz. of gold were obtained by this method. The richer gravels of the higher part of the river have not yet been worked, and better returns may be looked for in the future.

Canada.—For the first time in nine years the gold output shows an increase over the previous year. The Yukon output in 1908 is estimated at about $3,600,000, as compared with $3,150,000 in 1907, while a considerably increased production is also shown in British Columbia, derived chiefly from ores, the placer workings having shown a smaller output. In Nova Scotia the output in 1907 was $282,686. Complete returns are not yet available for 1908, but the output was probably not over $225,000. Of the total Canadian gold output in 1908, over 44 per cent. was obtained from placer and hydraulic workings and 56 per cent. from lode mines.

In British Columbia the production of placer gold in 1908 was still more disappointing than in 1907, and was the smallest of any year since 1898, having been only $682,000. The greater part of the decrease was in Cassiar district, the estimated shortage from Atlin division of that district having been $183,000, and of the Liard and Stikine divisions $25,000. Individual miners produce comparatively little gold in Atlin camp now, and for one reason or another the hydraulicking companies recovered much less gold in 1908 than in several previous years. There is, though, much gold-bearing gravel still available on three or four of the Atlin creeks, so it is to be expected that with an adequate water supply, which has recently been provided by one or two of the larger producers of the camp, a better result will be obtained in 1909. The three divisions of the Cariboo district are each credited with an increase, their mineral production being entirely placer gold. The Similkameen division of Yale district appears to have increased its yield by $9000, but Lillooet shows $7000 less than in 1907. The figures for the Kootenay districts and the Coast do not exhibit any special changes in the production of placer gold.

The decrease shown against Cassiar district is partly the result of the transfer of a portion of Skeena mining division to the Coast district, while that of Lillooet is the result of a temporary stoppage of dredging on that part of the Fraser river.

In lode gold, of which the total production was $5,291,520, the largest proportion of the increase was made by the copper mines. Of the two larger gold mines of the province, the Ymir in Nelson division, and the Nickel Plate, near Hedley, the latter is understood to have maintained its production on a similar scale to that of several recent years, but the former was practically a non-producer. The development of several

small gold mines in Sheep Creek camp, was one of the features of progress in 1908.

GOLD AND SILVER PRODUCTION OF BRITISH COLUMBIA.

	1905		1906		1907		1908	
	Oz.	Value. (a)	Oz.	Value. (a)	Oz.	Value. (a)	Oz.	Value. (a)
Gold, placer....	48,465	$ 969,300	47,420	$ 948,400	41,400	$ 828,000	34,100	$682,000
Gold, lode.....	238,660	4,933,102	224,027	4,630,638	196,179	4,055,020	256,000	5,291,520
Total gold....	287,125	$5,902,402	271,447	$5,579,038	237,579	$4,883,020	290,100	$6,973,520
Silver.....	3,439,417	1,971,818	2,990,262	1,997,226	2,745,448	1,793,519	3,037,000	1,606,536

(a) Placer gold is valued at $20 per oz.; lode gold at $20.67 per oz.; silver at average market quotations.

The production of silver in British Columbia was larger in 1908 than in 1907. West Kootenay continues to hold its lead among the districts producing silver. Further, by far the greater portion of the silver produced in East Kootenay in 1908 came from one mine, the St. Eugene, while in West Kootenay a dozen or more of the larger mines and a score or so of the smaller ores combined to make up the production. West Kootenay's production was from the following localities: Slocan, 953,000 oz.; Ainsworth division (which includes part of the district generally known as the Slocan), 330,000 oz.; Lardeau, 206,000 oz.; Rossland, 156,000 oz.; and Nelson, 72,000 oz.; together 1,742,000 oz. East Kootenay produced 675,000 oz., from St. Eugene, North Star and Sullivan mines. The Boundary district's production was 596,000 oz., chiefly from the copper mines of the Granby and British Columbia copper companies. More than three-fourths of the silver from the Lardeau in 1908 was from the Silver Cup mine, with the True Fissure contributing nearly all the remainder from that district. In Nelson division the La Plata and Silver King mines produced most of the silver.

In Quebec, the River Gilbert valley, Beauce county, where since 1878 workings have been carried on in irregular fashion and on a small scale, is estimated to have produced $2,000,000 of gold within a distance of two miles. Numerous quartz veins containing gold have been discovered, but so far gold in commercial quantities has not been obtained. In Marston township a deposit of auriferous quartz occurs and a local syndicate has been formed to operate it. This is the first instance of the finding of gold in the quartz in the Chaudiere valley in apparently commercial quantity, though the actual test as to this has yet to be made.

The silver production of Ontario in 1908 was 19,424,781 oz., this being almost wholly from the Cobalt district. The bulk of the Cobalt ore is shipped to smelters in the United States, but a portion is now being refined in Canada. Thus, the Deloro Mining and Reduction Company,

Ltd., of Deloro, Ont., shipped about 2,500,000 oz. of pure silver bullion to London in 1908.

(By John Power Hutchins.)—The gold production for the Klondike district in 1908 was about $3,600,000. The figures given in the accompanying table show a rapid increase to a maximum output and then a

GOLD PRODUCTION OF YUKON TERRITORY.

Year.	Amount.	Year.	Amount.	Year.	Amount.	Year.	Amount	Year.	Amount.
1896.	$ 300,000	1899.	$16,000,000	1902.	$14,500,000	1905.	$7,000,000	1908.	$3,600,000
1897.	2,500,000	1900.	22,275,000	1903.	12,250,000	1906.	6,000,000		
1898.	10,000,000	1901.	18,000,000	1904.	10,350,000	1907.	3,150,000		

gradual decrease, a history similar to that of other placer districts. For the last three years the Klondike has been in a transition period. The one-man enterprise has had its day, and now a powerful corporation has control of several of the richest creeks. During these three years preparations have been going on to work these creeks on a large scale with dredges and other machines and a system to supply water for hydraulicking has been in the course of construction. It is expected that this will be finished early in 1909. When operations on a large scale are under way, an increase in the gold output may be expected.

The discovery of rich and extensive gravel deposits in the Klondike district in the future is improbable. There is no such intricate and large system of "dead rivers" as in California and the total volume of gravel in the Klondike that may be worked profitably is considerably less than the profitable hydraulic mining ground remaining in California, but which can not be mined because of anti-débris legislation. The principal deposits that will be worked are the creek beds and the "white channel." The "white channel" is what is left of the ancient creek beds; it has a course approximately parallel to the present creeks but at an elevation from about 100 to 300 ft. above them. Where it has not been eroded, this channel usually appears with one rim completely removed. Locally, it is called a "bench deposit." There is one notable exception in the occurrence of the "white channel;" this is the deposit between Lovett Gulch and the Klondike river where the two rims are intact for several thousand feet.

There was little novelty in the mining methods introduced in 1908. Dredging was carried on with better results than in 1907. This was due to several things. Steam thawing was employed on a larger and better scale, and there were fewer attempts to dredge frozen gravel. Stripping had been found insufficient to thaw gravel to bedrock unless carried to such a depth as to leave a shallow gravel section. It had been found

that thawing progressed to but a slight depth into gravel below the water level. Draining the gravel with cuts will be applied. It is expected that stripping and draining with cuts 20 to 50 ft. apart will result in progressive thawing to bedrock, after the thawing of one or more summer seasons, and that this thawed ground will remain unfrozen. It is yet to be determined whether ground thawed during the summer, no matter by what method, will not freeze again during the subsequent winter.

Since the merger of claims on Bonanza, Eldorado and Hunker creeks there has been practically no winter drifting, and there has been less of this kind of mining on most of the other creeks. There were a few dumps taken out on Quartz creek where "lays" were about 65 to 70 per cent. of the output to the "layman." The labor supply was augmented by men brought in from Vancouver, and it was ample until autumn, when the demand was greater than the supply. There were no labor troubles, although difficulty was threatened when it was thought that large numbers of cheap laborers would be brought to work for the merged interests.

No paying quartz mines have been developed in the district. For several years work was done on a vein on Bonanza creek, but this has been stopped. At present considerable interest is being taken in some quartz prospects on the Dome, a mountain dominating the topography near Dawson and from the flanks of which several of the richest Klondike creeks radiate.

Hydraulic mining was carried on as usual; about three times as much material, or about 3,500,000 cu.yd., was washed as in 1907. The cost of hydraulicking did not vary much. It is probable that when water from the new ditch is available, a considerable reduction will be made. A duty of 8 cu.yd. per miner's inch per 24 hours was attained some years ago where about 275 miner's inches was used. Several factors influenced this result, the most important being an 8-per cent. grade, good dump and gravel of a favorable admixture of fine material and having no large boulders. A bedrock favorable for running and maintaining cuts assisted in keeping them close to the banks at all times so that little time was lost in "driving." There will be an opportunity of making some new records in the duty of the miner's inch, when a large volume of water is applied on the White channel. The Klondike climate, with its extreme cold and the frozen ground, is hard on earth work. It is difficult to maintain roads, dams and ditches. Thawing results in the sloughing off of material from the banks of ditches, and where the course of the ditch is over "muck" and on northern exposures the strictest vigilance is needed to prevent serious breaks.

The cost of dredging is said to have varied from about 15c. per cu.yd. for a 7-cu.ft., close-connected bucket dredge in unfrozen ground to about

20 to 35c. for similar dredges working in ground originally frozen in part and thawed by stripping and steam thawing. An antiquated dredge of small capacity operating in partly frozen ground showed a cost of about 60c. per cu.yd. Thawing for dredging costs from 10 to 20c. per cu.yd. No figures are yet available for dredging solidly frozen ground, and the cost of thawing such material is not yet known. All the dredging done up to the present time has not been in what is locally called badly frozen ground.

Wood hauled in 16-ft. lengths on wheels cost about $14 per cord. Supplied in large quantities it was sold for about $8 per cord, where it could be rafted and floated down the Klondike river without much hauling. Less coal was burned in 1908 than in previous years. The coal available has been found to possess poor steam-producing qualities; it makes dirty fires and much ash and clinker. One ton of coal has been found to have about the same calorific value as one cord of wood, and since wood can usually be purchased for less than coal it is generally used.

The following letter to the stockholders of the Yukon Gold Company was issued by President S. R. Guggenheim on Jan. 30, 1909:

At the time our engineers made their first examination in 1905, the famous placer fields of the Yukon were fast becoming depleted of their rich ground; the volume of gravel workable at a profit by hand method was rapidly being reduced, and the cost of working, under the most approved methods then in vogue, precluded the extension of the operations to take in the large bodies of lower-grade material. The principal factors making toward the high cost were:

(1) The disconnected nature of the holdings (the claims being not over 500 ft. in length and owned largely by individuals), which reduced the workings to a number of small-scale operations, and prevented systematic mining upon any larger scale.

(2) The scarcity of water. The miners, both those working upon the creek levels and those upon the bench and hill deposits, were entirely dependent upon the rain-fall for their supply of water. As the snow and rain-fall is very irregular in the Klondike district, a corresponding irregularity entered into the mining operations, with a consequent increase in cost. No large scale hydraulic operations were possible with such limited and intermittent water supply.

(3) The high cost of fuel, labor and supplies. Wood fuel is available in the Klondike for mining purposes, but its cost is high and constantly increasing. Coal is to be had, but it is as expensive as wood. Labor is high, due to the short season for outside work, and supplies expensive

because of high transportation charges, and the small and irregular character of the purchases.

Our investment in the Yukon was based upon three fundamental hypotheses: (1) That we would be able to acquire large and connected blocks of ground for systematic working. (2) That we would be able to secure and make available at sufficient elevation a continuous water supply. (3) That we would proceed to equip the properties with the most modern mechanical appliances, such as dredges and elevators, for handling the gravels upon a large scale and at a low cost, depending for power not upon local fuel, but upon electrical energy generated in our own hydro-electric plant.

All of these things have been accomplished. At the present time our holdings amount to about 90 per cent. of the claims located on the hills and creek bottoms of Bonanza, Bear, Hunker, and Eldorado valleys, which were wanted for the enterprise. These properties have been acquired mainly by outright purchase, although in some cases, where unacquired claims adjoin the properties of this company, advantageous working agreements have been made with the owners.

Early in 1906 construction was begun upon a water system which will insure to the Klondike a continuous water supply for all time. This was in itself a tremendous undertaking, involving the diversion of Tombstone river (the main right fork of the Twelvemile), carrying it a distance of about 62 miles by flume, ditch and pipe lines, through a country which was an absolute wilderness when our engineers first went upon the ground. In its course the main ditch follows the contour of the country, which, due to its rugged and partly frozen character, required miles of continuous flume construction. The line crosses five depressions, making an aggregate of nearly 12 miles of pipe line. The ditched sections, aggregating 37 miles, required the use of seven steam shovels in excavation, and the employment of more than 1000 men throughout three summers. The water is carried across the wide Klondike valley in an inverted siphon, which for length, size and pressure involved, is not, so far as we know, equalled in the world.

The construction of this water system was carried on in the face of great difficulties. It involved the distribution of thousands of tons of supplies and construction material; the manufacture, distribution and use of more than 8,000,000 ft. of lumber, produced by our own logging and sawmill operations; the hauling over winter roads; and distribution of several thousand tons of steel and wood-pipe line; and its installation at an average distance of 35 miles from the base of supplies. In the construction of the flume, ditch and pipe line, many engineering difficulties were encountered and overcome, which would not be met anywhere,

except in the far northern country. The complete work today is evidence of persistent and untiring effort upon the part of the men in charge. Many of the engineering feats accomplished have prompted praise from the technical press. They mark an era in Alaska mining history.

Simultaneously with the construction of the water system, a hydro-electric power plant was installed on the Little Twelve-mile river, capable of delivering 2000 h.p., which is transmitted to the placer fields by a high-tension line, 36 miles straight across country, with three substations and 18 miles of secondary line. A huge dam has been built at the head of Bonanza creek to conserve the water of the upper Bonanza watershed for use upon the company's Bonanza and Eldorado Creek properties. Equipment plans have been carried forward in a manner to evoke enthusiasm and inspire confidence in a speedy consummation of the enterprise as a whole. Seven dredges have been erected, the last of which was completed and put in operation late in the season of 1908. These dredges are of large capacity and great strength, able to withstand the most severe service. Their operation in the latter part of 1908 demonstrated the entire success of this method of mining, as applied to the gravels of the Yukon. The difficulties due to frozen ground have been overcome within the limit of a reasonable working cost, and this cost will, without doubt, be greatly reduced when the main water supply is available for stripping the gravels in advance of the dredges.

Three mechanical elevators were installed, two of which were put into operation during the season of 1908. It is a great pleasure to say that the mechanical elevators, electrically driven, which were designed by our own engineers and used here for the first time in any country, were entirely successful. They will be presently operating upon the ground from which the gold could not be thoroughly extracted by any other method of mining known to us. The successful operation of the elevators solves one of the most difficult problems in connection with our Yukon business—that of handling flat, shallow gravels, largely frozen, where the bedrock is rough and irregular and the gold extends to some depth into the crevices.

Since acquiring the properties in the Yukon, we have carried on hydraulicking to a limited extent, and dependent upon the local intermittent water supply. The operations have shown a profit, and have proved not only the value of the gravel, but also that a very low working cost can be reached. In the season of 1909 we expect to have the equipment working up to at least one-half of its ultimate capacity. In the season of 1910 we fully expect to show a production which will make the Yukon Gold Company take rank as one of the largest gold mines in the world.

We have only proved our holdings in part, first, because of the advisability of concentrating our engineering staff upon the installation of the present equipment, and second, because of the expense attendant upon the thorough prospecting of ground in the north. Our engineers having reported that we now have nearly $40,000,000 net in proved ground, which is approximately 100 per cent. profit on the shares at par, and provides for many years' dividends, we concluded to postpone further prospecting of our holdings, until our present plant and equipment are running to full capacity. There is good reason to believe that additional workable ground will be developed and economies introduced which will greatly increase the ultimate net profit.

For the last season final results have not as yet been actually determined, but incomplete figures show that the net earnings to Dec. 31, 1908, will be about $550,000. Without doubt, we should have reached our estimated net of $800,000, had it not been for delay in delivery at Dawson of machinery, which was held up at Whitehorse by unusually low water in the Yukon river immediately after the opening of navigation. This delay cut the length of our season materially, but possibility of repetition may be omitted from future calculations, now that our equipment has been delivered and installed. Our engineers have estimated the probable earnings for 1909 at from $1,100,000 to $1,500,000. The wide difference in these figures is due to the unknown factors entering into the water supply, and the uncertainties attendant upon the first operation of a new ditch line of such length, and constructed under such difficulties. If we have comparatively few washouts or accidents on the main ditch system or the equipment, our earnings will probably reach the higher figure. In 1910 the net profits of the company should be from $2,000,000 to $2,500,000, and in 1911 and thereafter, $2,500,000 net per annum should be the minimum.

China. (By T. T. Read.)—China is a buyer of silver, and as a producer of the metal is entirely insignificant. Gold is rather widely distributed throughout the Empire, but in the extreme southwest and the extreme north are the most notable localities. Very little is known of the production in the southwest, but the different explorers, Richtofen, Garnier, Baber, Hosie, and the rest, all mention numerous places in which it was being washed from the stream-courses. It would be hazardous to estimate the amount annually produced in this part. In the north, in Shantung, Manchuria and Mongolia both alluvial gold and quartz veins occur. The most important localities are in Mongolia, just south of the Ussuri river. Probably the larger part of the production comes from this place. $3,200,000 worth of dust and bars passed through the customs in 1907. What proportion of the total production this is

cannot be estimated. The quartz veins seem to be just large enough and rich enough to stimulate efforts to develop them, without yielding any profit. There is one valuable vein in Shantung, but it is not now worked.

Colombia. (By John Power Hutchins.)—Colombia is credited with a total gold production of $895,000,000. At present this republic has a few profitable gold lode mines. The most important of these is the Frontino & Bolivia near Remedios, in the department of Antioquia. A good deal of investigation of gold lodes has been carried out in southwestern Colombia for several years by Americans. Several promising properties are under development in this region near the boundary of Ecuador. Difficult transportation has been and will continue to be a serious obstacle.

Cuba.—Recently the production of gold became sufficient to merit mention, the Holguin-Santiago Mining Company having produced as high as $20,000 a month. The adjoining mine, La Casualidad, employs about 30 men. An undeveloped mine on the same vein as the Holguin-Santiago and La Casualidad is called Relámpago, while to the east of that is La Caridad. The outlook for the early development of gold mining on an important scale in Cuba is reported to be promising.

Dutch Guiana.—The *Indische Mercuur* gives the gold yield of this Colony for the last three years as follows: 1908, 1,209,781 grams; 1907, 1,104,396 grams; 1906, 1,188,205 grams. The Guyana Goud Placer Maatschappij recently sent out experts to report on its property. In the report a large portion of the ground is deemed suitable for dredging on a large scale. The management has decided to make a beginning with dredging. In the gravel formation of the placer several diamonds were found. The Nederlandsch-Surinaamsche Goud Maatschappij has resumed work with its dredge. The colonial regulation requiring the holder of a mining permit to employ at least four workmen, has been changed to three. The regulation compelling the holder of a permit to supply a detailed return as to the workmen in his service immediately after his arrival, has also been altered. For this return a space of four weeks from the day of arrival is permitted.

Ecuador.—Lode mining is carried on only in the Zaruma district, where the cyanide process has made it possible to do profitable mining. A small amount of placer gold is produced on the head waters of the Amazon river. This is an extremely inaccessible region and will probably remain undeveloped for many years.

French Guiana.—According to J. Bel, the gold production of this colony in 1906 was 3388 kg., valued at 9,146,692 francs, though the actual production was doubtless between 12,000,000 and 15,000,000 francs, the difference being represented by gold smuggled out of the country. Since 1906 the production has been almost stationary. Employment of better

methods of mining, improvements of transportation facilities, and an abatement of the exorbitant taxes would greatly stimulate the production of gold in French Guiana.

French Guinea.—The goldfields of French West Africa attracted considerable attention in France in 1908. The deposits of Upper Guinea, which are practically all in the district of Siguiri, have been examined by many companies. This district may be easily reached via Dakar. From Dakar the traveler goes by rail to St. Louis in Sénégal (264 km.); thence by steamboat up the Sénégal river to Kayes (900 km.), a journey of five to six days; again by rail to Bamako (496 km.) and then by boat on the Niger river to Siguiri (300 km.). The journey from Bordeaux to Siguiri, may be covered in 20 to 25 days at the time of high water, viz., August, September and October. According to Commandant Villiaume, the gold deposits of eastern Guinea have many features in common with those of Rhodesia. Most of the region is underlaid by old clayey sediments changed by an intense metamorphism into banded crystalline schists and slates that dip at a high angle. The gold deposits are closely related to eruptive rocks; the closer the deposit is to the eruptions, the richer it is. All the rocks, especially the crystalline rocks and slates, down to permanent water level are decomposed into an earthy and brittle material called laterite.

The country is very flat but presents some depressions which owing to the concentration of gold in them are of special interest from the mining view-point. The width of these depressions generally varies from 30 to 150 m. The main valleys or depressions are 25 to 60 km. long, while the secondary ones are only 2 to 10 km. long.

The Compagnie Minière de Guinée and the Compagnie des Mines de Siguiri have begun to work their properties and are installing machinery to ascertain the best way of treating the auriferous clay. In Kobako valley the Compagnie Minière de Guinée will dig the laterite with two excavators of the ladder type each having an estimated capacity of 500 cu.m. per day of 10 hours. The Compagnie des Mines de Siguiri is to work its Fatoya hill property. Small canals fed by a centrifugal pump will transport the ore to a system of sluices with undercurrents. The quartz separated by means of grizzlies will be crushed and amalgamated in two milling plants, both similar to the one erected by the Compagnie Minière de Guinée. These plants are expected to be ready to run early in 1909.

Gold Coast Colony and Ashanti. (By W. Fischer Wilkinson.)—The gold production during 1908 was about £1,183,000, which is somewhat in excess of that obtained in 1907. The increase was mostly due to the Taquah mine, which milled the whole year, while in 1907 its production was limited to the closing months. This mine is working banket ore. The

mill of 50 stamps is run by a gas engine and producer plant, the first on the coast. The plant is reported to be an unqualified success. From September, 1907, to June, 1908, the mill crushed 46,234½ tons (2000 lb.) for a return of gold of the value of £162,243 or 68.36s. per ton. During this period the mill ran only 59.55 per cent. of its possible full time owing to want of ore and breakdowns of the old machinery in use. The costs including development redemption (6.5s.) and London expenditure (2.4s.), but not depreciation, were 42.1s. per ton milled. With better running time the costs will no doubt be less but the grade seems high compared with the other mines along the banket section. The ore reserves are given as 233,941 tons, value not stated, but they include 176,971 tons assaying 21.6 dwt.

The Abosso mine, which is managed by the same men as the Taquah, has a 50-stamp mill, 20 having been added in February, 1908. During the financial year ending June 30, 1908, 44,021 tons were crushed, the recovery being 50.4s., and the costs, including development (5.25s.) and London expenditure (1s.) but exclusive of depreciation, 35.2s. per ton milled. The costs were above the normal owing to a series of accidents at the mine, the lower levels being more than once flooded. The ore reserves on June 30, 1908, were estimated at 210,801 tons of 18.7 dwt. value.

The Ashanti Goldfields Corporation showed a net profit for the year ending June 30, 1908, of £34,646. This company operates a group of quartz mines round Obuassi and treats the ores partly by wet stamp milling and cyaniding, and partly by dry crushing, roasting and cyaniding. For the 12 months ending June 30, 1908, there were crushed wet 43,135 tons, and dry 16,021 tons, total 59,156 tons of a total value of £171,033, equal to 56s. per ton. The costs are given as 25s. 1d. London expenditure, depreciation and sundries foot up to another 16s. per ton, making a total cost of 41s. per ton compared with 40s. during the previous year.

The event of the year was the discovery by J. N. Justice, and since called "Justice's Find," of a large orebody composed of a soft oxidized and decomposed schistose rock turning in depth (below the zone of oxidation) into a fairly hard basic rock still showing schistose structure, highly impregnated with pyritic mineral, chiefly arsenical, and carrying payable quantities of gold. How far the value is maintained in depth is not yet known, but the indications are promising. Enough work has already been done to show a large tonnage of 14-dwt. ore which can be cheaply won by open workings. The ore is refractory and the bulk of the gold cannot be detected by simple panning. Although refractory, a good extraction is obtainable, without roasting, by sliming and filter-pressing.

The Prestea Block "A" mine made a large gold production, but a yield of about 34s. does not allow of much margin of profit as costs are at present. In the November report it is officially stated that notwithstanding present conditions the mine is paying its way. Improved results are expected when the railway connection with Tarkwa is completed, the completion being expected next autumn. The Abbontiakoon Block 1, the Bibiani and the Wassau have also grades of ore that cannot leave much profit, if any, judging by the costs of the other mines. The Broomassie mine with 20 stamps treats very high-grade ore.

The results of 1908 in these goldfields were not very profitable, yet there is a promise of better in the future. The industry continues to gain by experience, and costs are coming down. Gold undoubtedly exists in large quantities and over a wide area and if only the cost of working can be reduced to 20s. a ton instead of 40s. a ton as it now appears to average, when development redemption, depreciation, etc., are allowed for, a prosperous future awaits the industry. In spite of the disappointing results that the country has given to those who have put their money into mines, there are not wanting those who have faith in a large reduction in working costs being possible. It was announced late in the year that the Consolidated Goldfields of South Africa was providing working capital for three mining companies in the Wassau district, the Effuenta, the Fanti mines and the Wassau West Amalgamated.

GOLD PRODUCTION OF GOLD COAST COLONY AND ASHANTI, SINCE 1880.

Year.	£	Year.	£	Year.	£	Year.	£	Year.	£
1880....	32,865	1886.....	74,878	1892.....	98,805	1898.....	63,837	1904.....	345,608
1881....	45,240	1887.....	81,168	1893.....	79,099	1899.....	51,299	1905.....	657,330
1882....	61,188	1888.....	86,510	1894.....	76,795	1900.....	38,006	1906.....	877,568
1883....	52,435	1889.....	103,200	1895.....	91,497	1901.....	22,186	1907.....	1,154,885
1884....	66,188	1890.....	91,657	1896.....	86,186	1902.....	96,880	1908.....	1,183,066
1885....	89,981	1891.....	88,112	1897.....	84,797	1903.....	254,790		

PRODUCTION OF GOLD COAST COLONY AND ASHANTI, 1908.

Company	Stamps.	Tons.	Value.	Value per Ton.
Taquah	50	62,803	£216,788	68s.
Ashanti Goldfields	40	62,285	156,753	50
Prestea, Block "A"	50	89,643	153,489	34
Abosso	50	48,761	115,453	47
Abbontiakoon, Block 1	50*	65,290	114,333	34
Broomassie	20	19,547	111,607	114
Bibiani	45	49,876	97,531	39
Wassau	40	62,200	86,664	28
Akrokerri	40	33,660	61,677	
Dredging and other small producers			67,771	
Total	385	494,055	£1,183,066	51.75

* Ball mills equivalent.

GOLD PRODUCTION OF WEST AFRICA.

Year.	Fine Oz.	Year.	Fine Oz.	Year.	Fine Oz.	Year.	Fine Oz.	Year.	Fine Oz.	Year.	Fine Oz.
1897.....	23,555	1899...	14,250	1901...	6,088	1903...	70,763	1905...	165,844	1907...	271,426
1898.....	17,733	1900...	10,557	1902...	29,890	1904...	94,815	1906...	199,432	1908...	279,320

Japan.—Gold production in first half of 1908 was 35,905 oz., against 36,847 in the corresponding period of 1907. Silver, 1,706,443 oz., against 1,306,784.

Korea.—According to a recent report by Thomas Sammons, U. S. Consul General at Seoul, the exportation of gold from Korea is about $2,250,000 per annum, but the actual amount is larger, owing to the clandestine traffic which escapes the official accounting. About half of the reported production is made by the Oriental Consolidated Mining Company, an American concern, operating in the Unsan district of northwestern Korea. This company has already produced upward of $10,000,000, its production in 1907 being approximately $1,150,000. The company is said to have ore reserves amounting to 1,000,000 tons averaging $5 to $5.20. The annual crushing is about 300,000 tons, operating expense being about $2.50 per ton. American interests are developing promising mines at Suan Chiksan. At the former a large body of ore has been developed, while the latter is known to have extensive deposits, both quartz and placer. The Chiksan district has been one of the most profitable placer fields worked by the Koreans, but conditions for working are rather unfavorable. However, for dredging the conditions are regarded as fair.

Madagascar.—Practically all of the gold production of this Colony is obtained from alluvial workings. About 30 per cent. comes from the district of Diego-Suarez in the northern part of the island, where rich placer deposits were found three years ago. The Government charges a royalty of 7 per cent. on all gold produced. Production was 72,100 oz. in 1907, against 64,844 oz. in 1906.

Mexico. (By H. A. Horsfall.)—The gold output in 1908 increased at a rapid rate, notably in the districts of Guanajuato, El Oro and Dolores. The increase was partly due to the discovery of new mines, but principally to the increased use and continued improvement of the cyanide process for treating gold and silver ores. A large number of new mills using this process were put into commission during 1908. Notwithstanding the adverse conditions prevailing generally, the quantity of silver produced shows a material increase over previous years, new producers having come to the front and the old ones having increased their output. In prospecting greater attention was given to gold properties and many promising prospects were located.

Netherlands East Indies.—The Keta Loen mine in 1908 crushed 39,670 tons of ore, producing 25,773 oz. of gold and 38,440 oz. of silver, at a profit of £42,750. The mine is estimated to have ore reserves of 162,000 tons, assaying 0.6 oz. gold per ton. The Redjang Lebong mine in 1908 crushed 89,760 tons of ore, yielding 82,868 oz. of gold and 471,630 oz. of silver, at a profit of £266,916. The ore reserves at the end of 1908 were estimated at 308,000 tons, assaying 0.95 oz. gold per ton.

Nicaragua.—The correspondent of the London *Mining Journal* gives the following figures for gold in 1908: Bluefields exports, $468,171; Cape Gracias exports, $265,000; miscellaneous, $200,000; total, $933,171. The irregularity of production is due to unsystematic management and working. There are several good mines stopping because the machinery is to be changed and mills are to be erected on new mines; also some cyanide plants are in erection, so that 1909 will be much better. The number of working mines is very small in proportion to those which cannot be worked for want of capital, which is, however, coming in slowly. There are many mines in the interior. The La Libertad-St. Domingo mining district had 26 mines running—more than ever since its discovery. Several mines also ran successfully in the Matayalpa district. All this gold goes to the Pacific side—partly to San Francisco, partly to England, and sometimes to Germany.

Rhodesia.— (By W. Fischer Wilkinson.)—The gold production for 1908 was about £2,525,000, which is £346,114 in excess of the production of 1907 (£2,178,886). The accompanying table, compiled by the Rhodesia Chamber of Mines, gives the total gold production from Southern Rhodesia to March 31, 1908. In this table the ounces are partly bullion

SEPTEMBER PRODUCTION SOUTHERN RHODESIAN GOLD MINES.

	Tons Milled.	Stamps.	Total Yield.		Tons Milled.	Stamps.	Total Yield.
Matabeleland:				*Mashonaland:*			
Bell	1,128	10	£3,092	Joker	447	4	£4,429
Bush Tick	6,115	25	5,266	Battlefields	1,686	10	3,537
Nelly	1,690	15	4,077	Giant	6,252	15	(b)2,733
East Gwanda mines	9,848	70	12,207	Ayrshire	8,221	60	3,289
Gaika	2,953	5	4,610	Eldorado	5,123	20	10,271
Globe and Phoenix	6,412	40	10,775	Jumbo	2,215	30	6,647
Selukwe	5,650	40	5,435	Penhalonga	11,100	60	8,595
Surprise	2,586	20	4,305	Rezende	3,100	25	4,525
Wanderer	15,252	(a)100	5,851				

(a) Dry crushing process.
(b) Owing to collapse of main shaft in August the yield is not normal. The average for January to July was £10,700 per month.

ounces. It has been arranged for all declarations of gold from April, 1908, to be made in fine ounces; and the value of the ounce will be calculated as in the Transvaal at £4.24773. The output is largely made up of the returns from small mines or tributers working with an average of

five stamps. At the close of 1907 the small workers were contributing about 35 per cent. of the output. The principal mines are given in the accompanying list, compiled from the September returns. The total production of southern Rhodesia for that month was valued at £204,261 from 159,934 tons.

GOLD PRODUCTION OF RHODESIA TO MARCH 31, 1908.

	Tons Milled.	Gold, Oz.	Value.	Value per Ton.
Prior to Sept. 1, 1898	(a)	6,471	£23,456	
Sept. 1, 1898, to June 30, 1899	81,841	48,847	177,072	43.26s.
July 1, 1899, to June 30, 1900	104,746	57,621	208,877	39.88
July 1, 1900, to March 31, 1901	140,716	89,258	320,457	45.54
Year ending March 31, 1902	249,667	180,910	640,661	51.32
Year ending March 31, 1903	338,156	201,107	709,461	41.96
Year ending March 31, 1904	516,747	234,693	845,359	32.71
Year ending March 31, 1905	787,936	309,516	1,113,068	28.25
Year ending March 31, 1906	1,100,609	435,019	1,556,741	28.28
Year ending March 31, 1907	1,417,723	560,750	2,012,543	28.39
Year ending March 31, 1908	1,652,019	624,764	2,289,452	27.71
	6,390,160	2,748,956	9,897,147	30.97

(a) No details available

In the presidential address delivered to the Chamber of Mines on May 20, the following average costs of 11 of the principal mines, omitting the Wanderer mine, which is a low-grade property, working under exceptional circumstances, are given: The 11 mines crushed 534,332 tons and treated 271,443 tons of sand and concentrates. The costs per ton milled were as follows: Mining, 7s. 10.51d.; sorting, crushing and tramming, 1s. 5.01d.; milling, 4s. 4.17d.; cyaniding and concentrating, 2s. 3.98d.; general charges, 1s. 11.65d.; development redemption, 4s. 2.55d.; total, 22s. 1.87d. The average recovery was 30s. 10.08d., leaving an average working profit of 8s. 8.21d. One of the mines was working at a loss.

One of the successful mines of Rhodesia is the Giant, which for the year ending June 30, 1908, milled 65,074 tons with a 15-stamp mill and one tube mill, obtaining a recovery of 36.8s. per ton milled. The working costs, including development, totaled 15.5s., or 18.5s. if depreciation and London expenditure be added. Owing to the caving in of the main shaft, the successful operations have recently been thrown back, and about six months will be required to sink a new shaft. The orebody has lately widened out considerably at the sixth and seventh levels, being more than 100 ft. wide and all in pay ore, a discovery which has greatly improved the future prospects. To meet the cost of the new shaft and to double the mill capacity (a policy which the new discovery of ore mentioned above has influenced) £50,000 is being raised by a debenture issue. The Giant is one of the dividend-paying mines, having paid £17,812 in

1905-6, £35,625 in 1906-7 and £41,562 in 1907-8 on a capital of £250,000. Another dividend payer is the Globe & Phœnix, one of the oldest of the Rhodesian mines. The depth of the workings is now 2300 ft. For the six months ending June 30, 1908, the 40-stamp mill crushed 37,546 tons and obtained a gross profit of £28,475, excluding depreciation and London expenses, the yield and costs per ton being, respectively, 42s. 1d. and 26s. 11d. Battlefields is another dividend-paying mine. The Eldorado mine, working on a conglomerate or so-called banket ore, commenced crushing in September, 1907, and up to Oct. 31, 1908, treated 43,802 tons for a yield of £111,408. The estimated profit was £53,000. The Wanderer is a low-grade mine or quarry which dealt with 190,377 tons during the year ending April 30, 1908. The yield per ton was 8s. 8.83d., and the working costs, 6s. 5.7d.

GOLD PRODUCTION OF RHODESIA.

Year.	Oz. crude.	Year.	Oz. crude.	Year.	Oz. crude.
To Sept. 1, 1898	6,471	1902	194,169	1906	551,894
Sept. 1,'98–Dec. 31,'99	83,389	1903	231,872	1907	612,652
1900	91,940	1904	267,737	1908	(a) 594,407
1901	172,062	1905	407,096	Total to end of 1908	3,212,489

(a) Fine ounces.

Russia.—Gold mining in Russia appears to have become a fruitless occupation. As a result, the production remains practically at a standstill, although the resources are enormous. This was substantially the testimony of the district engineers of the gold mining region at the Congress of the Russian gold and platinum miners held in February and March, 1907. A long essay might be written upon the reasons for this situation. It is sufficient to summarize them by saying that the industry suffers from bad laws, bad management, and general incompetence, part of the latter being inherent in the population upon which the industry depends; such incompetence can be eradicated only after generations. Some bold British companies have become interested in Siberia. They will enjoy good management, but will suffer from the incompetence of the native population above referred to. Dredging has already been extensively introduced into Siberia, but so far this new method has been unable to swell the production.

Looking at the conditions more narrowly, the Semipalatinsk region in the southern Ural has promising lode mines, but capital for their development has not yet been available. Some lode mining enterprises in the Altai and Nerchinsk districts, of western Siberia, and the Trans-Baikal regions have gone into liquidation. As to placer mining, the conditions

in the Ural are favorable, but the ground is gradually being exhausted. In 1907, there were 25 dredges in operation in this district. In the once famous *taiga* of Yenisei, the production of gold has ceased except by the dredges, of which 38 were in operation in 1907. The Vitim and Olekminsk districts in the provinces of Yakutsk and Irkutsk require large capital, owing to the great depth of the overburden above the pay gravel and the frozen condition of the latter, together with the superabundance of water. The Lena Gold Company is said to have spent already in the neighborhood of 25,000,000 rubles in the development of its property. However, this is at present the most productive region of Russia, yielding about 35 per cent. on the total production. In the Amur and Priamursky provinces, gold mining is marked by a considerable development of single labor, but dredging and other improved methods are becoming more and more frequent. Scarcity of good labor is, however, a difficulty. In 1908, the Chukchi peninsula, in the far northeast, was opened for prospecting, which already is said to have been quite satisfactory. However, it is too early to speak about the importance of this new region. Also nothing definite can yet be said about the numerous discoveries of gold in the Trans-Baikal province, the Irkutsk province, and along the line of the Ussuri railway. A writer in the *Viestnik Finanssoff* in 1908 stated that very many of the gold deposits of the Priamursky district are not only unexploited, but are not even taken up, although gold is found over the whole area of the district. This is the youngest of the Russian gold mining districts, and it has the advantages of good waterways, a mild climate, and absence of perpetual frosts, from which the Amur province suffers

(By I. I. Rogovin.)—The gold production in Russia in 1908 increased. According to the official statistics gold was delivered, in terms of the fine metal, as shown in the accompanying table:

GOLD PRODUCTION OF RUSSIA.

District.	1907			1908		
	Poods.	Fine ounces.	Value.	Poods.	Fine ounces.	Value.
Ural Region	63.70	33,555	$ 693,582	87.90	46,292	$ 956,856
West Siberia	130.98	68,996	1,426,147	133.65	70,388	1,454,920
Irkutsk Province, including Bodaibo	874.06	460,412	9,516,716	987.95	520,316	10,754,932
Transbaikal and Amur	695.53	366,366	7,572,785	643.37	338,842	7,003,864
Total Government assay offices	1,764.27	929,329	$19,209,230	1,852.87	975,838	$20,170,572
Authorized private banks	449.88	236,975	4,898,273	707.88	372,811	7,706,003
Total official	2,214.15	1,166,304	$24,107,503	2,560.75	1,348,649	$27,876,575
Allowance for gold not reported	221.42	116,630	2,410,750	256.07	134,865	2,787,659
His Majesty's Domains	.....	.....	.....	25.33	13,562	280,327
Total	2,435.57	1,282,934	$26,518,253	2,842.15	1,497,076	$30,944,561

The amount of fine gold delivered to the Government laboratories for a series of years has been as follows: 1904, 1735 poods, 30 lb.; 1905, 1510 poods, 8 lb.; 1906, 1523 poods, 10 lb.; 1907, 1764 poods, 5lb.; 1908, 1852 poods, 33 lb.

The principal increase in 1908 was due to the district of the river Rodaibi in the Irkutsk government and the Yakutsk district, which is at present the most productive region and furnishes about one-third of the total amount of gold yielded in Russia. The Lena Gold Mining Company alone produced 520 poods in 1908, this being the most productive single undertaking in Russia. However, if the amount of sand washed be considered, it will be seen that the increase in 1908 was due to the unexpected improvement of the gold percentage. Of the other regions, the Priamursky showed an increase.

However, the economical results of gold mining industry in 1908 do not appear very successful. The rentability of gold mining undertakings has considerably decreased and many of them have turned out to be unprofitable. On looking closer to our gold mining we must point to three factors affecting it in a depressive way: (1) An exhaustion of gold fields; (2) an increase in the cost of labor, materials and provisions; (3) lack of capital. The first factor causes the working of poorer fields and the compensation of the lower percentage of gold in the sands by cheaper methods of extraction and working of same. The second leads to the same results as the first. Both of them lead to the intensification of this branch of industry. We see accordingly a considerable development of dredging in the Ural and in the Yenisei regions, where favorable conditions for the working of goldfields with hand labor exist. But unfortunately many circumstances have led to the failure of dredging in Russia. In 1908 there were in Russia about 65 dredges at work; 10 of them gave profit, but the remainder caused their proprietors more or less considerable losses. The principal causes of the failure of dredging in Russia are as follows: (1) Lack of culture and civilization, (total absence of roads, workshops, etc.); (2) insufficient prospecting of mines; (3) faults in the design of dredges, such as small capacity of buckets and defects in the washing appliances; (4) lack of sufficiently skilled labor, etc. The necessary intensification of the gold mining industry in Russia is heavily handicapped by lack of capital. The working of quartz gold in some regions which seem to have a great future requires also considerable capital. Consequently, in 1908, as well as in the foregoing years, there was a strong tendency to draw foreign, especially English, capital to Russian undertakings. But the failure of two or three Russian gold mining concerns which were not capitalized in a perfectly honest way on the English financial market has unduly affected the con-

fidence of English capital, and now even undoubtedly serious undertakings cannot find necessary capital abroad. The large increase of the yield of gold in 1908 in spite of all the above-mentioned unfavorable conditions shows that the quantity of gold in Russia is still enormous.

Santo Domingo.—The British vice-consul reports that veins of auriferous quartz are found all along the central mountain chain, or Gran Cordillera, the richest lodes being always in metamorphic rocks, near the crystalline rocks. Alluvial gold exists along the upper Jaina river, in the province of Santo Domingo. In the northern part of the island, that is, on the northern flank of the Cibao, alluvial gold is found in a number of places, especially in streams flowing from the Sierra de Cibao into the Yaque river. The Rio Verde and Sabaneta placers, in the Cibao region, have acquired a good reputation. The mines and placers known to exist in the province of Santiago are the Cerro de Piedra Blanca, in Paralimon; San José de las Matas, about one and a half miles from San José; El Pinar, San José de las Matas, on the road to Guava Cuano; Loma de la Mina, in the vicinity of the Magu river, with decomposed gold-bearing prophyry; Las Guasimas and La Cajera in San José de las Matas. The principal placers are along the Dicayagua, Bao and Yaque rivers.

Transvaal. (By W. Fischer Wilkinson.)—During 1908, the mining industry of the Transvaal expanded considerably, both as regards gross production and profits. Several new mines commenced crushing and many producing companies increased their reduction plant and output during the year. Comparing the month of December, 1908, with December, 1907, there was a net increase of 672 stamps in the Witwatersrand district. The tonnage crushed per stamp was considerably increased, due to increased weight and to the assistance given by tube mills. The statistics for these two months are given in an accompanying table. The

INCREASE OF MILLING IN THE WITWATERSRAND.

	Stamps at Work.	Tons Milled.	Tube Mills.	Tons per Stamp Month.
Dec., 1907	8,383	1,349,580	67	161
Dec., 1908.....	9,055	1,702,385	117	188

new companies contributing to this increase were the Aurora West United with 60 stamps, and the West Rand Consolidated Mines with 80 stamps, both of which companies made their first return during September. Of the old companies the Ferreira Deep added during the year 40

stamps to its reduction plant; the Durban-Roodepoort Deep, 40 stamps; and the New Kleinfontein, 20 stamps. In October, the production was still further increased by the entry into the list of producing mines of the Jupiter and Simmer Deep, which have a joint 300-stamp mill, and of the Cinderella Deep, which has a 50-stamp mill. In November, the Geduld company, which also has a mill of 50 stamps, made its first return.

Adding the production of the outside mines, the total gold production of the Transvaal for 1908 had a value of £29,957,610, which is a record production. Large as the figures for 1908 are, a further expansion in the early months of 1909 may be confidently awaited, as there are several companies with reduction plants in course of construction. Among these may be mentioned the Knights Central, which will start with a mill of 100 stamps, the Vogelstruis Consolidated Deep with a mill of 40 stamps, and the Bantjes with 35 stamps. The only company that retired during 1908 was the Bonanza, a small property which had a life of exceptional prosperity. The mill was bought by the Robinson Central Deep mine and the stamps are now crushing ore from that property. There are two other mines nearing exhaustion, namely the Champ d'Or and the Jubilee. It may be remarked that when mines are worked out, the mills are still available to crush ore from other mines, so that the productive power of the field as a whole does not necessarily suffer from the exhaustion of a mine.

The most noticeable development of new ground that took place during 1908 was in the eastern section of the Rand. The successful results obtained by the New Kleinfontein, the Van Ryn, and the New Modderfontein mines, which are the most eastern producing mines, led to increased activity in this neighborhood, and the Main Reef series may now be regarded as proved without doubt as a payable ore deposit as far as Geduld. The extension of the reef farther east or south is still something of a problem; but that it will be traced farther in the near future seems probable. The Witwatersrand beds are in this district covered by a younger formation, so that the exact position of the reefs and of the lay of the beds has to be determined by bore holes. The theory of the extension of the Rand formation most generally accepted is that the Main Reef takes a turn to the south, running toward the Nigel district. While there is some uncertainty as to its course after leaving Geduld, all doubt as to its position and value as far as that farm has now been cleared up. The Geduld mine is now one of the crushing mines, and active development is proceeding on the Van Dyk, the Brakpan, the Rand Collieries, and the Apex.

Below Modderfontein, a new company called the Modderfontein B. Gold Mines, Ltd., was recently formed to amalgamate into one block of 1467 claims, of which 67 are north of the outcrop, the claims of the Modderfontein Extension, Ltd., North Eastern Claims, Ltd., Transvaal Land and Mines Proprietary Syndicate, Ltd., and South Eastern Claims, Ltd.

TRANSVAAL GOLD PRODUCTION BY YEARS.
(Chamber of Mines Returns.)

Year.	Witwatersrand District.			Outside Mines Value.	Transvaal Total.
	Tons Milled.	Value.	Value per Ton Milled.		
		£	s.	£	£
1884-9	1,000,000	2,440,000	48.8	238,231	2,678,231
1890	730,350	1,735,491	47.4	134,154	1,869,645
1891	1,154,144	2,556,328	44.2	367,977	2,924,305
1892	1,979,354	4,297,610	43.4	243,461	4,541,071
1893	2,203,704	5,187,206	47.0	293,292	5,480,498
1894	2,830,885	6,963,100	49.2	704,052	7,667,152
1895	3,456,575	7,840,770	45.2	728,776	8,569,555
1896	4,011,697	7,864,341	39.2	739,480	8,603,821
1897	5,325,355	10,583,616	39.74	1,070,109	11,653,725
1898	7,331,446	15,141,376	41.3	1,099,254	16,240,630
1899	6,872,750	15,067,473	43.84	661,220	15,728,693
1900	459,018	1,510,131	65.82	.....	1,510,131
1901	412,006	1,014,687	49.25	81,364	1,096,051
1902	3,416,813	7,179,074	42.00	74,591	7,253,665
1903	6,105,016	12,146,307	39.79	442,941	12,589,248
1904	8,058,295	15,539,219	38.46	515,590	16,054,809
1905	11,160,422	19,991,658	35.82	810,416	20,802,074
1906	13,571,554	23,615,400	34.80	964,587	24,579,987
1907	15,523,229	26,421,837	34.04	981,901	27,403,738
1908	18,196,589	28,810,393	31.6	1,147,217	29,957,610

This company is expected to have the reef in no place at a greater depth than 2000 ft. vertical, and the active development of the property will shortly be commenced. On Daggafontein, another farm in this district, boring operations located the reef and a shaft is projected; and on Cloverfield the sinking of a shaft is also being taken in hand.

Although none of the developing mines in the far eastern Rand has been opened yet on a large scale, sufficient work has been done to demonstrate that good ore occurs and that the district will in the future yield a large gold production. Except for the break caused by the Witpoortje fault there is now a practically unbroken line of mines from Randfontein to Geduld, a distance measured along the line of outcrop of about 50 miles. There are a few gaps here and there in this line where there are no producing mines. Between Randfontein and the West Rand Central there is a stretch not yet developed, and in the West Rand a small length of outcrop belonging to the Roodepoort Gold Mining Company is standing idle. But this will shortly again be worked, an amalgamation with the Roodepoort United Main Reef having been announced in December. In the eastern Rand there is an unworked piece between what was the Cason (now part of the East Rand Proprietary Company) and the New

Kleinfontein. With these exceptions, the Main Reef beds have been exposed and developed over the whole distance from Randfontein to Geduld.

The decision of the Transvaal government to terminate the employment of Chinese at the mines was referred to in last year's review. At the close of 1907 there were 37,118 Chinese employed, which number was reduced to 12,275 at the end of 1908. Fortunately the industrial position was not seriously affected by this large reduction of labor, owing to the increased supply of Kafirs. It is partly due to more successful recruiting and partly to a decreased demand for this labor in other parts of South Africa, notably at the diamond mines, that the Rand has not experienced a labor famine as might have been expected. On the contrary, for the greater part of 1908 the demand for unskilled labor was easily met.

It is satisfactory to be able to report that the efficiency of the native has undergone improvement, largely brought about, it is claimed, by the good example set by the Chinese. The standard of work, too, has been raised, at all events, at some of the mines. At the Simmer & Jack, for example, a 48-in. hole is now expected from hammerboys in place of a 36-in. hole, and, in the case of machine drilling, an 8-ft. hole instead of a 6-ft. hole. Greater efficiency in unskilled labor was also obtained by extending the period of engagement of natives to 12 months instead of the shorter periods which have previously been customary.

While there is no great difficulty at the present time in obtaining a sufficient supply of unskilled labor, anxiety has been expressed that in consequence of the great expansion in the mining industry that is now taking place, the labor question may again become a serious problem. At a meeting of the Knights Deep mine, held in November, the chairman, a manager of one of the larger financial houses, said that the labor question was only temporarily solved, but he expressed the hope that an adequate supply of native labor would be obtained from Central Africa and from Zululand and the neighboring territories, from which districts so far only a small number of natives have been drawn. There is also reason to hope that a further economy of labor may be obtained by substituting machine drilling for hand drilling, a change which is dependent on the discovery of a thoroughly reliable machine drill.

Economy in native labor is also obtained by employing unskilled whites and white apprentices on an increased scale. In some of the mills, for instance, natives are excluded, their places being filled by young white men. The extent to which unskilled white labor can be employed is regulated by considerations of cost. The low rate of native wages pre-

vents the white man from competing successfully with the Kafir for the lowest class of manual labor; but for work where intelligence counts, it is different. Here, owing to increased efficiency, he can enter into successful competition with the Kafir, and it is satisfactory to find that those who control the mines are encouraging this competition, and are giving facilities to young men of the country to make themselves efficient miners, enabling them to find employment in positions which were for-

PERSONS EMPLOYED IN RAND GOLD MINES AT END OF MONTH.

	White.	Colored.	Chinese.		White.	Colored.	Chinese.
1902—July	8,162	32,616	.....	1906—June	17,959	90,882	52,352
December	10,292	45,698	.....	December	17,495	98,156	52,917
1903—June	11,825	66,221	.....	1907—June	17,166	111,962	57,517
December	12,695	73,558	.....	December	17,697	129,618	37,118
1904—June	13,413	74,632	1,004	1908—June	18,181	147,557	21,460
December	15,023	83,639	20,885	December	19,605	164,826	12,275
1905—June	16,939	104,902	41,340				
December	18,159	93,831	47,267				

merly occupied by natives. The total labor employed in all mines of the Transvaal, that is, including gold, coal, diamond and base metal mines, during December, 1908, was: Whites, 21,277; colored labor, 185,453; Chinese, 12,275.

During 1908 a number of amalgamations of companies were carried through. From one end of the Rand to the other the policy of creating larger units than formerly has been adopted. Whereas the standard size for a mining block used to be 200 claims, the area now selected as a unit is 1000 claims and more. In support of the adoption of this policy, it is urged that larger units produce economy in capital expenditure and a reduction in working costs, principles which will not be contested. The differences of opinion that have arisen have not been so much on the question of the size of a mine area, as on the valuation placed on the different interests included in the amalgamation, and generally on the question whether increased area and increased life, even though accompanied by a reduced cost of working, were worth the price. Many shareholders in a producing mine sufficiently developed to allow of there being no uncertainty as to its future, prefer their property to be worked out to a finish, rather than have a speculative piece of additional ground added at a considerable increase to their capital and consequent reduction in their dividend.

Although economy in working has been the main reason put forward in support of consolidation and amalgamation, the policy has also been largely influenced by financial considerations. In some cases the deep level areas, which have been amalgamated with producing mines immediately above them, have belonged to companies which had either exhausted

their working capital, or had not sufficient funds to bring them to the producing stage. The mining houses who controlled them expressed their inability to obtain the necessary capital from the shareholders, or the public, and were not prepared to finance these companies from their own funds. The Simmer Deep, the City Deep, the Knights Central, the Jupiter, and the East Rand amalgamations may be quoted as examples of consolidation brought about largely for financial reasons. A later example is the proposed amalgamation of the South Nourse with the Nourse Mines, the latter a company which had not long before absorbed the Henry Nourse, an amalgamation from which the outcrop mine has suffered.

The position at the commencement of 1908 was that many deep-level companies were in a desperate financial condition. Their funds were exhausted, or insufficient to bring them to the producing stage. Fresh capital was wanted, and was not obtainable, and it was evident that even if fresh capital were obtained it would be a long time before these properties would be able to pay dividends. The adoption of the policy of amalgamation saved the situation. By consolidation with the producing mines the necessity of raising fresh capital was avoided. The shareholders in the deep mines became partners in the dividends of the producing mines, giving in return their ground and whatever cash assets they happened to possess. The producing mines received as compensation for the increase to their capital the prospect of an extended life and were enabled with the funds supplied by the deep mines to work on a larger scale and consequently at a reduced cost of working.

The arguments used in favor of amalgamation mostly employed have been the longer life and reduced working expenses which the working of larger areas brings about. There can be no question that the working of larger units tends to economy. There will be obviously a saving in administration expenses, while economy will be gained in having larger power plants. Fewer shafts are required, and the development of a mine can be carried out more advantageously with a large than with a small area. The statistics of the working costs of the mines, which will be referred to presently, demonstrate the advantage of large units.

In principle, therefore, amalgamation into large units is sound. The question whether amalgamation or consolidation is advantageous or not depends on what valuation is put upon the various interests. And it is on this point that differences of opinion as to the justice of the amalgamations that have been carried through have arisen. Valuations are governed by many factors, some of which are arbitrary, such as the scale of operations, capital expenditure, rates of interest assumed in calculating the present value of profits, so that it is not surprising that con-

flict of opinion should arise as to the fairness of any scheme of amalgamation. The question is, what is the most economical life of a mine, and the relation between capital expenditure and profits has been the subject of much discussion, and prominent engineers, such as Mr. Kötze and Ross Browne, have shown that the economic life of a given area comes inside 10 years. If that is so, the argument of long life, which deep-level interests put so much weight upon, has to be treated with caution by the producing mines. They have to decide whether the present value of the profit to be secured from the deep ground is worth the price demanded, taking into consideration the decreased cost of working that the amalgamation will bring about owing to the increased scale of operations, a result which, it may be noticed, can be secured without amalgamation by devoting some of the profits to extension of plant.

From what has been said, it will be understood that the amalgamation schemes require very careful valuation work, and it is only fair to say that the Rand mining houses have, at all events, in all the most important consolidations that have been carried through, employed competent engineers to value the various interests involved. The undesirable feature in these amalgamations is that the sellers are those who practically decide the price. Valuations are made by engineers, it is true, but if the valuation does not agree with the views of the financial house, the business may be dropped. In several cases the complaint has been made that the shareholders in the producing and dividend-paying mines have been forced to become partners in ground of unproved and speculative value without any real necessity. Unamalgamated, such mines had prospects of a certain, though limited, prosperous career. The Henry Nourse, the Village Deep, and the East Rand producing companies could all have been worked to a finish without amalgamation. The shareholders in these mines hear of the advantages to be gained by amalgamation, but not the disadvantages, and probably if more prominence had been given to the disadvantages, the shareholders in the dividend-paying mines would have refused acceptance of the respective schemes of amalgamation. Opposers of a scheme have not, however, the same chance of making themselves heard as the promoters, and consequently the financial house, although perhaps representing a minority interest in the producing mine, generally wins the day.

This may be a convenient place in which to notice the methods employed to restart some of the mines whose working capital has been exhausted. The usual practice is now for the capital to be first reduced by giving the old shareholders one share for each two, three or four held before. The capital is then immediately raised and the fresh capital is offered *pro rata* to the shareholders with options over further shares at varying

prices and payable at varying dates, the controlling house taking options over some of the shares. Examples of this form of reconstruction are the Bantjes and Vogelstruis Consolidated Deep.

It seems a pity that the finance of mining companies cannot be conducted on simpler lines and that the complicated system of reducing capital and then immediately raising it, coupled with all sorts of option conditions, cannot be avoided. These complicated methods of finance, approaching almost to juggling, are exceedingly puzzling and unpopular with the ordinary shareholder.

Another matter connected with Rand finance that seems worth calling attention to, as it is something of an innovation, is the building up of a reserve fund out of the profits and its investment in a new undertaking. The Jumpers mine is an example of this. This mine is nearing exhaustion, but the profits are not being fully distributed to the shareholders, but are being reinvested in another mine in the east Rand, viz., the Chimes West and Benoni. This practice of investing profits in outside

LIST OF RECENT AMALGAMATIONS ON THE RAND.

Date of Amalgamation.	Name of Enlarged Company.	Companies Absorbed.
1906	Simmer Deep: 1034 claims	South Geldenhuis Deep. South Rose Deep. Rand Victoria Mines. Rand Victoria East.
1907	Jupiter: 500 claims	Simmer West.
1907	West Rand Consolidated: 1785 claims	West Rand Mines. Violet.
1907	Randfontein Central: 477 claims	Randfontein Block A. Mynpadit Randfontein. West Randfontein.
1908	East Rand Proprietary Mines: 2896 claims	Dreifontein. Angelo. Comet, Cason, H. F. Company, Angelo Deep. Dreifontein Deep (part of). Hercules, and sundry claims.
1908	Consolidated Langlaagte: 501 reef claims	Croesus Deep.
1908	Knights Central: 893 claims	South Knights.
1908	City Deep: 1084 claims and farm.	City Deep. South City. South Wolhuter. Wolhuter Deep. Farm Klipriversberg.
1908	Village Deep: 529 claims	Turf Mines.
1908	Witwatersrand Deep	Dreifontein Deep: 125 claims.
1908	Chimes West	Benoni.
1908	Van Ryn Deep: 754 claims	Klipfontein Deep.
1908	Modderfontein B. Gold Mines	Modderfontein Extension. Northeastern Claims. Transvaal Lands and Mines. Proprietary Syndicate. Southeastern Claim.
1908	Nourse Mines	South Nourse.
1908	Roodepoort Gold	Roodepoort United. Main Reef.
1908	Crown Reef: 1852 claims (a)	Crown Deep. Crown Reef. Paarl Central. Langlaagte Deep. Robinson Central Deep. South Rand and Central Deeps.

(a) Amalgamation not yet completed.

properties, as well as the practice now largely prevalent of increasing the reduction plants out of profits instead of by an issue of new capital, is evidence that the Rand mining houses, having failed to get money from the public for new enterprises or for extensions, have adopted a policy of making the shareholders in the dividend-paying mines find the money.

The report of the Mining Industry Commission was issued in March. The Commission had been appointed by the Government to inquire in

what direction, and how far, the use of mechanical appliances could be extended on the mines, and what effect such extended use would have upon (a) the employment of white labor, (b) the cost of production, (c) the demands of the mining industry for native labor and (d) on the mining industry generally. The commissioners were also asked to report on the means best calculated (a) to increase the employment of white labor on the mines, (b) to secure the more efficient use of native labor and (c) generally to secure greater economy and efficiency in mining operations. The commissioners were not unanimous in their findings and there was a majority report representing the views of F. H. P. Creswell, a prominent advocate of all-white labor for the mines, and a minority report, that expressed the views of the average mining man on the Rand, which are that the present condition of things does not call for any drastic changes by legislation. The Government appears to share this opinion. At all events no action has been taken to carry out the principal recommendations of the majority report.

DIVIDENDS OF GOLD MINES OF THE TRANSVAAL.

Year.	£	Year.	£	Year.	£	Year.	£
1887.....	12,976	1893.....	955,358	1899(a)...	2,946,358	1905.....	4,857,539
1888.....	112,802	1894.....	1,527,284	1900(a)...	.....	1906.....	5,735,161
1889.....	432,541	1895.....	2,046,852	1901(a)...	415,813	1907.....	7,131,612
1890.....	254,551	1896.....	1,513,682	1902(a)...	2,121,126	1908.....	8,751,282
1891.....	334,698	1897.....	2,707,181	1903.....	3,345,502		
1892.....	879,320	1898.....	4,848,238	1904.....	3,927,830		

(a) War period, Oct. 11, 1899, to May 31, 1902.

During the latter part of 1907 a competition for machine-stope drills, promoted by the *South African Mining Journal,* was held, and excited much interest. Eight different makes of drill entered, the prize being awarded to the Gordon drill. The advantages to be gained from such trials are recognized by the mines, and a further competition on a larger scale is being organized by the Chamber of Mines to take place about April, 1909. The competition is expected to last more than six months, so that a thorough testing of the competing drills may be made. The object of the competition is to obtain a small drill, weighing less than 100 lb., that can be used in narrow stopes. The first prize of £4000 will be awarded to that drill for which the total cost, when divided by the total footage drilled by all machines of that entry during the competition, is least. There will be a second prize of £1000, besides special prizes for the drillmen. Having regard to the great saving in native labor that the discovery of a suitable drill would bring about, the competition promises to be of great importance to the industry.

A new gold law, gazetted on Jan. 1, 1909, although not entirely satisfactory from the point of view of the mining community, is regarded

as an improvement on the old law. It is more liberal than the former law as regards prospecting. The principle of open prospecting has been agreed to by the Government, but its application is limited to government land and private farms not beneficially occupied.

The dividends paid by mining companies totalled £8,751,282, of which £214,509 was contributed by three mines in the outside districts.

In metallurgical work several improvements appeared during 1908. The new stamp batteries are being equipped with heavier stamps than formerly, the latest mills having stamps weighing 1650 and 1700 lb. Appliances are being put in for dewatering sands. A 15-ft. rotary sand filter, after the designs of W. A. Caldecott, is now running at the Simmer & Jack, and others are to go up at the new Simmer Deep plant. Attention is being paid to the removal from tailings pulp by a magnetic separator of metallic iron, which otherwise accumulates in the tube mill. At the Robinson Deep a magnetic conveyer belt pulley is going in to pick out broken drill ends, etc., so as to keep them out of the mortar boxes. At the Bantjes mill the plates are to be placed in a separate building, while an outside stockpile and belt system will allow of reduced bin construction. Warm solutions for slime treatment are doing well, and the system will probably be extended. At the Simmer Deep plant a large vacuum air pump, drawing 420 cu.ft. per min. has been installed and connected up to the sand-leaching pipes, so as to drain charges thoroughly and draw large volumes of air through the sand. Electric stamp driving is carried on at some mills, some of the mines obtaining their power from their own central station while others purchase from an independent supply company. H. C. Behr's dense-air system is at work at the Simmer Deep mine. The exhaust of a winding engine worked by air at a very high pressure is delivered directly into the air mains for operating machine drills. At some of the mines centrifugal pumps are being put in to do the work of the large tailings elevator wheels which are so conspicuous all along the reef.

During 1908 there was a general reduction in the cost of working. The statistics of the mines, however, probably exaggerate the improvement in the economic conditions. In comparing, for instance, the working costs of any mine with those of previous years, it will be noticed that in many cases part of the improvement is due to charging a lower amount than formerly for development redemption. The average grade of ore worked has been lowered and consequently a larger tonnage is available for mining than formerly for the same cost of development, thus automatically reducing the cost per ton. It is important to remember, in considering the industrial situation, that the ton of today is not the ton that was mined last year or in former years. Moreover, part of the

reduction shown in the statistics is due to the increased scale of operations, the fixed charges, for example, being proportionately less per ton. Making allowance, however, for these points, which have made a reduction in costs more apparent than real, there has no doubt been a general reduction in the costs due to increased efficiency, and a reduction in the average white wage. The policy now generally adopted of working on a large scale and of mining ore down to a minimum of profit certainly produces very low costs. Whether equally good results or better could not be obtained by raising the grade is an open question upon which the *South African Mining Journal* published during the early months of 1908 an interesting series of articles. The accompanying table, compiled from the monthly statistics of the Chamber of Mines, shows the reduction in costs that has taken place between January and December.

WITWATERSRAND MINES.

1908	Tons milled.	Yield excl. Gold from Accumulations.	Working Costs.	Working Profit.	No. of Companies.
		s.	s.d.	s.d.	
January.....	1,416,139	31.91	19.5	12.7	62
February.....	1,343,672	32.43	19.5	13.1	63
March.....	1,456,375	31.57	18.8	13.3	63
April.....	1,418,687	32.06	18.6	13.7	63
May.....	1,490,952	31.37	18.1	13.5	63
June.....	1,458,499	31.52	18.0	13.8	63
July.....	1,515,505	30.96	17.9	13.5	60
August.....	1,535,793	30.8	17.4	13.7	60
September....	1,534,755	30.67	17.5	13.5	63
October.....	1,672,895	30.43	17.3	13.4	63
November....	1,650,932	29.99	17.2	13.	66
December	1,702,385	(a)29.46	17.1	(a)12.7	67
Total	18,196,589	Average 31.39	18.	13.5	

(a) Excluding £154,492 gold reserve declared in December.

There is generally a wide difference between the working profits and the dividends, owing to deductions for profit tax, depreciation, interest on loans, debenture interest, etc. On an average the dividends are about 70 per cent. of the working profits. The accompanying table, compiled

COSTS AND DIVIDENDS.

Year.	Value per ton.	Costs per ton.(a)	Dividend per ton.	Year.	Value per ton.	Costs per ton.(a)	Dividend per ton.
	s.	s.	s.		s.	s.	s.
1897.....	39.7	29.6	10.2	1905	35.8	27.2	8.5
1898.....	41.3	28.0	13.0	1906.....	34.8	26.4	8.2
1899.....	43.8	34.8	8.9	1907.....	34.0	25.0	9.0
1903.....	39.8	28.8	11.0	1908.....	31.6	22.2	9.4
1904.....	38.5	29.0	9.5				

(a) Taken as value minus dividend.

from the returns of the Witwatersrand mines only, i.e., exclusive of the outside mines, illustrates the reduction in grade that has taken place

since 1897, omitting war years, and gives the dividend profit per ton and the cost of working calculated as the difference between dividend and yield. In order to illustrate the effect of large outputs in reducing costs, another table has been compiled from the returns for the month of August, 1908. The Randfontein Mines are omitted. The seven largest mines are the Crown Deep, the Robinson, the Village Main Reef, the Knights Deep, the Robinson Deep, the Simmer & Jack, and the East Rand Proprietary, the latter a group of mines consolidated into one company.

WITWATERSRAND MINES.
Returns for August, 1908.

Tons milled per month.	No. of Cos.	Yield s.	Costs s.	Profit. s.
Under 10,000 tons	10	33.4	23.4	10.0
10,000 to 20,000	13	29.6	20.0	9.6
20,000 to 30,000	13	28.8	17.0	11.8
30,000 to 40,000	10	31.0	17.0	14.0
40,000 and upward	7	32.6	14.8	17.8

Another table has been compiled from the statistics of the Rand Mines group to illustrate how reduced costs are obtained at the expense of grade. In four cases the reduction in costs is overbalanced by the reduction in yield. While the net profit per ton is less in some cases, it must be remembered that the total available ore supplies of the mine are increased by the lowering of the grade. The figures are instructive and indicate that it is uneconomical to lower costs too far at the expense of grade.

REDUCED COSTS OBTAINED AT EXPENSE OF GRADE.

Mine.	Year ending.	Working costs. s. d.	Quarter ending.	Working costs. s. d.	Yield per ton milled.		Reduction in costs. s. d.	Reduction in grade.	
					1907 dwts.	1908 dwts.		dwts.	s. d.
Glen Deep	July, '07	20.11.9	July, '08	19. 9.3	7.439	7.000	1. 2.6	0.439	1.10.1
Geldenhuis Deep	Dec., '07	19. 8.5	Sept., '08	18.10.9	7.085	6.712	0. 9.6	0.373	1. 6.8
Jumpers Deep	Sept., '07	24. 6.5	Sept., '08	23. 8.7	6.905	6.659	0. 9.8	0.246	1. 0.3
Nourse Mines	July, '07	23.11.2	July, '08	20. 7.	8.453	7.371	3. 4.2	1.082	4. 6.5
Rose Deep	Dec., '07	18. 8.4	Sept., '08	16. 3.7	6.563	6.266	2. 4.7	0.297	1. 2.9
Ferreira Deep (a)	Sept., '07	19. 3.6	Sept., '08	16. 5.4	11.697	11.030	2.10.2	0.667	2. 9.6
Crown Deep	Dec., '07	19. 2.9	Sept., '08	14.11.	9.226	8.240	4. 4.	0.986	4. 1.6
Langlaagte Deep (b)	July, '07	23. 0	July, '08	18. 1.3	7.062	6.350	4.10.7	0.712	2.11.8
Durban Rpt. Deep	Dec., '07	26. 3.3	Sept., '08	24. 6.7	7.681	7.424	1. 8.6	0.257	1. 9.5

(a) 120 stamps in 1907, 160 in 1908.
(b) 170 stamps in 1907, 200 in 1908.

The mining industry of the Transvaal at the close of 1908 was in a healthy condition and promises to increase in prosperity in the next few years. The improved results obtained have enabled many of the mines to pay off debts and debentures and a larger number will in the future

figure in the dividend list. The policy of working on a large scale has added immensely to the value of the gold field. Owing to the reduction in costs that results from large outputs many millions of tons and many millions of ounces of gold hitherto worthless have been brought within the bounds of profitable working. Mining is now contemplated over an area and at depths undreamt of a few years ago, making it difficult to put limits on the gold resources of the Colony. The native labor supply may not keep pace with the demands that the increased scale of operations projected may make upon it, but there can be little doubt that the production of gold, large as it is at present, will in the next few years increase to still larger dimensions.

THE LONDON SILVER MARKET IN 1908.

The following review of the market in London, where most of the silver produced in the world is handled, was made by Messrs. Pixley & Abell, of that city, at the beginning of 1909:

The heavy fall in silver, which took place toward the close of 1907, was followed during the opening months of 1908 by a reaction, due chiefly to large purchases for the Indian bazaars, and during January the quotation rose to 27d., the highest of the year, an advance of 2⅛d. from the closing price of 1907. This steadiness continued until the beginning of April, when weakness again set in, and from that time the tendency of the market, with the exception of a temporary advance in June, was downward until Dec. 2, when 22d., the lowest quotation of the year, was touched. From this point the market steadily improved until the end of the year, when the closing quotations were 23 7/8d. for spot and 23⅛d. for forward silver. Important purchases for China and some buying for India, where the position had become more satisfactory, accounted for this sharp advance during the last few weeks. The average price for the whole year was 23 13/32d. per ounce.

Shipments to the Indian bazaars were throughout 1908 on an unusually large scale. The net imports, chiefly from Great Britain, into India probably amounted to 70,000,000 oz. against 44,500,000 in 1907, and 32,750,000 in 1906. This demand may to a great extent be attributed to a desire on the part of the natives, attracted by the low prices, to convert their hoards of rupees into bar silver, and this action has doubtless been partially responsible for the increase of rupees in the Indian treasury. There was an unusual number of marriages during the early part of 1908, as in July commenced a period (which will extend until October, 1909) that is considered by the Hindoos to be unpropitious for matrimony, and this fact doubtless stimulated purchases during the earlier months. The cheapness of silver also probably conduced to the purchase of this

metal instead of gold, of which only about £3,100,000 in bars was sent to India in 1908, against £5,700,000 in 1907, the difference representing in silver some 25,000,000 ounces.

Indian speculators during the last six months operated largely both in London market and Bombay. Though these speculative accounts were not liquidated at the end of the year, the immediate result was to transfer large amounts of silver from the London market to Bombay, where, in spite of some substantial purchases for China, the stocks at the end of 1908 were still heavy, amounting, it was estimated, to nearly £400,000, while in addition there was £840,000 in transit.

The Indian Government made no purchases during 1908. Its holdings of rupees largely increased and amounted at the beginning of December, when the highest point was reached, to about 29 crores in the paper currency reserve and to 18½ crores in the gold standard reserve. Since then, owing to the opening of the export season, the paper currency reserve fell to just under 28 crores (£18,500,000), and the gold standard reserve to 18 crores (£12,000,000). At the commencement of 1908 the government's holdings were 21 crores (£14,000,000) in the paper currency reserve and 6 crores (£4,000,000) in that of the gold standard. Under these conditions the prospects of further purchases during 1909 are rather remote. To coin one crore of rupees requires 3,437,500 fine ounces of silver. As 15 rupees equal £1, a crore of rupees equals £666,666⅔.

Though China was at times a seller in 1908, and shipped about £220,000 in sycee to London for sale, yet she was, on the whole, a considerable buyer. The low exchanges ruling in China greatly stimulated the export trade, and adversely affected the import trade so that the banks were forced to buy silver as cover for or as a hedge against their exchange operations. Part of these purchases were shipped to China, but a large amount remained under the control of the China banks. Whether this will be resold or shipped to China remains to be seen. In addition to her purchases in London, China bought fair amounts in Bombay and also received regular shipments, amounting to about £1,000,000, from San Francisco. An important feature in the future will be the effect of the prohibition of the import of opium into China, should this result from the recent efforts to suppress the use of this drug. Unless the import of other commodities should greatly increase, it would seem probable that China will be a still more important buyer of silver than heretofore. In Shanghai the stock of sycee is unusually heavy, amounting to about 19,000,000 taels, while in additon there are 10,000,000 Mexican dollars.

A convention was signed in November by the members of the Latin Union whereby it has been arranged to increase the circulation of sub-

sidiary coins from 12 to 16 francs per head of the population. This convention comes into force on April 1, 1909, and the increase is to be made at the rate of 60 centimes per head per annum. There are also provisions to allow Italy and Greece, where the circulation is now below 60 centimes per head, to bring the circulation up to that amount. As a large part of the silver required will be provided by recoining five-franc pieces, or coins of similar value, the amount of new silver required will be unimportant. Excepting 5000 kg. bought during December, the Paris Mint made no purchases by tender in 1908. Silver, however, was bought for French account during the earlier part of the year, and this presumably was required for colonial coinage. Switzerland was a moderate buyer at times, while Portugal bought £250,000 in October. In Germany a bill passed the Reichstag for the purpose of increasing the silver currency of that country by five marks per head during the next five years. This is estimated to require about 1,250.000 kg. of silver; but so far, we understand no purchases have been made. During November, the Mexican Government offered to buy 5000 kg. monthly for currency purposes.

With silver at low level—almost down to that of 1902-3, when 21 1/16d. was the lowest quotation touched for spot silver—it becomes interesting to review the present situation and consider the prospects for the future. Though the Indian Government is unlikely to be a buyer for some time to come, and though China has unusually large holdings of silver, the larger offtake by the Indian bazaars and the possible falling off in production will, in all probability, suffice to prevent any further material decline. Much, however, will depend upon whether China resells any of the silver under her control in London, or whether fresh purchases are made on her behalf.

Sharps & Wilkins, of London, report as follows: In August speculation in India became more active, and large amounts of silver for forward delivery were continually pressed for sale, causing the market to weaken considerably, and the prices fell to a new level of about 1d. per oz. below those formerly current. The quotation, however, was fairly well maintained until it became evident that the stocks in the bazaar had become excessive, owing to an attempt which was made to corner the market in Bombay for the mid-November settlement, which result was prevented by the sellers in India ordering out large totals to counteract the attempted upward movement. The price in Bombay then gave way suddenly, and the market in India found itself with such an accumulation of bars that orders for shipment no longer came forward, and the cessation of buying weakened the quotation to 22 on Dec. 2, the lowest price of the year, within 5/16 per oz. of the minimum on record—21 1/16 in Novem-

ber, 1903. The final fall was acclerated by the weakness of the China exchanges and consequent sales from Hong Kong and Shanghai.

Mocatta & Goldsmid report as follows: The relative price between spot and forward has been interesting, cash being at a discount during the early part of the year, the difference at one time being as much as ¼ per oz. This, however, gradually narrowed, and the two quotations became practically identical in May, and continued so till October, when spot silver became scarce and commanded a premium, for the first time since

AVERAGE PRICE OF BAR SILVER IN LONDON, 1833-1902.
In pence per standard ounce, 0.925 fine.

Year.	Pence.	Year	Pence.	Year	Pence.	Year.	Pence.	Year.	Pence.	Year.	Pence.	Year.	Pence.
1833	59.1875	1843	59.1875	1853	61.5000	1863	61.3750	1873	59.2500	1883	50.5625	1893	35.6250
1834	59.9375	1844	59.5000	1854	61.5000	1864	61.3750	1874	58.3125	1884	50.6250	1894	28.9375
1835	59.6875	1845	59.2500	1855	61.3125	1865	61.0625	1875	56.8750	1885	48.6250	1895	29.8750
1836	60.0000	1846	59.3125	1856	61.3125	1866	61.1250	1876	52.7500	1886	45.3750	1896	30.7500
1837	59.5625	1847	59.6875	1857	61.7500	1867	60.5625	1877	54.8125	1887	44.6250	1897	27.5625
1838	59.5000	1848	59.5000	1858	61.3125	1868	60.5000	1878	52.5625	1888	42.8750	1898	26.4375
1839	60.3750	1849	59.7500	1859	62.0625	1869	60.4375	1879	51.2500	1889	42.6875	1899	27.4375
1840	60.3750	1850	60.0625	1860	61.6875	1870	60.5625	1880	52.2500	1890	47.6875	1900	28.2500
1841	60.0625	1851	61.0000	1861	60.8125	1871	60.5000	1881	51.6875	1891	45.0625	1901	27.1875
1842	59.4375	1852	60.5000	1862	61.4375	1872	60.3125	1882	51.6250	1892	39.8125	1902	24.0900

AVERAGE PRICE OF SILVER.
New York, in cents per fine ounce.

Year.	Jan.	Feb.	March.	April.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
1902....	55.56	55.09	54.23	52.72	51.31	52.36	52.88	52.52	51.52	50.57	49.07	48.03	52 16
1903....	47.57	47.89	48.72	50.56	54.11	52.86	53.92	55.36	58.00	60.36	58.11	55.375	53.57
1904....	57.055	57.592	56.741	54.202	55.430	55.673	58.095	57.806	57.120	57.923	58.453	60.563	57.231
1905....	60.690	61.023	58.046	56.600	57.832	58.428	58.915	60.259	61.695	62.034	63.849	64.850	60.352
1906....	65.288	66.108	64.597	64.765	66.976	65.394	65.105	65.949	67.927	69.523	70.813	69.050	66.791
1907....	68.673	68.835	67.579	65.482	65.981	67.090	68.144	68.745	67.792	62.435	58.677	54.565	65.327
1908....	55.678	56.000	55.365	54.506	52.795	53.663	53.115	51.683	51.720	51.431	49.647	48.769	52.864

London, in pence per standard ounce, 0.925 fine.

1902. ..	25.62	25.41	25.00	24.34	23.71	24.17	24.38	24.23	23.88	23.40	22.70	22.21	24.09
1903....	21.98	22.11	22.49	23.38	24.89	24.29	24.86	25.63	26.75	27.89	27.01	25.73	24.75
1904....	26.423	26.665	26.164	24.974	25.578	25.644	26.760	26.591	26.349	26.760	26.952	27.950	26.399
1905....	27.930	28.047	26.794	26.108	26.664	26.910	27.163	27.822	28.528	28.637	29.493	29.977	27.839
1906....	30.113	30.464	29.854	29.984	30.968	30.185	30.113	30.529	31.483	32.148	32.671	32.003	30.868
1907....	31.769	31.852	31.325	30.253	30.471	30.893	31.866	31.637	31.313	28.863	27.154	25.362	30.188
1908....	25.738	25.855	25.570	25.133	24.377	24.760	24.514	23.858	23.877	23.725	22.933	22.493	24.402

July, 1907. Forward purchases on China account have lately, in spite of heavy "bear" sales, prevented there being much difference in the two quotations, and we close the year with spot silver at 23 $\frac{7}{16}$ and forward 23$\frac{1}{8}$ per ounce standard. There have been no transactions in Mexican dollars throughout the year. The gold imported into Great Britain during 1908 shows a considerable decrease as compared with the previous year, the figures being £45,000,000 against £57,000,000 in 1907. This decrease is entirely due to there being less shipped from the Continent,

those shipments last year having been exceptionally heavy to meet the drain on London by America during the financial crisis in New York. The imports from South Africa, however, show an increase of nearly two millions over those of last year. The price in January was 77s. 9d., and by the end of March the Bank of England purchased over £5,000,000 in bars, having secured a share of each week's arrival. It was not till April that the competition for gold—by Paris and other Continental centers—became more keen, and the price was raised to over 77s. 10d. Since then there has been a strong export demand for the arrivals of gold in the open market, the quotation advancing to over 77s. 11d. in October. Paris has nearly always been the best buyer, and the exports to France amount to the exceptionally large total of nearly £23,000,000 during the year. The withdrawals from the Bank of England have, however, not been important, and the chief movements have been the usual amounts for Egypt and South America.

Commercial Movement of Gold and Silver.

The general course of the movement of gold and silver is shown in the accompanying tables:

GOLD IMPORTS AND EXPORTS, UNITED STATES.

	1903	1904	1905	1906	1907	1908
Imports	$65,267,296	$84,803,234	$50,203,405	$155,579,380	$143,398,072	$50,276,293
Exports	44,346,834	121,138,415	46,794,467	46,709,158	55,215,681	81,215,456
Balance	I $20,920,462	E $36,335,181	I $3,498,938	I $108,870,222	I $88,182,391	E $30,939,163

GOLD IMPORTS AND EXPORTS, GREAT BRITAIN.

	1903	1904	1905	1906	1907	1908 (a)
Imports	£28,657,393	£33,876,588	£38,567,895	£46,042,500	£57,088,547	£45,206,188
Exports	27,766,512	33,039,138	30,829,842	42,617,267	50,866,009	50,206,824
Excess Imports	£890,881	£837,450	£7,738,053	£3,425,233	£6,222,538	(c) E £5,000,636

GOLD IMPORTS AND EXPORTS, FRANCE.

	1903	1904	1905	1906	1907	1908 (b)
	Francs	Francs	Francs	Francs	Francs	Francs
Imports	314,259,000	656,063,000	779,648,000	430,473,000	492,336,000	947,790,000
Exports	129,800,000	123,976,000	131,494,000	165,087,000	154,572,000	22,004,000
Excess Imports	184,369,000	532,087,000	648,154,000	265,386,000	337,764,000	925,786,000

(a) Based on 11 months' returns. (b) Based on 10 months' returns. (c) Excess exports.

GOLD HOLDINGS OF THE LEADING EUROPEAN BANKS.

	1904	1905	1906	1907	1908
Bank of England	$149,980,465	$142,651,255	$145,322,390	$165,383,645	$165,087,420
Bank of France	469,307,905	575,671,135	541,150,235	538,787,750	689,929,615
Bank of Germany	154,370,000	166,300,000	137,940,000	137,140,000	192,155,000
Austro-Hungarian Bank	231,165,000	224,325,000	233,045,000	228,795,000	244,880,000
Bank of Russia	426,375,000	576,215,000	589,520,000	607,125,000	606,805,000
Bank of the Netherlands	21,036,000	33,019,500	27,690,000	38,239,100	41,933,000
Belgian National Bank	15,613,335	16,233,335	17,076,665	17,610,000	20,103,335
Bank of Italy	108,520,000	134,345,000	159,440,000	193,320,000	187,000,000
Bank of Spain	72,780,000	75,117,000	76,840,000	78,210,000	78,965,000
Bank of Sweden	.....	18,900,000	19,780,000	20,325,000	21,215,000
Total	$1,649,147,705	$1,962,775,225	$1,947,794,290	$2,024,935,295	$2,248,053,390

UNITED STATES: EXPORTS AND IMPORTS OF SILVER.

	1904	1905	1906	1907	1908
Exports	$50,312,745	$57,513,102	$60,957,091	$61,625,886	$51,837,671
Imports	26,087,042	35,939,135	44,227,841	45,912,360	42,224,130
Excess, exports	$24,225,703	$21,573,967	$16,729,250	$15,713,526	$9,613,541

GREAT BRITAIN: EXPORTS AND IMPORTS OF SILVER.

	1903	1904	1905	1906	1907	1908(b)
Exports	£11,466,726	£13,263,694	£14,561,677	£18,865,285	£15,813,329	£12,877,212
Imports	10,310,330	11,687,339	12,992,014	17,288,063	14,667,024	10,453,520
Excess, exports	£1,156,396	£1,576,355	£1,569,663	£1,577,222	£1,146,305	£2,423,692

(b) Based upon returns for first 11 months of one year.

EXPORTS OF SILVER FROM LONDON TO THE EAST. (a)

	1903	1904	1905	1906	1907	1908
India	£7,423,330	£9,527,618	£7,230,421	£15,129,627	£10,531,354	£8,412,390
China	310,060	512,792	898,847	433,967	417,350	641,400
The Straits	821,879	79,268	38,299	1,750	691,150	164,885
Total	£8,555,269	£10,119,678	£8,155,567	£15,565,334	£11,639,854	£9,218,675

(a) As reported by Pixley & Abell.

UNITED STATES MINT PURCHASES OF SILVER IN 1908.

	Jan. 1-Apr. 30	Dec.	Year.
Ounces	4,978,300	125,000	5,103,300
Average Price, cents per oz	55.723	49.118	54.825

Note—No purchases made between Apr. 30 and Dec. 1.

Gold Production and Commodity Prices.[1]

By Walter Renton Ingalls.

No one can deny that the value of gold periodically appreciates and depreciates. The value of gold is what it will buy and that is constantly changing, but inasmuch as there must be something as the standard, gold is so adopted, and the prices for commodities are said to fluctuate with reference to gold, rather than the reverse.

The great rise in the value of commodities, which began about 1896 and culminated in 1907, naturally excited the attention of producers, manufacturers and consumers, and equally that of the bankers who regulate industrial developments and economists who study them. The paramount question pertained to the cause of this extraordinary rise. A remarkable increase in the production of gold occurring contemporaneously, it was easiest to explain by *post hoc ergo propter hoc* arguments, that the rise in commodity prices was due to the increased production of gold, meaning that the value of gold had depreciated. The purpose of this paper is to show the fallacy of that idea.

Prof. J. E. Cairnes, of London, in 1859 undertook an inquiry as to how increase in gold production might influence the markets of the world. He pointed out that the wages of labor in the gold-producing countries rose to such hights, on account of the enormous profits, as to divert labor from other industries, necessarily ending in raising the prices of their outputs; the sudden wealth of the successful miners, spent profusely for necessaries and luxuries, created fresh demands for commodities and thereby elevated prices; finally, the addition of the new gold to the banking reserves of the world enabled banks to give largely extended credits and thus facilitate expansion of trade and industry.

Professor Cairnes' conclusions were based upon a sound study of the evidence existing at his time, but in his time gold mining as an industry was scarcely 10 years old, and we know now that (1) a great increase in gold production may occur without increasing wages and without attracting any large number of men away from previous occupations; (2) the profit derived from the production of gold is rarely expended at the place of production, but is distributed among many investors in the commercial countries of the world; and (3) much of what is considered to be profit is really not so, a subject to which I shall revert farther on.

Increased Demand for Commodities.—The only way in which an increasing supply of gold can affect the prices of commodities is by increasing the demand for them. The new gold is partly expended by the producer himself, and partly invested by the stockholder who receives

[1] This is a condensation of an article that appeared in *Eng. and Min. Journ.*, of Nov. 28, 1908, under the title of "Has the Value of Gold Depreciated?"

it in the form of a dividend. The part invested is expended for commodities by somebody else. If deposited in a bank, by the person who borrows it from the bank; if invested in a bond, by the seller thereof. In the words of Horace White, "There is no stopping place between the new gold and the commodities—food, clothing and other necessaries and luxuries—unless the gold is hoarded." In the latter case it has no more effect on prices than it had before it was taken out of the ground. In short, gold is itself demand for goods, new gold is new demand, an increasing supply of it is an increasing demand, and this causes a rise of prices, other things being equal.

So much for the theory. An increase in the production of gold will cause an increase in the prices for commodities, other things being equal. But as a matter of fact, other things do not remain equal. Population and the production of the necessaries of life may increase in the same ratio as the production of gold, so that the latter merely supplies the increased requirement for it, and under that condition will not influence prices one way or the other. Industrial changes in processes of manufacture, means of transportation, and many other conditions may affect prices so that the average of the latter will go down, while the production of gold goes up, or *vice versa*. The crises of 1873 and 1890 were each followed by several years of severe commercial depression, but in each case, while prices were falling the production of gold was increasing, showing that gold of itself may not create sufficient demand to maintain prices. In this connection it is important to study the experience of the world from the time when records first became available.

Statistics of Gold Production.—Statistics of the world's production of gold, year by year, have been compiled as far back as 1851. In the early part of this period the statistics were not of contemporaneous compilation by reliable authorities, but are largely the result of the retrospective investigations of later statisticians. For this reason there are discrep-

I. GOLD PRODUCTION OF THE WORLD, 1851-1908.

Year.	Value.	Year.	Value.	Year.	Value.	Year.	Value.
1851	$ 67,600,000	1866	$121,000,000	1881	$103,102,000	1896	$211,242,081
1852	132,800,000	1867	104,000,000	1882	102,000,000	1897	237,833,984
1853	155,500,000	1868	109,700,000	1883	95,400,000	1898	287,327,833
1854	127,500,000	1869	106,200,000	1884	101,700,000	1899	311,505,947
1855	135,100,000	1870	106,900,000	1885	108,400,000	1900	258,829,703
1856	147,600,000	1871	107,000,000	1886	106,000,000	1901	260,877,429
1857	133,300,000	1872	99,600,000	1887	105,775,000	1902	298,812,493
1858	124,700,000	1873	96,200,000	1888	110,197,000	1903	329,475,401
1859	124,900,000	1874	90,800,000	1889	123,489,000	1904	349,068,293
1860	119,300,000	1875	97,500,000	1890	118,848,700	1905	378,411,754
1861	113,800,000	1876	103,700,000	1891	130,650,000	1906	405,551,022
1862	107,800,000	1877	114,000,000	1892	146,292,600	1907	411,294,456
1863	107,000,000	1878	119,000,000	1893	158,437,551	1908	444,382,312
1864	113,000,000	1879	109,000,000	1894	182,509,283		
1865	120,200,000	1880	106,600,000	1895	198,995,741		

ancies among the reports of various statisticians, which gradually disappear as the work of collecting the statistics of the gold production of the world was undertaken annually by reliable authorities such as the director of the United States Mint and THE MINERAL INDUSTRY. In Table I are given statistics of the world's annual production of gold beginning with 1851. The figures for 1851 to 1891, both inclusive, are from the report of the director of the United States Mint; those for subsequent years are as reported by THE MINERAL INDUSTRY. A different set of figures for 1851-72 is given in a British Parliamentary report for 1876, based on the figures of Tooke, Newmarch, and the London *Economist*. They are shown in Table II.

II. GOLD PRODUCTION OF THE WORLD.
1851-1872.

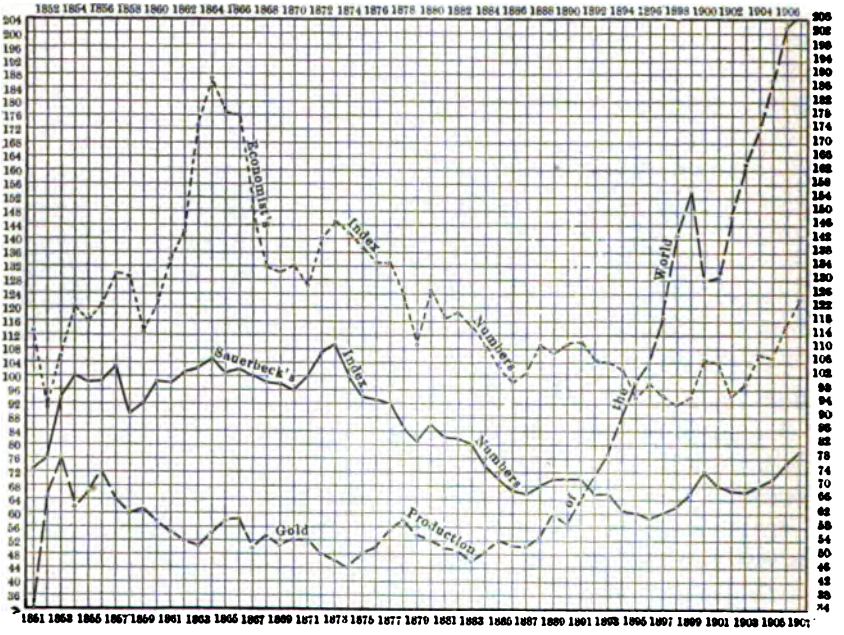
Year.	Adopted by U. S. Mint.	Parliamentary Report.	Year.	Adopted by U. S. Mint.	Parliamentary Report.
1851	$ 67,600,000	$ 67,600,000	1862	$107,800,000	$110,590,000
1852	132,800,000	135,150,000	1863	107,000,000	100,575,000
1853	155,500,000	140,400,000	1864	113,000,000	94,980,000
1854	127,500,000	141,400,000	1865	120,200,000	101,270,000
1855	135,100,000	151,200,000	1866	121,000,000	108,600,000
1856	147,600,000	161,250,000	1867	104,000,000	103,500,000
1857	133,300,000	138,225,000	1868	109,700,000	97,570,000
1858	124,700,000	122,055,000	1869	106,200,000	106,415,000
1859	124,900,000	107,290,000	1870	106,900,000	92,250,000
1860	119,300,000	93,415,000	1871	107,000,000	108,965,000
1861	113,800,000	112,270,000	1872	99,600,000	87,845,000

Index Numbers.—The average prices for commodities are represented by index numbers computed by several authorities, of which the best known are those of the London *Economist* and Augustus Sauerbeck. The statistics of both these authorities antedate 1851. Unfortunately, however, the statistics of the *Economist* at present available are only for certain dates of each year, those commonly quoted being of January 1.

The *Economist's* and Sauerbeck's index numbers, together with the world's production of gold, in dollars, have been plotted in Diagram I. It will be understood, of course, that the appearance of these curves may be altered by changes in the scale of the abscissas and ordinates, although their general relationship will be the same. In order to bring the three curves upon the same plate, the *Economist* index numbers were divided by 20, while the world's production of gold is plotted in units of $2,000,000. The Sauerbeck index numbers are plotted normally.

I am aware that comparisons between gold production and the index numbers have previously been made by economists, of whom some are inclined to put but little weight upon it, because of (1) inaccuracies in the index numbers, (2) the chance that they may differ from "weighted" averages, and (3) the well-known fact that the status of credit may

entirely offset the effect of gold production. I hope to forestall criticism on these grounds by pointing out that (1) the index numbers of various authorities agree fairly well, (2) in the cases of many important commodities as to which it has been possible to make the test the "quotational" average and the "weighted" average over a series of years have not been far apart, and (3) while it is true that the effect of gold production may be offset by the status of credit, it is upon the former alone that the advocates of the depreciation theory hang their case. Consequently, I hold that a comparison between gold production and the index numbers affords a substantial test of the theory.



I. GOLD PRODUCTION AND COMMODITY PRICES.

Non-Coincidence.—Diagram I does not show any constant coincidence between the production of gold and prices for commodities. On the contrary, the courses of the two curves are to a large extent the opposite. From 1851 to 1859 there appears to be a parallelism, if we disregard the depression in gold production in the middle of the period, which may be due to statistical errors, the statistics for the early times being more or less subject to doubt as previously explained. The statistics of gold production in the British Parliamentary report do not show this depression. In fact there is considerable variation between the Parliamentary statistics and those adopted by the U. S. Mint between 1859 and 1873,

although the divergencies of the Parliamentary statistics are not sufficient to throw doubt upon the showing of the other statistics which I have

III. INDEX NUMBERS OF COMMODITY PRICES.

Year.	Sauerbeck's.	Economist.	Year.	Sauerbeck's.	Economist.	Year.	Sauerbeck's.	Economist.
1851	75	2,293	1870	96	2,689	1889	72	2,187
1852	78	1,863	1871	100	2,590	1890	72	2,236
1853	95	2,167	1872	109	2,835	1891	72	2,240
1854	102	2,445	1873	111	2,947	1892	68	2,133
1855	101	2,357	1874	102	2,891	1893	66	2,121
1856	101	2,459	1875	96	2,778	1894	63	2,082
1857	105	2,645	1876	95	2,711	1895	62	1,923
1858	91	2,612	1877	94	2,715	1896	61	1,999
1859	94	2,304	1878	87	2,529	1897	62	1,950
1860	99	2,426	1879	83	2,225	1898	64	1,990
1861	98	2,727	1880	88	2,538	1899	66	1,918
1862	101	2,878	1881	85	2,376	1900	75	2,145
1863	103	3,492	1882	84	2,435	1901	70	2,126
1864	105	3,787	1883	82	2,342	1902	69	1,948
1865	101	3,575	1884	76	2,221	1903	69	2,003
1866	102	3,564	1885	72	2,098	1904	70	2,197
1867	100	3,024	1886	69	2,023	1905	72	2,136
1868	99	2,682	1887	68	2,059	1906	77	2,322
1869	98	2,666	1888	70	2,230	1907	80	2,499

plotted. I am disposed to think that the Parliamentary statistics for 1851-56 are the more accurate, but less accurate for 1857-72, this opinion being based upon the appearance of the two curves.

IV. INDEX NUMBERS OF COMMODITY PRICES.

Year.	U. S. (a)	Bradstreet's.	Year.	U. S. (a)	Bradstreet's.	Year.	U. S. (a)	Bradstreet's
1890	112.9		1896	90.4	5.912	1902	112.9	7.876
1891	111.7		1897	89.7	6.116	1903	113.6	7.936
1892	106.1	7.777	1898	93.4	6.571	1904	113.0	7.919
1893	105.6	7.532	1899	101.7	7.210	1905	115.9	8.100
1894	96.1	6.685	1900	110.5	7.884	1906	122.5	8.418
1895	93.6	6.435	1901	108.5	7.575	1907	129.5	8.905

(a) As computed by the Department of Commerce and Labor.

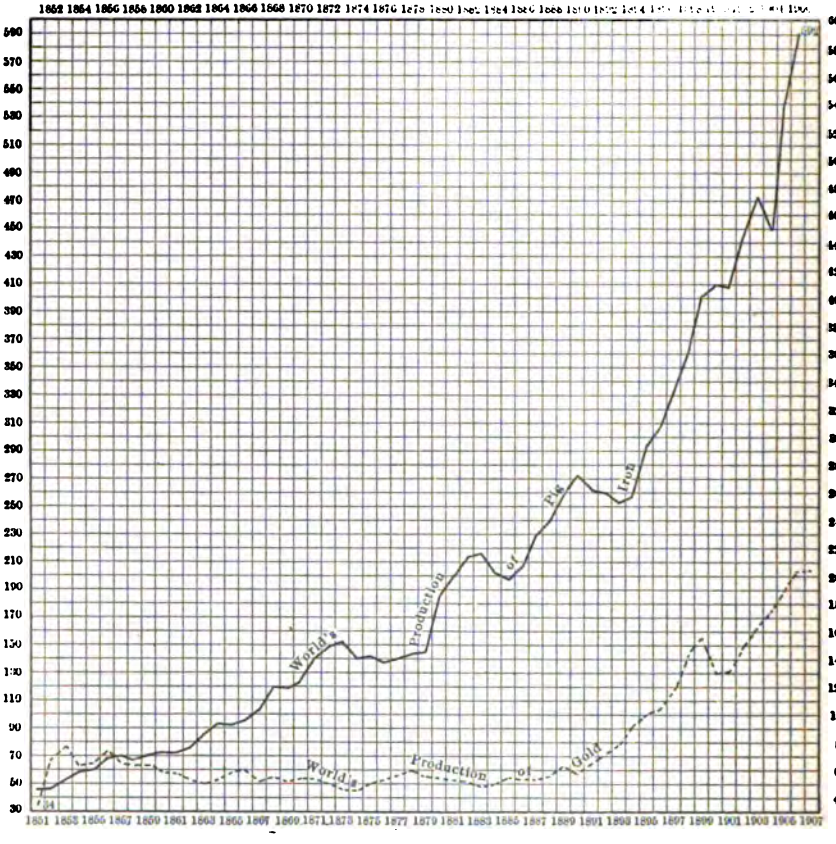
After the crisis in 1857, the prices for commodities rose steadily until 1864, while the production of gold was going down. Prices then declined

V. WORLD'S PRODUCTION OF PIG IRON. (a)
(In Metric Tons.)

1851	4,584,232	1866	9,782,142	1881	20,039,461	1896	30,850,412
1852	4,688,156	1867	10,224,013	1882	21,461,867	1897	33,451,220
1853	5,315,672	1868	11,996,325	1883	21,729,202	1898	36,555,361
1854	5,706,490	1869	11,889,312	1884	20,150,042	1899	40,447,900
1855	5,909,757	1870	12,302,372	1885	19,792,390	1900	41,028,479
1856	6,909,605	1871	13,874,315	1886	20,862,989	1901	40,950,692
1857	7,051,760	1872	14,861,974	1887	22,977,983	1902	44,342,579
1858	6,582,517	1873	15,170,115	1888	24,016,705	1903	47,113,730
1859	6,961,216	1874	13,994,179	1889	26,030,266	1904	46,089,501
1860	7,291,521	1875	14,140,081	1890	27,431,229	1905	54,054,783
1861	7,179,754	1876	13,807,343	1891	26,219,525	1906	59,074,861
1862	7,782,631	1877	14,073,446	1892	25,978,619	1907	60,680,014
1863	8,726,547	1878	14,352,605	1893	25,266,746		
1864	9,368,260	1879	14,473,964	1894	25,847,352		
1865	9,351,073	1880	18,626,160	1895	29,354,597		

(a) The statistics for 1851-1899 are compiled from data in Neumann's "Die Metalle;" those for subsequent years are from *The Mineral Industry*.

until 1870, while gold production was fluctuating between comparatively narrow limits. In 1871 prices started upward and rose until 1873, while gold production was declining. Prices then fell steadily until 1879, while gold production was increasing. After a slight rise of prices in 1880 the decline was resumed, and this time coincided with a declining gold production until 1883, when gold production began the rise which has continued with only slight interruptions ever since, while the decline in



II. PRODUCTION OF GOLD AND PIG IRON.

commodity prices kept on until 1896. Thus it may be seen that from 1858 to 1896 the prices for commodities frequently went down as the production of gold increased, and *vice versa*. In 1896 began for the first time a steady and prolonged rise both in prices and gold production.

Gold and Pig Iron.—Comparison between the statistics of gold production and commodity prices being so negative as to a relationship between the two, it appeared useful to consider what relation there has been

between the production of gold and the world's requirement for it. The world's production of pig iron was selected as the best indication of the advance in the arts and general civilization, the status of the industrial countries of the world being well established by the amount of pig iron that they use. Pig iron is a substance for which reliable annual statistics are available as far back as 1851.

The world's production of gold and of pig iron is plotted in Diagram II. In this diagram the production of gold is plotted in units of $2,000,000. The production of pig iron is plotted in units of 100,000 tons. Inasmuch as the plotted unit of gold production is $2,000,000, and 100,000 tons of pig iron may be reckoned on the average as being worth approximately $2,000,000, the world's production of these two substances is represented substantially on terms of commercial equality, which consequently affords a reliable means for comparison.

A mere glance at these two curves discloses the striking story that they have to tell. The general tendency of each is to ascend, that of pig iron from the beginning and that of gold since 1883, but the production of pig iron has increased in much greater ratio than the production of gold. If I am correct in the assumption that the production of pig iron is a reliable barometer of the advance in commerce and industry, it is evident that the production of gold has not even kept pace with the commercial requirement of the world for it. If this deduction be sound, there has been no superfluity of gold to manifest itself by a rise in commodity prices, and such increase of the latter as has occurred since 1896 must have been due to other causes. It might even be argued that the production of pig iron, and inferentially the need for a medium of exchange in transacting business, has increased so much more rapidly than the production of gold, that the relative lag of the latter might have led to commercial disturbance, but we find the explanation for the normality of the curves in the knowledge that banking facilities and the means for transacting business have improved, so that we have been able progressively to get along with less and less gold relatively.

It appears to me to be conclusive from the above statistical analysis that (1) during the last 60 years there has been but slight parallelism between the production of gold and the average of commodity prices; and (2) the production of gold has not increased so rapidly as the production of pig iron and inferentially less rapidly than the advance of the world in commerce and industry. Then let us see in what way the exponents of the theory of the depreciation of gold go astray.

Profit in Production.—The economists who advance the argument that gold has been becoming cheaper to get would do well to dismiss the matter of working cost and fix their attention upon the formula $X=M-(o+$

$d+p$), in which X is the net profit per ton of ore, M the value of the metal extractable from a ton of ore, o the direct operating cost, d the proportional cost of development work, and p the amount that must be charged off for amortization of plant and capital invested in all forms. It is true that working costs have been greatly reduced during the last 10 years, which has enabled the utilization of lower and lower grades of ore, but it is doubtful if the net profit per ton of ore has been increased. On the contrary, it probably has decreased. The great increase in gold production has come from the Witwatersrand, but its mines have not been so profitable as many think, if all things be considered. This subject is too complex to be taken up in a brief discussion, but it pertains to the question whether in spite of the large dividends paid by the Rand mines they are not failing to return the actual investment in them with a fair interest upon it.

Bonanzas Become More and More Infrequent.—Any possible depreciation in the value of gold must arise from an increase in the ease of obtaining it. However, the experience has been that (1) rich placers are quickly exhausted, (2) the profit gradually diminishes as the poorer ground has to be worked, (3) the discovery of new placers become more and more infrequent, and (4) the proportion of gold obtained from placers as compared with lode mines steadily decreases. In the case of many of the remarkable lode-mining districts the history is similar. Cripple Creek and Kalgoorlie, for example, began with extraordinarily rich mines, but within a few years it became a matter of operating on a close margin; doubtless the history of Goldfield will be the same. The output of cheap gold from such sources has generally been, at any given time, small in comparison to the grand total, and never has lasted long.

The depreciation theory must stand upon two conditions, namely, (1) some regular relation between the amount of gold production and commodity prices, and (2) evidence that the cost of producing gold has diminished, while commodity prices have risen, or *vice versa*. These conditions are closely related, because it cannot be argued rationally that an increase in the production of gold depreciates its value, unless the gold be obtained at a reduced cost.

The failure to establish the first condition is shown by the comparison of the statistics of gold production with the index numbers representing the average prices of commodities, which is made in Diagram I. Equally destructive of the depreciation theory is the record of the Witwatersrand, the marvelous and rapidly increasing production of which is apparently the basis for the recrudescence of the theory. While the production of the United States, Australasia, Russia, Mexico, Canada, and India during the last 10 years has been increasing but slowly, that of the Transvaal has

risen by leaps and bounds. However, it does not follow from this that the Transvaal gold has become increasingly easier to obtain. Mining and metallurgical engineers know the contrary to be the case, and their opinions are supported by the statistics which for this district are singularly complete. What the statistics show as to the total production and total dividends appears in an accompanying table.

The total production in the five years 1890-94 was £22,482,671, out of which dividends amounting to £3,951,211, or 17.5 per cent. were paid. In the five years 1895-99 the production was £60,796,424, out of which dividends to the amount of £14,062,311 or 23.1 per cent. were paid. In the six years 1902-07 the production was £108,683,521 and the dividends were £27,118,770 or 24.9 per cent. The percentages in 1900 and 1901 are not computed, these being the war years. It appears that the average dividends of the six years since the war have been about 25 per

VI. PRODUCTION AND DIVIDENDS OF THE TRANSVAAL GOLD MINES.

Year.	Production. £	Dividends. £	Per Cent.	Year.	Production. £	Dividends. £	Per Cent.
1890	1,869,645	254,551	13½	1901	1,096,051	(a) 415,813	
1891	2,924,305	334,608	11½	1902	7,253,665	(a)2,121,126	29
1892	4,541,071	879,320	19½	1903	12,589,248	3,345,502	26½
1893	5,480,498	955,358	17½	1904	16,054,809	3,927,830	24
1894	7,667,152	1,527,284	20	1905	20,802,074	4,857,539	23
1895	8,569,555	2,046,852	24	1906	24,579,987	5,735,161	23½
1896	8,603,821	1,513,682	17½	1907	27,403,738	7,131,612	.....
1897	11,653,725	2,707,180	23	1908	.....	.....	.....
1898	16,240,630	4,848,230	30	1890–1894	22,482,671	3,951,211	17½
1899	15,728,693	(a)2,946,358	19	1895–1899	60,796,424	14,062,311	23.1
1900	1,510,131	(a)		1902–1907	108,683,521	27,118,770	24.9

(a) War period, October 11, 1899 to May 31, 1902.

cent. against about 23 per cent. as the average of the five years just before the war. The average of 1890-94, a period when the mining industry of the Transvaal was in its initial stage, was 17.5 per cent.

Such a record of dividends is instructive, but it must not be accepted as the last word in a case like this. Mining dividends are profit only in part; a large part is simply return of the principal invested. As to the amount of the latter, there is seldom any record. The increase in the dividends paid by the Rand has doubtless been to a large extent the result of the investment of great sums of money sent out from Great Britain and France, which have enabled the erection of mining and milling plants capable of operating on a vastly increased scale and at a materially reduced expense. Yet it does not follow that the apparent economy and increase in earnings thus resulting is all profit. On the contrary, there are engineers who argue that the net yield of the Rand mines is on the whole no more than the money put into them would have returned if invested in 3 per cent. consols. While this may be true insofar as the

rank and file of investors is concerned, it probably is not true with respect to the money actually invested in the mines, which is to say that the real profits have been anticipated (as is generally the case) by the promoters.

During the recent history of mining on the Rand, the production of gold has been increased greatly through the opening of new mines and the prosecution of mining generally upon a larger and larger scale. The grade of the ore extracted has rather steadily diminished, and the effort of the mine managers has been to reduce the cost of production in a greater ratio. To some extent they have succeeded in that, but at what expense of capital, i.e., accumulated money drawn from investors in Great Britain, France and elsewhere, it is difficult to determine. Consequently, we cannot talk in terms of the ultimate analysis about the cost of producing gold on the Rand. But disregarding this important consideration, it cannot reasonably be argued that a profit of 25 per cent. of the production in the six years after the war against 23 per cent. in the five years just before the war, these periods coinciding with the great increase in the world's gold production that has alarmed certain economists, indicates any such reduction in the cost of obtaining gold as would depreciate the value of the latter so as to account for the great rise in commodity prices. It is worth while in this connection to repeat that the Rand is far and away the most important gold-mining district of the world, and its output represents approximately 30 per cent. of the world's total.

Altered Conditions.—The conditions of gold mining in the '50s long since ceased to exist. Previous to the nearly contemporaneous discoveries of gold in California and Australia, the industry of gold mining and the world's production of gold were both insignficant according to modern standards. The statistics for the time previous to 1851 are too incomplete and unreliable to furnish a substantial basis for the study of the relations between gold production and commodity prices. Nevertheless, it is well established that prices were at certain times greatly affected by gold supplies. Thus, one of the chief causes of the decay of Spain after the reign of Ferdinand and Isabella is said to have been the extensive importation of gold and silver from the newly found Western continent, with the prevailing theory that actual money was the only form of wealth. Prices of commodities in Spain were turned topsy-turvy by the vast influx of the precious metals. But I may point out that the precious metals then brought into Spain were the result of conquest, not of direct mining, and were obtained comparatively cheaply.

In the early days of mining in Australia and California, the magnets which attracted men from other industries were placer deposits, the working of which was mining in its elemental form, i.e., the digging of a valuable mineral with the expenditure of nothing but labor. The adven-

turers who were able to obtain gold from rich river bars with a pan or a long tom were willing to spend their produce profusely and extravagantly, and their cheaply gained gold created commercial disturbances. However, just as the ability to obtain gold by conquest on the Spanish Main disappeared, so also did the ease of obtaining gold from rich placers in Australia and California. Although the early history of those regions was repeated recently in the Klondike and at Nome, the life of their rich placers was brief (as is always the case) and their output was too small in comparison with the world's total at their time to be of any material effect.

From this study of statistics and economic conditions I draw the conclusion that during the last 50 years there has been no cheapening of an ounce of gold; that the production of gold has not increased in so great a ratio as the production of pig iron, and probably the advance of the world in commerce and industry; and that gold production and commodity prices have failed to follow similar courses.

Other Explanations.—If, then, the recent rise in commodity prices has not been due to the increasing gold production, the explanation must be sought in other directions. Undoubtedly, the causes have been complex. On the one hand we have remarkable improvements in industrial methods tending to reduce the cost of production, and consequently prices. On the other hand we have exhaustion of certain natural resources, such as timber, tending to increase prices. Among the factors which undoubtedly have contributed to the increase in commodity prices during the last 10 years are the policy of labor unionism and the increasing inefficiency of labor; the industrial consolidations, accumulation of great fortunes through anticipation of profits, and an inauguration of general extravagance in living; waste of resources and reckless expenditure of money in replacing them; over-extension in enterprise; failure of agricultural development to keep pace with the growth of population, thereby increasing the cost of food and clothing and directly leading to advance of wages for labor in manufacturing. The extent to which these and other factors have operated is an important subject of inquiry by the economists. Less has been heard of the theory that gold has depreciated in value since the commercial depression began in 1907, affording a decisive demonstration that prices still can fall as well as rise. It is scarcely to be doubted that the prices for commodities will continue to fluctuate in the future as they have in the past, independently of the world's production of gold, the effect of which upon prices at any time in the world's history has been but ephemeral and has been due to conditions that long since ceased to exist. The steady increase in the production of gold is in itself evidence that mining for it is now carried on as an industry, and as in the case of

most industries at a small margin of profit on the whole; the gold supply is no longer swollen by the unearthing of buried treasure, by piratical or martial conquest, or by the discovery of placer bonanzas.

Developments in Gold Dredging During 1908.

By John Power Hutchins.

The strained financial conditions of 1908 had no bad effect upon the operation of gold dredges. Gold dredging, in having an unlimited demand and a constant price for its product, is an attractive kind of mining. However, the hard times interfered with searching for and equipping new dredging enterprises, although there was more or less search for new ground which was rewarded in several instances by good finds.

Ground that can be worked successfully with floating dredges is hard to find. The most common drawback is a lack of sufficient gold. It is remarkable that there are so many areas containing only a few cents per cubic yard immediately adjoining ground which furnished rich diggings. Many instances can be cited of creek valleys having all the essential characteristics of dredging deposits, except sufficient gold, surrounded by old bench diggings with records of wonderful yields. There are also numerous creek gravels with sufficient gold, as compared with California dredging fields, but lacking the features that would make it possible to exploit them with floating dredges. There is usually too low a content of gold, or if there is more than a few cents per cubic yard there are big boulders, or hard, rough bedrock, or too much clay, or hostile environment or some other factor to raise dredging costs to a prohibitive level.

There is at present an eager search for new dredging fields. Countries possessing cold climates and frozen ground are attracting the greater attention. It is probable that the investigation now being conducted in Siberia will result in developing important dredging fields. A similar activity is noted in tropic South America. Colombia is under investigation, and it is probable that several dredges will soon be installed there. There have been many dredging failures in South America, but only recently has a first-class modern dredge of California type been installed in that continent. This dredge was built on the Atrato river and is now being moved to the ground where it will be operated.

Review of Operations by States and Countries.

Alaska.—On Seward peninsula, dredging was successful wherever proper methods were employed. Where failures have occurred, management such as would have resulted in failure elsewhere was responsible.

There are, to be sure, a number of difficult problems connected with the hostile conditions of the Far North. Frozen ground is a drawback more than a "bugaboo." There are areas of unfrozen ground in Alaska and some of these will be dredged at good profit. Whether the large volumes of tundra gravel will ever be exploited successfully by the dredging method is problematic, but it is likely that if frozen ground can be thawed out successfully in the Klondike by stripping and draining, the tundra gravel may be treated similarly. However, the problem of preventing gravel thawed in summer from freezing solidly during the following winter is yet to be solved.

California.—Dredging in this State showed a healthy condition. Some very powerful dredges of large capacity were built and these are doing profitable work in ground that only a few years ago was thought to be undredgeable. Dredging at Oroville had no unusual incidents. A number of the older dredges were scrapped. The Anti-Débris Association was active and enjoined dredging operations even where no real damage was done. The city of Oroville enjoined one dredge company which carried on work in the river in a way supposed to cause choking of the river bed so as to give trouble during freshets. There was more or less agitation in cities situated on other rivers in which dredging is carried on. Sacramento took steps to investigate alleged damage done by the dredges near Folsom. None of these dredges work in the river, and the only possible damage that can be done is by the inconsiderable amount of material held in suspension in the water that runs from the dredge ponds. This probably amounts to not many hundredths of an ounce per gallon of water and it is of such character that it will not settle where there are river or tidal currents. The first large dredge to be installed in northern California began operations in 1908. This is the first large dredge to begin work in California in gravel differing from that at Oroville and at other places on the flanks of the Sacramento and San Joaquin rivers. The dredging province of California was extended somewhat by the investigation of rivers tributary to the San Joaquin and formerly considered undredgeable.

Another large merger of dredging companies was recently consummated in California. A total acreage equal to all of the dredging ground at Oroville was included. It is expected that lower working cost will be attained than when operating under several corporations. Standardization of dredge machinery in size and design will be attempted and the merged interests will build their own dredges. Dredging is a kind of mining that is particularly amenable to merger.

Idaho and Colorado.—Dredging was carried on in Idaho with success. In one instance ground worked several times by hand is now being

dredged. In this area the richest returns are often obtained in the ground that has seemingly been most thoroughly worked by hand methods. In one place, an improved dredge is being installed on ground that has been idle for three years after being dredged for five years.

A similar change is reported in Colorado where dredges failed in the early days, and at present modern powerful dredges of large capacity are working successfully. Two dredges originally built in Clear Creek were moved, one to a place near Breckenridge, Colo., and the other to Calaveras county, California.

Montana.—Some interesting dredging in Montana furnished an opportunity for a comparative test of a remodeled dredge fitted with gold-saving tables. This dredge formerly had a sluice box. It is reported that the tailings from the dredge in its original form with the remodeled machine yielded 5 per cent. of the original saving. This was about the same as found by drilling the tailings to ascertain their gold content. The sluice box was supposed to be less efficient as a gold saver than the tables, but these figures show that the loss was inconsiderable, and it is probable that if the tailings had been worked a second time by a sluice-box dredge, an equal saving might have been effected. There is, unfortunately, still uncertainty as to the relative merits of sluice box and table, the relative merits of which are not easy to determine. Any procedure to gain knowledge is expensive, and it is extremely difficult to make accurate determinations when handling such large volumes of material as in dredging. The gold dredge with the largest buckets in the world, 15-cu.ft. capacity, is owned by the same company. This dredge has a sluice box, and by reason of recent experience with the remodeled type and of other, though minor, considerations, it will not be immediately changed to the table type. A gold dredge is being installed near Butte. It will operate in material that is by some mining men not considered to be placer material in the limited sense of the word. The ground has some peculiar characteristics, and the operations of the dredge will be watched with interest. Montana has large areas of ground which may prove to be dredgeable.

The Klondike.—Dredging in the Klondike made important progress during 1908. Steam thawing was practiced on a large scale and at a lower cost, and less difficulty was experienced. Ground with a high gold content was found, and stripping accompanied with drainage so as to lower the water level produced progressive thawing. It is expected that this stripping and draining will, after a time, make it unnecessary to use steam thawing except in rare cases. Thawing will (it is thought) progress deeper each summer until it reaches bedrock and the ground will then remain unfrozen. There is, however, nothing certain about this, for

the ground may freeze again during the winter. The average temperature for the year is considerably below freezing. Gravel stripped, drained and thawed to a depth of more than 15 ft. has been known to freeze solidly during the next cold season. There is much ground that now has all the characteristics of stripped and drained creek beds, yet this ground thaws superfically in summer and freezes again in winter. It is to be hoped that stripping and draining will result in progressive thawing to bedrock in the Klondike creek beds, for if successful the practice could be applied elsewhere, thus extending the dredging province considerably.

Stripping by thawing alone did not prove successful in the Klondike, for it was found that material did not thaw to any considerable depth below water level. During 1908 the gravel was drained and much better results were accomplished. As predicted in the *Engineering and Mining Journal* (Jan. 4, 1908, p. 58, and Mar. 7, 1908, p. 512) it was found that gravel does not thaw below water level when it is of the depth generally encountered in the Klondike creeks. When there is a considerable depth of overburden, which may be stripped by hydraulicking, and a relatively shallow section of gravel is left for dredging, stripping (aided by draining) will usually result in seasonal thawing to bedrock.

Dredging in its limited meaning is applied only to unfrozen areas. Where frozen ground is encountered other methods incorporating some of the features of drift, hydraulic, or other open-cut mining practice are applied. There has been considerable progress in dredging in the Klondike, and the high gold content of some of the gravels will result in exploitation even though the working cost may be high. Dredging in unfrozen ground has been carried on with great success. It was a great advantage that where most of the dredging was prosecuted the ground was only partially frozen. Recent installations of dredges include buckets of 7-cu.ft. capacity. Several dredges with 5-cu.ft. buckets were installed two years ago. A 7-cu.ft. dredge installed in 1905 was successful from the beginning. The larger bucket seems to have shown its advantages in the Klondike as elsewhere. This experience is similar to that of dredge men in California, where there was for a time a retrogression toward small buckets for working ground suitable for large dredges.

Other Countries.—Investigations are being carried on in Siberia and new dredging ground is being developed. Where there has been such a large output of gold, most of it by placer methods, it is likely that there will be a considerable development of dredging ground.

Dredging in West Africa experienced the difficulties encountered elsewhere in the tropics. The bad sanitary conditions had a considerable

effect upon costs. In one instance much time was lost by reason of continued high water. Buried timber was also an obstacle.

In Australia, dredging was carried on as formerly. The water-power dredge, which is operated by impact water wheels, the water being conducted upon the dredge through an articulated pipe, worked satisfactorily. Dredges have failed on certain areas in Australia because of their inability to excavate difficult material or bedrock. It is probable that if large, powerful dredges of the California type were installed on some of these areas success would be achieved. Some of the California dredges are now operating in ground so hard that it cannot possibly be dredged at a profit by small, weak dredges, such as are generally used in Australia. It is well to compare the cost of a large California dredge, about $150,000, with that of the type used in Australasia, about $35,000.

In the Philippines the industry had its vicissitudes, but it is hoped that changes and improvements in the dredges will result in better working. Difficulties due to tropical environment are experienced in the Philippines as elsewhere in the torrid zone.

Another dredge was installed in Dutch Guiana and it is said that although it operated in rich material, mechanical difficulties caused much trouble. This dredge was manufactured in Europe and is lighter and much less powerful than the California type. Two dredges were installed in Brazil, where it is expected that both gold and diamonds will be extracted. Neither of these dredges has, so far as I know, worked profitably as yet. In Tierra del Fuego, dredging seems to have been successful in but few operations. In several cases, machines of poor design and material were installed. No dredges of the California type have yet been built in Tierra del Fuego. In one instance, peat excavated from the overburden ahead of the dredge is said to have been used as fuel. English coal at more than $15 per ton has been generally used. The dredging season is about 10 months in duration.

Review of Technical Progress.

Electric Elevators.—The electric elevators used in the Klondike in 1908 may be classed with dredges. The machines were installed in the creek beds where it was thought that dredging could not be prosecuted advantageously. The elevators consist of a chain of buckets mounted on a ladder, the buckets being heavier than the stacker buckets of a dredge because they must stand the strain of excavating material from a sump. The material is conducted to the sump by driving with a stream from a hydraulic nozzle. Two centrifugal pumps are used to raise the excess of water that the buckets do not lift. The method, which is a form of open-cut mining employing sluices for saving the gold, allows the sun

and air to do most of the thawing; it also exposes the bedrock and allows it to be thoroughly cleaned by hand. The elevators are not mobile, and since they must be moved occasionally they involve a loss of time.

A machine of this type used in Australia has some advantages in the matter of mobility. Material is hydraulicked into cuts which conduct it to a sump; it is not driven to the sump as in Klondike. It is raised by a centrifugal pump to the sluice, the pump being placed on a barge. When it is advantageous to move, the pit is flooded, the barge floated to a new position, the water pumped out, the barge settling on bedrock, and the work is resumed. This arrangement gives fair mobility. For the Australian plant, a working cost of 4.6 to 5.6c. per cu.yd. is given when gravity water is available and 6 to 9c. per cu.yd. when water must be pumped by steam power to the hydraulic nozzle.

It is interesting to note that the operation of the elevators in the Klondike is similar in all essential features to that carried on in 1903 on El Dorado creek, also in the Klondike. Material was hydraulicked with but small pressure to a bucket elevator operated by steam. A centrifugal pump run by steam handled the excess of water. This plant operated on a small scale, with but little water under low head, and the fuel cost was heavy. Its operation was discontinued and other methods were applied. The new operations with elevators will have plenty of water under high head, which will permit working a large pit, thus exposing large areas to the thawing action of sun and air. The use of electric power on the elevator and pump will also effect a great saving as compared with steam power. This enterprise is a good instance of what may be accomplished by a powerful corporation working on a large scale.

The open-cut method most commonly used by the one-man undertaking of recent years in the Klondike and the so-called elevator method recently installed possess several similar features. By the old methods even stripping was effected by ground sluicing, by means of horse or steam scrapers or in some other way. The bedrock gravel was shoveled into cars or skips or other receptacles and elevated to the sluices. As now worked the stripping and transportation of gravel to the elevator are effected by hydraulicking and the elevator lifts it to the sluice. Had a supply of water been available elevators would have been used very generally in the early days.

Design and Construction.—There were no radical changes in dredge design and construction during 1908, but there were improvements in details of design and manipulation. The close-connected bucket was again found to work more satisfactorily than the open-connected type. In several cases in which 3-cu.ft. close-connected buckets were substituted for 5-cu.ft. open-connected buckets, considerable gain in capacity and

saving in power were noted. This was particularly noticeable where indurated material was dredged on a headline anchorage. Considerably less wear, tear and breakage was also noted, for there was much less surging with the close-connected bucket. Heavier construction is more common and the limit does not seem to have been reached in this respect.

No gold dredges with steel hulls have yet been built in the United States. Wooden hulls have not been satisfactory, the life of the wooden hull in California being only about 10 years. Wooden hulls also become much distorted in a few years. To prevent distortion more structural steel has been used and longitudinal and lateral trusses are now incorporated. Time alone can determine whether these trusses will prove effective. Steel hulls would probably be more rigid and more durable than those of wood, and would also be more costly. There seems to be no immediate prospect of a dredge with a steel hull being built in the United States. Steel hulls should be more generally used in tropical climates where wood-boring insects and decay make the life of a wooden hull a short one. Easier transportation of the steel plates than lumber is also an important consideration where inaccessible regions are concerned.

The respective advantages of quicksilver riffles and cocoa matting were better determined during 1908 than ever before. It is now generally accepted that under normal conditions, where there is a clean dredge pond and clean gold, little difference in the respective percentages of saving will be noted. When the dredge pond is dirty, as generally occurs with dredging not conducted in a river, the quicksilver riffle is superior. Where there is gold that will not amalgamate, cocoa matting is the better. Considerations of convenience in cleaning up the gold-saving areas often govern the choice.

Costs.—Dredging costs in the United States did not vary materially in 1908 from those of former years. Operations on a larger scale in Klondike have lowered expenses. No work has yet been done in tropical South America with a first-class modern dredge so that no figures are available for comparison. It has been variously estimated, by competent authorities, that for an operation involving the dredging of about 1,000,000 cu.yd. per year, where transportation is not difficult, a working cost of 10 to 25c. per cu.yd. may be anticipated. To one familiar only with dredging in California and having no knowledge of the disadvantages of tropical environment, these figures will seem excessive; but until dredging is conducted on a considerable scale in the torrid zone, it will be advisable to allow a working cost of not less than 10c. per cu.yd. as the minimum figure.

Small Dredges.—The need of a small dredge for the exploitation of areas not suitable for a large dredge is recognized in some quarters. The tendency since the introduction of dredges in California has been to build larger dredges with larger buckets. To be sure there was a time after the installation of the first 5-cu.ft. close-connected bucket when several 3-cu.ft. close-connected bucket dredges were installed. This was because of constant difficulty with the first 5-cu.ft. close-connected bucket resulting from bad design and weak construction. Recent practice has approved the use of larger buckets.

The small bucket has come into disfavor largely because it was used as at Oroville in ground that was essentially suitable to working with large buckets. There is a distinct demand for a small dredge suitable to work inconsiderable areas or on areas of such shape and irregularity as to make it difficult to manipulate a large machine. There is also the consideration of cost which is a vital one. A dredge with 5-cu.ft. buckets costs about $100,000; a dredge with 2½-cu.ft. buckets should cost about $30,000. The capacities of these dredges should be about as three to one. The record of failures has given the small bucket a bad name that it does not merit. There are areas in the tropics where the small dredge can be used to good advantage. Its light weight makes transportation, installation and operation easy and cheap; high gold content will compensate for the small capacity. By its means shallow ground of comparatively small acreage can probably be made the basis of a company when the first cost alone of a large dredge would make final profit an impossibility.

The small dredge also has a function as a means of prospecting ground preliminary to the installation of large dredges. Small dredges have been used in this way in but few instances. Stewart river, in Canada, was prospected by this means in 1903. However, the common way of prospecting is to use steam or hand drills and results can thus be attained that are reliable and detailed. Drilling is cheap by comparison with dredging and a large acreage can be drilled so as to get detailed information at a much lower figure than the first cost of the prospecting dredge. The sectionalized gold dredge for muleback transportation is yet to be designed. The small dredge naturally will be more easily arranged for such transportation than larger and heavier dredges.

Protection Against Theft.—Gold dredging has been conducted with an unusual confidence in the honesty of the men who work on the dredges. On only a few dredges are there devices to prevent theft of gold from the gold apparatus. During 1908 it was found that considerable amounts of gold were being stolen from dredges in California. This incident will probably lead to the incorporation of devices to keep under lock that part of the apparatus where most of the gold is caught. This can be

done easily and it will be good practice on all dredges. Even if not necessary, such procedure will be no more than a good business precaution.

CYANIDATION DURING 1908.

CHARLES H. FULTON.

The most noteworthy feature of cyanidation in 1908 was the increased application of fine grinding and the treatment of the finely crushed material by means of vacuum filtration, chiefly by the Butters, Moore, and Ridgeway filters. Development along these lines has been aided by the rapidly extending use of the Brown air agitator, which fills the gap in the treatment between grinding and filtration in a particularly happy manner. Now that it has been practically perfected it seems a very simple device, which is its chief merit. No particular trouble *seems* to be experienced in treating the very finely ground mixed product of sand and slime by the vertical vacuum filtration methods. The Merrill press is being installed for filtration in a number of places where the nature of the ore is suitable. A persistent effort is being made, with a marked degree of success, to treat concentrates by cyanidation. In Mexico, where cyanidation is now applied in many different districts to silver and silver-gold ores, concentration is a distinct feature, and the direct treatment of the concentrates a matter of much moment to the mines. Some of the newer Mexican mills show a decided complexity of operation, but already the tendency is made manifest toward simplifying the treatment, and curtailing the elaborate concentration methods, by not concentrating so closely and frequently, but relying on the effectiveness of thorough agitation of the pulp by Brown agitators to gain extraction from the mixed finely ground sand and slime. The large amount of solution necessary for successful cyanidation in the mills treating silver ores has led metallurgists to begin looking for a method of precipitation less costly in precipitant than the zinc shaving method, and zinc dust is being used in some of the larger mills. It would seem as if the time were ripe for working out an electrolytic method for the precipitation of silver solutions.

Considerable work has been done on the chemistry of silver cyanidation, but much of it is still obscure.

In view of the fine grinding practiced, slime treatment has of course received the greater part of the attention, but progress has been in the line of perfection of the filters now in use rather than in the invention of new filters, except that of the Hunt continuous sand-bed slime filter, which is an ingenious device of much promise, inasmuch as it is designed to treat both fine sands and slimes as one product. It would seem as if

horizontal filters of this and the Ridgeway type were better suited to the mixed fine ground product than the vertical basket type of vacuum filters.

New Milling Practice.

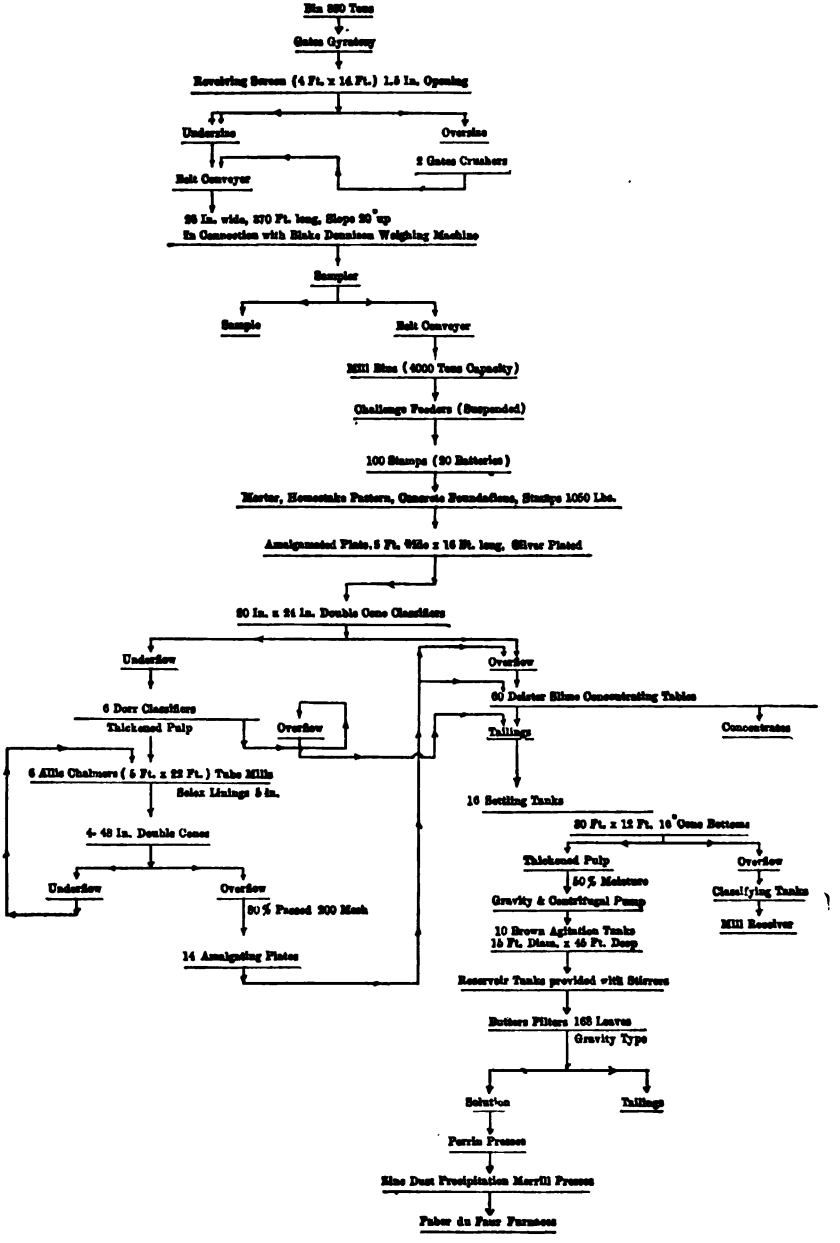
UNITED STATES.—The new 100-stamp mill of the Goldfield Consolidated Mines Company,[1] Goldfield, Nev., began operations on Dec. 26, 1908. The ore is a silicified dacite, containing from 50 to 75 per cent. SiO_2 as quartz; kaolinite, alunite (a hydrated sulphate of aluminium and potassium), and pyrite making up the remainder. Small amounts of bismuthinite and tetrahedrite are present in some of the Goldfield ores, but absent in others. Gold occurs partly in the native state, usually very fine, but some coarse, and partly associated with tellurium. Practically no silver is present. No zinc or lead minerals are found, but chalcopyrite in very small quantities is noted in some ores. The mill is two miles north of the mines and the ore will be transported over a railroad in 25-ton hopper-bottomed cars discharging into bins. The scheme of treatment is outlined in the accompanying chart. The cost of the mill is approximately $760,000, and with water supply and other items, about $900,000. An interesting feature is an adjunct to the mill in which it is expected to treat the concentrates from the Deister tables by means of sulphuric acid to remove base metals, and cyanide the residues after neutralization with alkalies. No details of this treatment are available. A milling cost of $2.75 per ton is calculated. This mill was designed on the data and experience gained in the Combination[2] mill of the same company.

There was an interesting discussion during 1908 concerning the relative merits of wet crushing by stamps, concentration, and cyaniding slimes and sands versus dry crushing in ball mills, roasting all the ore, and cyaniding sands and slimes, on Goldfield ores, with the advantages in favor of the first method.[3]

The tests on ore from the lower levels of the Mohawk and Combination mines made at the Combination mill in order to get data for the new mill described gave the following results: This ore represents the character of the material that the new mill will have to treat for a long time to come. On a test of 477 tons, based on the general plan as outlined in the mill scheme, the results were: Ore value, $36 per ton; extraction by amalgamation, 26.82 per cent.; extraction by concentration (3.85 per cent. of total weight of ore), 33.04 per cent.; extraction by cyanidation, 32.91 per cent. Total, 92.77 per cent. (22 per cent. of total weight =sand; 74 per cent. of total weight=slime.

[1] P. E. Barbour, *Eng. and Min. Journ.*, LXXXVI, 467; T. A. Rickard, *Min. and Sci. Press*, XCVI, 842; J. H. G. Wolf, *ibid.*, XCVIII, 12.
[2] *The Mineral Industry*, XV, 423.
[3] A. G. Kirby, *Min. and Sci. Press*, XCVI, 836; Lochiel King, *ibid.*, XCVI, 123.

A feature of the method of treatment is the attention paid to amalgamation and that concentration is attempted only on very fine mate-



FLOW SHEET OF GOLDFIELD CONSOLIDATED MILL.

rial, viz., finer than 150 mesh. The essentials of the treatment method are as follows: 1. Coarse battery screens, to discharge true slimes as soon as made and not unduly crush hard ore particles, in order to get battery capacity. 2. Tube mill crushing for material coarser than 150 mesh. 3. Large area for amalgamation; two amalgamations to insure removal of free gold from concentrates. 4. Large area for concentration, to remove thoroughly sulphides from the pulp. 5. Concentration of finely crushed material only, so as to thoroughly remove sulphides. 6. Thorough dewatering of slimes before agitation with solution, in order to get rid of mill water containing reducing salts, which interfere with the solution of the gold. Frequent renewal of solutions during treatment to insure a supply of oxygen. 7. The concentrates from the mill may be treated by roasting followed by cyanidation.

Nevada Goldfield Reduction Company, Goldfield, Nevada.—This company has a small custom mill of 100 tons per diem capacity, in connection with a customs sampling plant.[1] It treats both the oxidized and unoxidized ores from the Goldfield district, which naturally vary considerably in character. The ore is rough-crushed in the sampling mill and passed to twenty 1250-lb. stamps, crushing through 16-mesh screens, in 0.1 per cent. KCN solution. The pulp flows over amalgamated plates, and thence to eight Wilfley tables which make three products, viz., concentrates (with a value of $200 to $1000 per ton), sands and slimes. The sand is reground in a 5x22-ft. Allis-Chalmers tube mill, the pulp flowing over a second set of amalgamated plates at the head of which it is joined by the slime from the Wilfley tables. The plates discharge the pulp to a V-shaped classifier, making sands and slimes. The sands are reconcentrated on vanners, the tailings from which go to eight 22x5-ft. sand vats, arranged in superimposed rows for double treatment. The time of leaching is approximately 20 days. The slimes, after a concentration on slime vanners, go to four agitator vats, size 22x20 ft., in which they are agitated by mechanical stirrers and centrifugal pump. From these vats they go to storage tanks and a filter of the Butters type. Solutions are precipitated by zinc thread, and the precipitates treated by sulphurous acid, instead of sulphuric acid. The advantages claimed for this method are (1) cheapness, (2) no noxious fumes, (3) sulphurous acid dissolves lime and copper, making for a purer bullion. The sulphurous acid is made by burning sulphur and passing the SO_2 into the agitation vat containing the zinc precipitates in water. In tube milling, Danish pebbles were formerly used, but are now replaced by close-grained hard ore which makes a good substitute and effects a considerable saving.

[1] T. A. Rickard, *Min. and Sci. Press*, XCVI, 811; E. S. Leaver, *ibid.*, XCVII, 254; *ibid.*, XCVI, 411.

Pittsburg-Silver Peak Gold Mining Company.—In this mill[1] at Blair, Nev., 30 miles west of Tonopah, the ore is rough-crushed, by one No. 6 and two No. 3 McCully gyratory crushers, to a 1.5-in. ring, at the mine. The ore is conveyed by a bucket tramway 2½ miles long to the mill. The ore is quartz, carrying gold chiefly in the free state, and very little sulphide mineral is present; small amounts only of pyrite and galena are found. At the mill crushing is done by one hundred 1050-lb. stamps, making 104 seven-inch drops per minute. Needle-slot screens are used with an aperture equivalent to 35 mesh. Each 20 stamps are driven by one 50-h.p. electric motor. The pulp flows over amalgamated plates, where from 60 to 70 per cent. of the value of the ore is recovered. From the plates the pulp flows to two large settling cones, producing sands and slimes; the sands are further classified in a second set of 10 cones, which are of the Merrill type.[2] Sands are treated by percolation in sand vats, and the slimes are filtered in Merrill filter presses. Precipitation is carried out by zinc dust in Merrill triangular presses.[3]

COLORADO.—In this State the Golden Cycle mill was in operation during 1908, successfully competing with the chlorination plants. The treatment rates in effect at the beginning of 1909, were as follows:

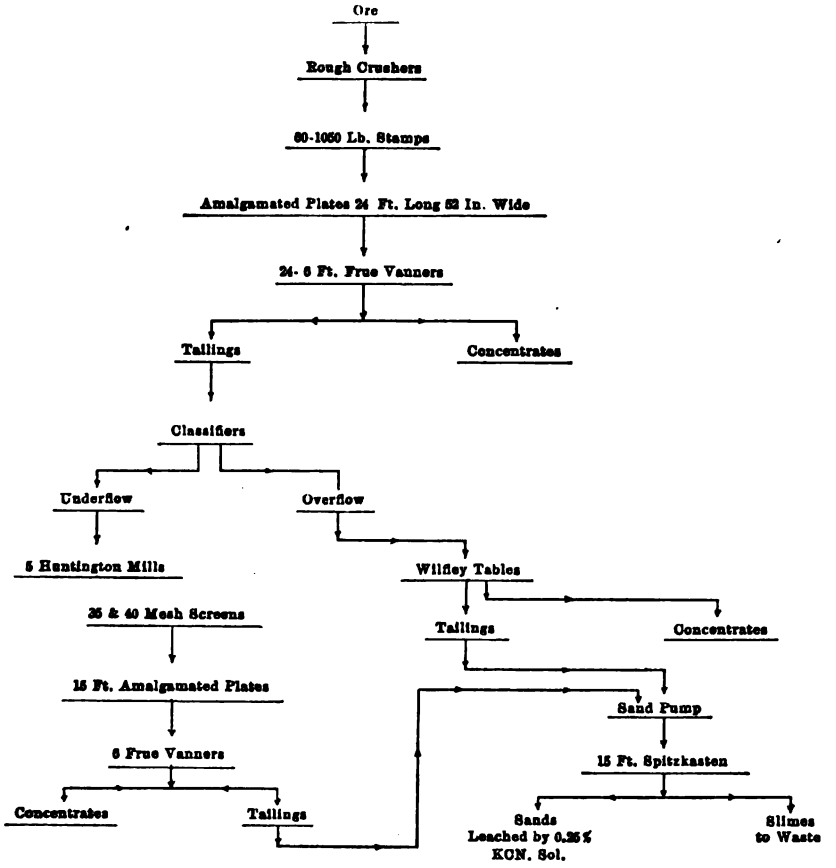
	GOLDEN CYCLE MILL. (Treatment and freight Contract rate.)	U. S. R. & R. Co. (Treatment and freight open rate, no contract.)
Ore up to 0.5 oz. gold	$4.50	$4.50
Ore up to 0.75 oz. gold	5.25	5.25
Ore up to 1.00 oz. gold	6.00	6.00

This recent change makes the rate for the two processes identical, except that one is an open rate and the other a contract rate. While the Golden Cycle rate was constant during 1908, the rate of the U. S. Reduction and Refining Company fluctuated, being lower than the Golden Cycle at the beginning of the year, then raised to a higher figure in October, and then again reduced to the present figure in December. The Portland Mining Company is erecting an experimental cyanide mill with a view of treating its large waste dumps. It is probable the large mill erected for the treatment of the Independence dumps will soon begin operations.[3]

Cyanide Annex of the Standard Chlorination Plant.—This mill at Colorado City treats the low-grade tailings of the chlorination plant, accumulated in former years as well as those discharged daily from the

[1] J. H. G. Wolf, *Min. and Sci. Press*, XCVIII, 13; *ibid.*, XCVI, 415.
[2] *The Mineral Industry*, XV, 418.
[3] W. W. Travell, *Min. and Sci. Press*, XCVIII, 42

chlorination barrels.[1] All the material has been dry-crushed through 12-mesh screens and roasted. The tailings from the barrels are dewatered in Dorr classifiers, the slimes going to waste and the coarse product to tube mills for regrinding. The old tailings are carried by belt conveyers to the tube mills without any classification. Lime is added to both types of tailings as they are charged into the tube mill. The eight tube mills are 5x22 ft. in size, and grind the tailings in weak cyanide solution, the pulp all passing an 80-mesh screen. The discharge from the tube mills goes to slimes settling vats, where the slimes are treated by agitation and decantation.



FLOW SHEET OF CAMP BIRD MILL.

Camp Bird Mill.—The present practice at this mill near Ouray, Colo., is described in the scheme outlined above.[2] The concentrates from the

[1] F. L. Barker, *Mines and Minerals*, XXVIII, 481.
[2] Thomas T. Read, *Min. and Sci. Press*, XCVII, 668.

tables (about 10 per cent. of the weight of the ore treated and containing 20 per cent. of the total value of the ore) assay 9 to 10 per cent. Pb, 12 to 15 per cent Zn, 11 to 15 oz. Ag, and 2.5 to 4 oz. Au per ton, and are shipped to smelters. The total recovery of gold from the ore is 95.5 pr cent. The cost of cyaniding the tailings from the tables is 70c. per ton. The former mill was wrecked by a snow slide in March, 1906, and shortly after was completely destroyed by fire. The present mill was put in commission at the beginning of 1907. All power is electric, furnished by several power companies.

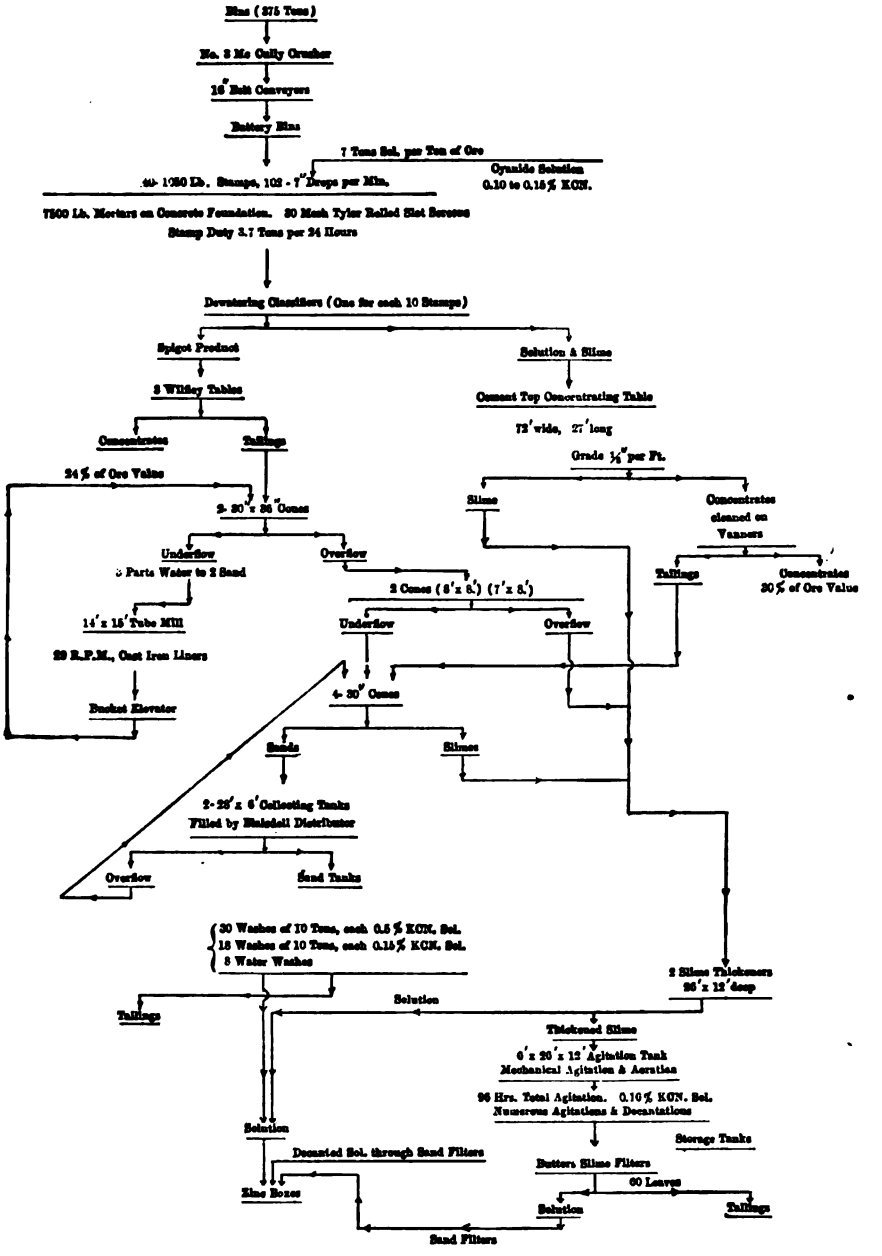
MEXICO.—The San Prospero mill at Guanajuato is one of the plants of the Mexican Milling and Transportation Company.[1] It began operations in November, 1906, and is treating typical Guanajuato silver ore,[2] at present mainly custom ore. It exemplifies good present practice, and is of particular interest as it is at this mill that experiments are being carried on to treat by cyanidation the rich silver concentrates produced at the different plants of the Guanajuato Development Company and the Mexican Milling and Transportation Company. This experimental plant treats concentrates by fine-grinding them, then by agitating in small Pachuca tanks, and then filter-pressing and precipitating silver from solution by zinc dust in Merrill presses. The scheme of treatment at the San Prospero mill is outlined in the accompanying chart. An interesting feature are the data as to extraction gained in the battery and classifiers, etc., by crushing in cyanide solution, before the ore reaches the cyanide department of the mill, as follows:

	Gold %	Silver %
Extraction in the batteries	31.2	4.8
Extraction between batteries and concentrators	5.4	5.6
Extraction between concentrators and sand plant	21.0	5.0
Total extraction before reaching cyanide department	57.6%	15.4%

San Rafael Mill, Pachuca, Mexico.[3]—This is the latest type of cyanide plant for the treatment of silver ore, in Mexico, using the all-fine-grinding and Pachuca or Brown vat agitation process. The ores at Pachuca are essentially silver ores, carrying only 5 grams of gold for each kilogram of silver. The average value of the ore milled in the district is 22.5 oz. Ag and 0.11 oz. Au per metric ton. The silver occurs as chloride or bromide in the oxidized ore and chiefly as argentite in the sulphide ore. Few other sulphide minerals of silver are present. The ore carries a con-

[1] T. S. Butler, *Min. and Sci. Press*, XCVII, 130; C. T. Rice, *Eng. and Min. Journ.*, LXXXVI, 949.
[2] *The Mineral Industry*, XVI, 536.
[3] Claude T. Rice, *Eng. and Min. Journ.*, LXXXVI, 654.

siderable amount of manganese and a little copper. The mill is equipped with forty 900-lb. stamps and twenty 1250-lb. stamps, crushing 300 tons



FLOW SHEET OF SAN PROSPERO MILL.

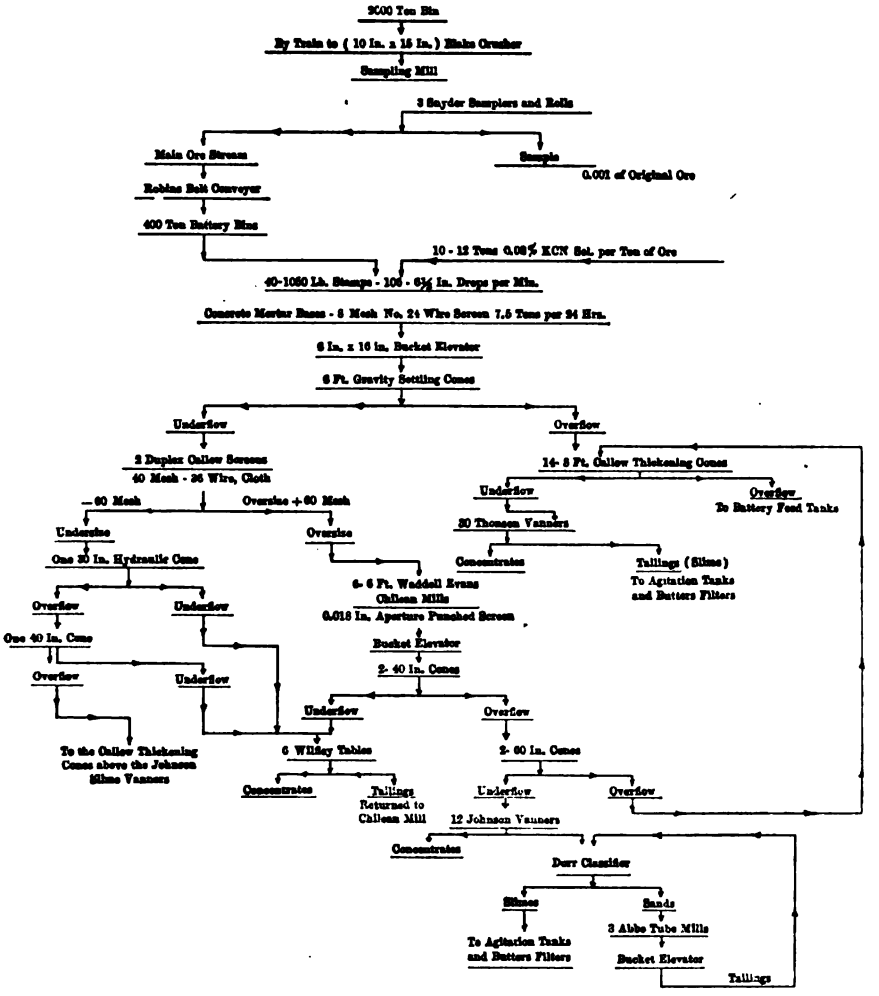
per day in weak cyanide solution through 20-mesh, No. 28 wire, screens. The pulp passes to classifying cones, the coarse product from which is passed to Wilfley tables, and the overflow to Dorr slime thickeners. These thickeners consists of a tank with a flat conical bottom on which the slime settles, and along with it is scraped by a series of plows attached to revolving arms, conveying the slime toward the central discharge at the apex of the cone. The tailings from the Wilfley tables after thickening are ground to pass 200-mesh in three 20x4-ft. Krupp tube mills, and join the thickened slime from the Dorr thickeners, passing to ten 45x15-ft. Brown agitation vats in which an 80-ton charge is agitated for 48 hours. From the agitation vats the pulp is passed to three storage tanks, partly by gravity and partly by centrifugal pump, and is then fed to Moore filters, two independent units of 100 leaves each. Zinc shavings are used for precipitation.

Guerrero Mill of the Compañia Minera de Real del Monte y Pachuca.[1] —This mill at Real del Monte, near Pachuca, was recently put into operation on silver ore of the same kind described for the previous mill. The accompanying flow sheet shows the scheme of operations. The concentrate obtained from the various tables has this composition: 10 per cent. Zn; 1 to 4 per cent. Pb; 0.8 to 1 per cent. Cu; 20 per cent. SiO_2 (insoluble); the rest chiefly pyrite. It assays from 250 to 1250 oz. Ag, and 1.20 to 6 oz. Au per metric ton and constitutes from 1 to 3 per cent. of the weight of the ore, and contains approximately 45 per cent. of its value. It is shipped to smelters for treatment. The slimes are agitated in twenty-four 10x30-ft. flat-bottomed treatment vats by means of mechanical stirrers and also with a centrifugal pump. The consistency of the sludge during agitation is 3.5 parts of 0.2 per cent. KCN solution to one part solid; the solution is decanted several times and replaced by new solution. The slimes are filtered by the Butters method. Lead acetate is added in small quantities during slime treatment to precipitate soluble sulphides. Precipitation is carried on by zinc thread. The consumption of chemicals per ton (2240 lb.) is NaCN, 2.64 lb.; Zn, 3.08 lb.; lead acetate, 0.726 lb.; lime, 5.50 lb.

It is of interest to compare the practice at this mill with that at the San Rafael, previously described, and also with the new mill of the Goldfield Consolidated company. The Guerrero mill at once stands out for its complex practice of concentration, following standard ideas of gradual crushing, classification and concentration, gaining the concentrates in as coarse a state as possible, and insisting on as complete a removal of the sulphides as possible, before cyanidation takes place. The San Rafael mill takes out but comparatively coarse concentrates, and per-

[1] Claude T. Rice, *Eng. and Min. Journ.*, LXXXVI, 648.

mits the fine concentrates to pass to the Pachuca agitation vats, where extraction is gained by cyanidation. The Goldfield Consolidated company's mill grinds very fine before attempting concentration, even tube-



FLOW SHEET OF GUERRERO MILL.

milling before concentration, but attempts the complete removal of the sulphides. Each three mills treat all of the ore as slimes, no sand percolation being practiced.

Rio Plata Mining Company.[1]—At the mill of this company in the Sierra Madre Mountains, in the southwestern portion of Chihuahua,

[1] H. J. Baron, *Eng. and Min. Journ.*, LXXXVII, 147.

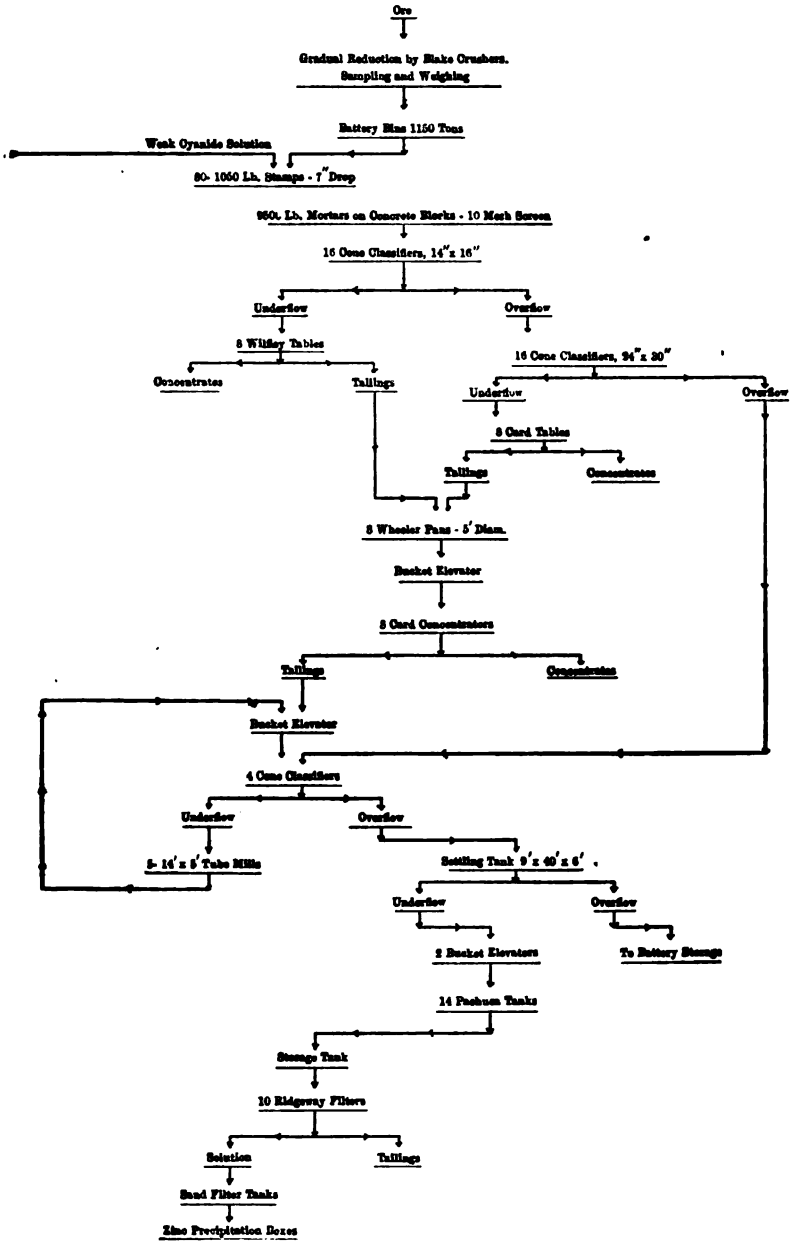
the ore is rough-crushed by two Blake crushers whence it passes to twenty-five 1000-lb. stamps, with 108 six-inch drops per minute, crushing in 0.1 per cent. KCN solution through 24-mesh Taylor woven wire screens. The battery pulp goes to a gravity settling cone, the underflow from which is concentrated on five Card tables. The overflow passes to two Wilfley slime tables and one Frue vanner. The tailings from the Card tables are reground in eight 5-ft. Wheeler pans, so as to pass a 150-mesh screen, and joining the tailings from the Wilfley slime tables and the Frue vanner, pass to two gravity settling cones, whence the slime is transferred to Pachuca agitating tanks. The overflow from the gravity settling cones goes to a 10x10x16-ft. Dimmick classifier, the clear overflow from which is pumped to the battery solution storage tanks. The four Pachuca vats are each 45 ft. deep and 15 ft. in diameter. The pulp is agitated for 12 hours with a 0.25 per cent. KCN solution, and then passed to a 28-leaf, 6x10-ft. Moore filter. The solution from the Moore filter is passed through sand filter tanks, and thence to zinc thread precipitation. All machinery in the mill is driven by electric motors. The ore is a blue quartz, containing some native silver, sulphide and bromide of silver and pyrite; very little gold is present. The concentrate amounts to 3.5 per cent. of the ore, but contains 50 per cent. of the value. It contains on the average 1500 oz. Ag and 0.25 oz. Au per ton.

Veta Colorada Mining and Smelter Company.[1]—The ore of this company, at Parral, Mexico, is highly silicious, and contains practically no gold. The silver occurs as argentite and as cerargyrite. There is about 1 per cent. Pb, 1 per cent. Zn, 2 to 3 per cent. of pyrite, and a trace of copper in the ore. The scheme of treatment is shown in the accompanying flow sheet. The mill presents an interesting feature in that both Wheeler pans and tube mills are used for regrinding, the former doing the coarser regrinding.

AUSTRALIA.—*The Bromo-Cyanide or Diehl Process at Kalgoorlie.*[2]—At the Hannan's Star mill, the ore from the rock breaker is dry-crushed in two No. 5 Krupp ball mills through 27-mesh steel wire screens. From the mills it is passed to a mixer and thence over an amalgamated plate to a classifier, the underflow from which passes to a tube mill, while the overflow goes to two sets of spitzlütten. The product of the tube mill is returned to the classifier. The underflow from the spitzlütten flows back to the tube mill, while the overflow goes to pointed settling boxes for the thickening of the slimes. The whole product, except 5 per cent., passes a 150-mesh screen, which is the necessary fineness for further treatment. From the thicken-

[1] C. T. Rice, *Eng. and Min. Journ.*, LXXXVI, 120.
[2] E. W. Nardin, *Min. and Sci. Press*, XCVII, 562; see also G. P. Williams, *Journ. Chem. Met. and Min. Soc.*, VIII, 232; and *Trans.* Australasian Inst. M. E., XII, 1907.

ing boxes the slimes go to four agitation tanks, each holding an amount of pulp equivalent to 50 tons of dry slimes. The exact tonnage in each



FLOW SHEET OF VETA COLORADA MILL.

tank is determined by measuring the volume of pulp in the vat and weighing the contents of a bottle filled to a containing mark, determining the specific gravity, from which data can be calculated the exact tonnage in the vat. This is necessary as the amount of bromocyanogen added is on the basis of this tonnage. A vat of slime is then charged with cyanide solution and agitated for 16 hours, when a dip sample is taken, and then the proper amount of bromocyanogen, dependent upon the gold in the residue from a former vatful, added and agitation continued for four hours more; a quantity of lime is next added to the pulp which is then transferred to the filter presses. It is useless to add bromocyanogen at the beginning of the agitation as it is comparatively rapidly destroyed. The proper time to add it is when no further extraction can be gained by plain cyanide solution. Before adding the bromocyanogen, the amount of which is not stated, it is essential to neutralize some of the alkalinity of the pulp solution by the addition of H_2SO_4 in order to prevent the rapid destruction of bromocyanogen to no purpose. The mill solutions have an alkalinity of from 0.02 to 0.03 per cent NaOH in lime equivalent; this is cut down to 0.01 per cent by H_2SO_4 just before the addition of the bromocyanogen and after the final agitation, when the pulp is ready to go to the presses, is raised again to 0.03 per cent. by the addition of lime, although this is said to have a slight effect of reprecipitating gold from solution.

The bromocyanogen is made at the mill. The necessary chemicals are: (1) potassium bromate and bromide salts, containing the proper relative percentage of the two salts, shipped from Hamburg, Germany; the proportion of bromine as bromide to the bromine as bromate is as 2:1; (2) 63 deg. sulphuric acid (chamber acid); (3) potassium cyanide (93 per cent. KCN). In order to make 30 lb. of BrCn, it takes 50 lb. of H_2SO_4; 20 lb. of KCN; and 36.8 lb. of the mixed salts. The reaction is as follows: $2KBr+KBrO_3+3KCN+3H_2SO_4=3BrCn+3K_2SO_4+3H_2O$.

The BrCn solution is made in a small wooden tank, provided with mechanical stirrers and a close-fitting cover, and holding about 200 gal. In making up a charge a portion of water and all the H_2SO_4 are mixed and then allowed to cool, the KCN, dissolved in a separate vessel in sufficient water to fill the stirring tank, is then poured in and at the same time the mixed salts are gradually added. The whole is agitated for six hours before the maximum amount of BrCn is developed. The cost of 30 lb. of BrCn made in this way is $21.82, or 73c. per pound. It was found that the presence of very finely divided iron in the ore, such as is produced by the wear and tear of grinding machinery parts, is very destructive to bromocyanogen. The same is true of very finely ground pyrite.

SOUTH AFRICA.—During July, 1908, there were in operation on the Rand 8500 stamps, and 86 tube mills, which crushed 1,527,000 tons of ore yielding 561,703 oz. (fine) of gold. There were 63 distinct companies operating, the average recovery from a ton of ore being $7.22, of which 62 per cent. is recovered by amalgamation and 38 per cent. by cyanidation.[1] It is expected that during the year, 735 stamp and 12 new tube mills will have been added to the Rand Equipment.[2] In September, 1908, there were 8625 stamps and 102 tube mills at work, and 1,534,755 tons of ore were milled.[3] The new mill of the Simmer Deep,[4] which began crushing in September, has two hundred 1670-lb. stamps. The mortar boxes rest directly on concrete foundations, with no anvil blocks and but a 0.25-in. rubber sheet between the box and the concrete. Each unit of 10 stamps is driven by a 40-h.p. motor, independent of countershafting. The pulp below the plates is classified by cones, instead of pyramidal boxes, as has been customary, the underflow going to four 5½x22-ft. tube mills. The overflow from the cones passes to 10 collecting vats, 50 ft. in diameter. The sand treatment vats are somewhat larger. The slimes are treated by agitation and decantation in 13 conical vats, 70 ft. in diameter, 12 ft. high at the sides, and with a total depth of 17.5 ft. in the axis.

For the fiscal year ending June, 1908, the Simmer & Jack mill crushed with 320 stamps 785,310 tons of ore, an average duty of seven tons per stamp. The distribution of recovery is as follows:

	Tons.	Oz. gold Recovered.	Per Cent.
Battery, by amalgamated plates	785,310	165,526	54
From shaking amalgamation plates below tube mills	332,053	43,939	14
From sands, by cyanidation	547,737	75,437	25
From slimes, by cyanidation	237,573	16,343	7

At the Luipaards Vlei Estate,[5] the 60-stamp mill in April, 1908, crushed 14,720 tons, making a duty of 9.26 tons per stamp. The stamps weigh 1550 lb. This is the best on record on the Rand.

Slime Treatment.

An important feature of 1908 was the increasing number of mills that grind all the ore fine and treat it by agitation and slime filtration methods. This feature is particularly noticeable in the treatment of silver

[1] *Min. and Sci. Press*, XCVII, 557 and 803.
[2] *South African Min. Journ.*, Sept. 19, 1908, 802.
[3] W. F. Wilkinson, *Eng. and Min. Journ.*, LXXXVII, 129.
[4] *Min. and Sci. Press*, XCVII, 212 and 850.
[5] *Min. and Sci. Press*, XCVI, 831.

ores in Mexico. The Brown agitation[1] vat, known also as the Pachuca tank, is widely applied in this connection for the agitation and aeration of the slimes and fine-ground sands, and is at present the most efficient and economical agitation device in use. The Brown vats are constructed almost invariably 45 ft. high and 15 ft. in diameter, of $\frac{3}{16}$-in. steel. Internally, the bottom of the tank terminates in a cone with a 60 deg. slope. The weight of the tank and contents is carried by the continuation of the outside cylindrical shell, stiffened with vertical angle bars, which are rivetted to a double angle iron resting on solid rock or concrete foundation. The lower end of the inside cone is closed by a cast-iron bottom. Manways in the lower part of the cone permit of entrance, when empty, for the inspection of valves, etc. The tank is discharged through a pipe and valve in the bottom of the cone part. The internal fittings of the tank consist of: (1) A central tube or air lift, one-twelfth the diameter of the tank. It is open at both ends and extends from about 18 in. above the charging level to the same distance from the lowest point of the cone, and is supported by means of brackets. (2) An air pipe in the center of the lift, closed below and resting on the bottom of the tank, has at the level of the inlet of the air lift a sleeve valve preventing the pulp entering and allowing the air to escape upward when the air pressure exceeds that due to the head of pulp in the tank. (3) A second air pipe outside of the lift which serves to keep the pulp in gentle motion while filling or emptying the tank, when the amount of pulp in the vat is insufficient to allow the air lift to work. (4) An apparatus consisting of an annular casting surrounding the air lift, supported on the sides of the cone and provided with a number of pipes having sleeve valves discharging against the cone, the apparatus being connected to the air, water, and solution pipes. This arrangement is useful in washing down sands which have settled on the cone after agitation has stopped and the tank has been discharged.

The method of operating the tank is as follows: The tank is filled with pulp, air is admitted to the pipe in the lift, which mixing with the contents of the lift lightens the column inside and causes it to overflow at the top, while other pulp runs in at the bottom. The initial air pressure has to exceed the hydraulic pressure in the column, but when once circulation has been established the pressure may be considerably reduced. For commencing operations, 50 lb. air pressure is usually employed, but when the whole contents of the tank are in rapid circulation 25 lb. of pressure is sufficient. The amount of air used depends upon the proportion of slimes and sands, the fineness of the pulp, and the ratio of solid to liquid in the pulp. In most cases, 100 cu.ft. of free air per minute will

[1] *Mex. Min. Journ.*, Jan., 1908, p. 20, and E. O. Daul, Oct., 1908, p. 16; F. C. Brown, *Min. and Sci. Press*, XCVII, 424.

keep the mass in lively motion and prevent the settling of sands. The consumption of power depends on the factors just stated. It will vary from 1 h.p. for slimes to 10 h.p. for slimes and sands with a unit of 100 tons per day. The agitation and aeration are very good. Even after weeks of rest, when the contents of the tank have packed to a hard mass in the cone, agitation can be readily started and after about an hour the mass is in perfect circulation. Sands can be treated as readily as slimes. An interesting comparison between agitation and aeration in Brown tanks and that in mechanical agitators, is made between the practice at the Loreto and the Hacienda San Francisco at Pachuca, Mexico. It is shown that the consumption of power for mechanical agitation at the former mill is 20 times that for air agitation in Brown vats at the latter mill in attaining the same extraction. The time of agitation necessary to attain the maximum extraction is also much lessened, thus 24 hours at the Hacienda San Francisco to 84 hours at the Loreto. The Brown vats furthermore rarely get out of order and require very little repair, which cannot be said for the mechanical agitators. The accompanying table gives data respecting air agitation in the Brown tank.[1] No increased consumption of cyanide has been noted with the Brown vat.

AIR AGITATION IN BROWN TANK.

Size of Tank.		Material.	Charge in tons.	Free Air. Cu. ft. per minute.	Pressure Lb. per sq. in.	Horsepower.
Diam. ft.	Depth ft.					
7.5	37	Slime	15	5	22	0.5
7.5	37	Concentrates	40	17	26	2.0
10	40	Slime	35	9	22	0.75
13	55	Slime	110	16	33	1.75
10	40	Sand	50	25	22	2.25

Adair-Usher Process.[2]—The Usher process as in use at the Crown Reef mine treating 8000 tons per month is described below. The plant consists of the following apparatus: Five collecting vats, 30x10 ft.; five treatment vats, 32x14 ft.; three treatment vats, 50x13 ft.; two filter vats, 20x5.5 ft.; two filter vats, 24x8.0 ft.; one solution pump; three precipitation boxes, 30x6x3 ft. deep. The Usher apparatus in the treatment tanks consists of a 10-in. pipe reaching down at the center axis of the tank and projecting 6 ft. above the top of the tank, with enough 1-in. pipes radiating from the distributor at the bottom and of the large pipe so that the distance between the ends of the pipes at the periphery of the vat is not more than 2 ft. In each pipe $\frac{1}{8}$-in. holes are drilled at an

[1] Mark R. Lamb, *Eng. and Min. Journ.*, LXXXVI, 901.
[2] Alfred Adair, *Journ. Chem. Met. and Min. Soc.* of S. A., VIII, 331; Geo. Melville, ibid., IX, 96; *Eng. and Min. Journ.*, LXXXVI, 463.

angle of 30 deg. to the bottom of the tank with centers varying from 12 in. near the middle of the tank to 3 in. near the periphery. The radiating pipes are placed about 2 in. from the bottom of the tank.

About 100 tons of slime are collected in a vat for each charge, and when settled and decanted the slimes are transferred to one of the treatment tanks with 0.02 per cent. KCN solution obtained by siphoning from the next tank to be discharged, care being taken that at the beginning and during the transference and treatment barren solution is constantly introduced into the large central pipe, and permitted to discharge through the holes in the radiating pipes in the bottom of the tank. After the transfer is complete the charge is circulated for two hours by being drawn off at the bottom by the centrifugal pump and returned over the top for two hours or more, and washing is continued with barren solution until the assay value of the clear solution, overflowing from the tank, is about 20c. per ton when the charge is allowed to settle. The solution above the settled slimes is then used to transfer another charge as stated above. The tailings samples are taken by means of a long tin tube 2 in. in diameter, with a flap valve at the bottom end. The rate of washing is regulated by the settlement of the suspended slimes and the filter capacity, and is as high as possible to shorten the time. With 100-ton charges in the smaller tanks (30x10 ft.) washing is done at the rate of 10 to 15 tons of barren solution per hour, and is complete in 48 hours. Settlement takes 36 hours longer. It is important that every hole in the radial pipes be clear before transferring a charge. This cleaning is done, when necessary, by hose (high pressure water being attached to the periphery end of the radial pipes). The chief advantages derived from the process over the ordinary decantation process are as follows: 1. Lower tailings assays. 2. Less dissolved gold in the residues. 3. Saving in power, and in pump repairs. 4. Reduction in time of treatment, increasing the capacity of the plant. 5. Less work for the shiftmen.

With slimes assaying $2.25 gold, the tailings by ordinary decantation assayed 45c. per ton, as against 30c. per ton by the Usher process. The Adair process as such consists in the addition of umber to the slimes charge, as an oxidizing and settling agent, but it has found but little application; the Usher process, however, as described, was employed by 26 companies on the Rand at the end of 1908.[1] It is used under the name of the Adair-Usher process.

The process is probably chiefly important through the saving of time effected, increasing the capacity of the plant. This amounts to 30 per cent. approximately. There is much difference of opinion as to whether any better extraction is obtained by the method, or whether any saving

[1] Alfred James, *Min. and Sci. Press*, XCVIII, 47.

is made by recovery of dissolved gold above that obtained by the decantation process. The theory of the process is that the rising barren solution shall actually displace gold-bearing solution, and not mingle with it. It is very questionable, however, whether this object is accomplished. It is also essential[1] that once the process is in operation no changes of "head" of the barren solution, or its temperature as it is pumped into the radial pipes, shall occur, as this at once causes distinct ascending streams of solution through the pulp, which have no effect of washing or displacing gold-bearing solution. The method is also stated to increase the bulk of solution to be handled to five to six tons of solution per ton of dry slime.

A recent improvement in the process has been suggested to make it applicable to fine sand and slimes. The improved plant consists of conical-bottom decantation vats arranged in series, in step form, each succeeding cone being 2 ft. above the last one below. Each tank has an outflow at the bottom and an overflow at the top. The entering sand and slime is fed through a distributor near the top; the solution enters through the radial pipes previously described, which are placed in this instance near the middle of the vat. When in operation the pulp slowly settles in the rising current of solution while the enriched solution flows out from one vat to the solution distributor in the next vat below. The pulp as it accumulates in the cone bottom slowly feeds by means of an injector to the pulp distributor at the top of the next vat above. From the highest vat the pulp is discharged to a settling vat or to a filter plant. The solution thus travels from one vat to another by gravity while the sands and slimes travel upward in an opposite direction. It is difficult to see wherein this method has any advantage over thorough agitation in Brown vats, followed by a filtration method. As already outlined, the method is an adjunct to the decantation process as practiced, improving it. Where new installations are being considered it is questionable whether it would serve the purpose as well as the more modern methods.

An interesting method of settling slimes and separating the same from solution is advocated by H. G. Nichols.[2] His idea is to remove the solid matter as fast as it reaches the bottom of the tank in which it is settled.

The effect of doing this is twofold, viz., (1) the solid matter settles very compactly if not allowed to accumulate; and (2) its removal prevents the thickening of the liquid above, which factor is one of the principal causes of the slow and imperfect settlement of slimes. Settling of solid slime particles under these conditions gives the "free settlement" of slimes, eliminating the retarding effect due to an increase of the

[1] *Min. and Sci. Press*, XCVII, 241.
[2] *Trans.* I. M. M., bulletins 41, 42, 46. Feb. 13, March 12 and July 9, 1908; *Min. and Sci. Press*, XCVII, 563.

density of the medium during the settlement. Julian and Smart[1] state that, "Retardation of settlement is a function of the depth of the slimes pulp measured from the top of the still turbid portion of the liquid to the bottom of the vessel," and, "increase of suspended matter in the pulp causes a decrease in the rate of subsidence." The experimental apparatus used in the work consisted of a pyramid-shaped tank 4x5 ft. in cross section, connected at the bottom (through an 8-in. square opening) with a long flat inclined closed box in which a 10-in. belt was made to travel slowly, the power for the belt being supplied through a head roller. Where the apex of the pyramidal tank joined the inclined flat belt box, a trap door was placed just above the 8-in. opening. This trap could be removed from above as desired, thus establishing communication between the belt box and the tank. A charge of pulp from an agitator vat, 40 per cent. solids and 60 per cent. solution, was run into the tank, and the flat belt box was filled with water to the same level as the pulp in the tank. The solids were kept in suspension by air agitation, and then the trap door removed, agitation stopped and the belt set in motion at the rate of 4 ft. per minute in a direction so as to discharge slimes settling upon it at the top of the flat belt box. The removal of the slimes from the belt was effected by a scraper. The accompanying table gives an idea of the nature of the slimes removed.

NICHOLS' EXPERIMENTS. Slime=40 per cent. solids, 60 per cent. liquid.

Time in minutes after settling commences.	Wet weight of slimes removed at end of interval.	Dry weight of slimes removed at end of interval.	% of moisture in the slimes removed.	% of the total slimes removed, expressed in dry weight.
15	127.75 lb.	99.00 lb.	22.5	.....
30	154.25	117.20	24.0	37.64
45	128.00	89.50	30.0	.....
60	87.25	60.25	31.0	67.34
75	62.25	42.00	32.5	.....
90	45.25	28.25	37.5	75.97

It will be noted from the table that in 90 minutes, 75.97 per cent. of the solids of the slime charge were removed. The moisture percentage in these slimes as calculated from the above data was 27.9. The remarkable "dryness" of the slimes is noteworthy, when it is considered that usually fully settled slimes, by ordinary methods, contain 50 per cent. moisture. The above product more nearly approaches that discharged by vacuum filters. In practice the method would be applied as follows:

(1) The method could be used as an intermittent process in which a charge of pulp from an agitator vat would be run directly to a separator as described, which should be of sufficient capacity to hold the whole of such charge in addition to a residual charge from a previous operation.

[1] "Cyanidation of Gold and Silver Ores," 1907.

The object of this provision is to allow of extracting on the belt only such proportion of the solid contents of the charge as could be effected with the economic minimum of moisture. The belt discharge from this separator would be delivered into a second similar separator with addition of weak solution or wash water and the liquid so added could be sprayed on the discharging belt so as to loosen and disintegrate the discharged slime, or a mixer might be added for this purpose. After withdrawal of the required proportion of solids, and clarification and decantation of the solution left in the separator, to the original level, it is ready for the second charge. A third separator in series may also be added.

(2) The method may be applied as a continuous process by the introduction of a suction and filter. In this, the suction to take off clear solution, and the belt are so correlated with regard to a given supply of pulp that while the belt removes the solids, the suction removes liquid in such proportion that with a quietly inflowing fresh pulp supply, properly regulated, the contents of the separator will always be at the same level, and of the same gravity. If the gravity of the pulp in the separator is thus kept constant, it will be seen from the above table that the percentage of moisture in the discharged slime will be constant, and since the percentage of moisture carried over in the discharged slime varies inversely with the percentage of solids in the separator charge, this can be initially so proportioned as to give a final slimes product with the minimum amount of moisture, viz., about 23 per cent. Practically the same course of procedure will be followed in the wash separator as outlined for method 1.

It will be noted that the method deals with thickened slimes only, and would serve to replace filters. Preliminary settling, and agitation of the thickened pulp to obtain solution, would be carried out as in present processes.

Vacuum Filters and Pressure Filters.—The vacuum filters received greatly increased application during 1908, particularly the Butters, Moore and Ridgeway filters.[1] The last has been introduced in Mexico at the Pastita plant of the Guanajuata Development Company and at the Rio Plata mill in Chihuahua. Ten of the Ridgeway machines are in use at the Great Boulder company's plant in Kalgoorlie, Australia, and it is being used in India in an experimental way. It is also being used in Korea. At Kalgoorlie, accumulated slimes on the waste dumps are to be treated with Butters-Cassel filters.

The Barry filter[2] is similar to the Moore type but has a special frame of corrugated sheet metal. No absorbent material is used inside the filter

[1] See previous volumes of *The Mineral Industry*.
[2] Alfred James, *Min. and Sci. Press*, XCVIII, 52.

cloths nor are distance pieces of wood employed on the faces of the frames to offer resistance to the stripping of the slimes cakes.

The Burt pressure filter[1] is in operation in several mills at El Oro, Mexico. At the El Oro mill,[2] five Burt filters treat 600 tons per diem, at a very low cost. The slimes are not washed at present, due, as stated, to insufficient filtering area.

An interesting piece of slimes apparatus is the Dorr slimes thickener.[3] It consists of a vertical shaft suspended from a frame work over the center of a tank, and reaching nearly to the bottom, where four radial arms are fastened, inclined slightly upward from the center. Pieces of angle iron are attached to the arms and so placed as to move the thickened settled material to the center of the tank, where it is discharged continuously through a pipe. The whole mechanical device may be readily raised out of the tank when desired, as during a shut down. The thin pulp flows into the tank at the center, just below the surface, causing no disturbance, and the clear solution overflows continuously at the periphery of the tank. The shaft with its scraping arms rotates very slowly. It employs the principles outlined by Mr. Nichols, but has been used in a practical way by Mr. Dorr for a number of years at the Mogul Mill, Pluma, S. D. It is used by a number of large mills. The following data from the Liberty Bell mine are of interest:[4] Speed of rotating arms, 4.8 rev. per hour; power, 0.14 h.p. when running, 0.20 h.p. when starting after eight hours shut down; feed, to center, 15.4 per cent. solids, 84.6 per cent. liquid; discharge at bottom, 30.0 per cent. solids, 70.0 per cent. liquid; decantation at top of tank, clear; screen test of feed: On 50 mesh, 4.2 per cent.; on 100 mesh, 11.5 per cent.; on 200 mesh, 21.0 per cent.; through 200 mesh, 63.3 per cent. Three tanks fitted with the thickener replaced five tanks settling intermittently in the ordinary way, and save the work of one man per shift.

The Hunt Continuous Sand Vacuum Filter.[5]—This machine has a stationary filter bed consisting of a horizontal annular chamber, the top of which constitutes the filtering surface. The filter is a form of sand filter. The walls of the structure, and filter bed, may be made of concrete, and these being raised a couple of inches above the top of the filter bed constitute a circular track upon which a carriage revolves. The filter bed is formed of slats of triangular section placed side by side, but not touching, a space of $\frac{1}{4}$ to $\frac{3}{8}$ in. being left between the bases of the slats. Carefully screened gravel as nearly uniform in size as possible and too large to pass through the opening between the bases of the slats is then placed

[1] See previous volumes of *The Mineral Industry*.
[2] D. L. H. Forbes, *Eng. and Min. Journ.*, LXXXVI, 460.
[3] *Mex. Min. Journ.*, VIII, No. 2, p. 18.
[4] Private communication.
[5] Bertram Hunt, *Min. and Sci. Press*, XCVII, 430.

in the lower part of the bed. Above this, finer gravel or coarse sand is spread in a layer about 1 in. deep to form a floor on which is spread clean sand of 8 to 12-mesh size, to the thickness of an inch or so above the tops of the triangular slats. This forms the permanent filter bed and is supported strongly enough to remain perfectly rigid on the application of a vacuum in the chamber beneath. This chamber is connected with a wet vacuum pump which withdraws the filtered solution and maintains the vacuum. The carriage which runs on the filter bed has an iron scraper in front, of the same width as the filter bed, and removes the layer of residues from the permanent filter bed. The residue is removed from the scraper by a screw conveyer or by sluicing and is carried to the hollow central pillar of the machine from which it is carried out by water or a conveyer.

Following close behind the scraper on the carriage is placed a distributor, similar to that used on a Frue vanner, which distributes the sandy portion of the pulp; behind this is a second distributor which spreads the slimy portion of the pulp over the layer of sand just laid down by the first distributor. On the central column of the machine are two concentric hoppers or funnels, which are connected by pipes with the two distributors and receive the pulp. Before reaching the filter machine the pulp is roughly classified so that a portion of the sand is separated in a clean condition and this clean sand is fed upon the filter bed by the first distributor, the rest of the pulp going to the second distributor. In this way the slimy portion of the pulp is always distributed over a layer of clean fine sand. As the carriage moves continuously the deposited layer of sand and slime is also continuously removed by the scraper, while fresh material is deposited by the distributors as described above. Pipes to spray wash solution or wash water follow the carriage at suitable intervals so that the residue can be thoroughly washed. The machine by special devices may be arranged to wash with strong and weak solution, but where this is necessary two machines may be arranged to work in series on the same pulp, one with strong and the other with weak solution. The machine has an extreme diameter of 15 ft., the annulus being 3 ft. wide, giving a filter surface of 113 sq.ft. On the supposition that the layer of residue contains 50 per cent. moisture and weighs 109 lb. per cu.ft. and that it is 0.25 in. thick, it will amount to 2.26 cu.ft. and will weigh dry 123 lb. At the speed of 1 r.p.m. of the carriage, this represents a capacity of 80 tons per 24 hours. The power required is 1 h.p. to drive the machine and 4 h.p. for maintaining vacuum. When it is desired to clean the filter bed, the scraper which is adjustable is lowered, to remove the lower layer of sand, which is then renewed. It will be noted that the filter treats fine sands as well as

slimes, and seems well adapted to modern conditions where fine grinding is practiced. It would seem as if this comparatively simple and ingenious machine holds out more promise than many of those now on the market. A trial machine is in successful operation in San Francisco.

Chemistry of the Cyanidation of Silver Ores.[1]

The chief silver minerals occurring in Mexican ores are: 1, native silver; 2, argentite (Ag_2S); 3, proustite ($3Ag_2S$, As_2S_3); 4, pyrargerite ($3Ag_2S$, Sb_2S_3); 5, stephanite ($5Ag_2S$, Sb_2S_3); 6, cerargyrite (AgCl); also the bromide, the chloro-bromide, and argentiferous tetrahedrite containing silver as argentite; and galena and blende containing silver as argentite.

Of these the native silver is very slowly soluble in cyanide solution, so that practically no extraction can be obtained from it. Argentite is readily soluble, as is also the chloride, the bromide and the chlorobromide. Proustite, pyrargerite and stephanite are sparingly soluble in cyanide solution, but are more readily soluble in a KCN solution containing mercury salts. The extraction of silver by cyanide from argentiferous galena, tetrahedrite, and blende is a question yet to be solved. The interfering constituents in silver ores which give trouble in cyanidation are chiefly the oxide and carbonate of copper, just as in the case of gold ores. With silver ores preliminary treatment with sulphuric acid has been successfully applied, and with gold ores cuprous ammonium cyanide has been used[2] to remove copper. The one other important interfering base metal salt—ferrous iron—when existing in the form not removable by preliminary treatment, is readily overcome in mill solutions by adding mecurous chloride. Manganese salts, when occurring in sufficient quantities to cause trouble, can be precipitated during the process by adding to the solution an excess of calcium hydrate, provided they do not exceed 3 to 4 per cent. Mn. Soluble sulphides formed during treatment are neutralized by adding lead acetate. It is essential to use comparatively strong solutions and large volume of solution.

Silver Sulphide.[3]—This is dissolved according to the following reactions:

(1) $Ag_2S+4KCN=2KAg(CN)_2+K_2S$. This reaction is reversible, dependent on concentration and temperature and to keep silver in solution as $KAg(CN)_2$ and prevent its reprecipitation as Ag_2S, it is necessary in N/10 solutions (according to Bethelot) to have practically 100 KCN molecules per 1 molecule of K_2S.

[1] Francis J. Hobson, *Min. and Sci. Press*, XCVII, 159 and 182.
[2] See previous volumes of *The Mineral Industry*.
[3] F. J. Hobson, loc. cit., and W. J. Sharwood, *Min. and Sci. Press*, XCVII, 418

(2) $96KCN+Ag_2S = 2KAg(CN)_2+92KCN+K_2S$. Practically 30 parts by weight of KCN are required for every part by weight of Ag_2S.

In the presence of oxygen in the solution the following further reaction takes place, but rather slowly. (3) $K_2S+KCN+O+H_2O=KCNS+2KOH$.

If equations 1 and 3 are combined, there results: (4) $Ag_2S+5KCN+O+H_2O=2KAg(CN)_2+KCNS+2KOH$. Of course, the relation of KCN to Ag_2S as above stated must still be observed, and as the second part of the reaction, equation 3, proceeds rather slowly some K_2S will be found in the solution.

W. A. Caldecott[1] gives also the following secondary reactions which remove soluble sulphides from the solution:

(*a*) $2K_2S+2O_2+H_2O=K_2S_2O_3+2KOH$,
(*b*) $K_2S_2O_3+2KOH+2O_2=2K_2SO_4+H_2O$.

As this proceeds rapidly and does not destroy cyanide it shows the desirability of ample aeration to remove sulphides.

In the cyanidation of silver ores in order to prevent the accumulation of K_2S and to prevent the reversal of the reaction with the consequent reprecipitation of silver as sulphides, as the solution of the silver sulphide in the ore proceeds, mercury salts such as $HgCl_2$ or soluble lead salts are added; lead acetate, lead nitrate and even lead oxide may be used.[2] The following reaction then takes place, there being caustic alkalies present in solution as above:

(5) $Pb(C_2H_3O_2)+4KOH = K_2PbO_2 + 2K(C_2H_3O_2)+2H_2O$. That is, alkaline plumbites are first formed, which then react as in 6.

(6) $K_2PbO_2+K_2S+2H_2O=PbS+4KOH$. And further, according to Caldecott, as in 7.

(7) $PbS+KCN+O=PbO+KCNS$. But how far this last reaction proceeds is questionable. It would show that the removal of sulphides by lead salts is a cyclic reaction and a given amount of lead salt will destroy an indefinite amount of soluble sulphide, but with the consumption of cyanide.

Native silver is but sparingly soluble in cyanide solution according to Hobson;[3] what does dissolve follows Elsner's reaction, 8.

(8) $2AG+4KCN+O+H_2O=2KAg(CN)_2+2KOH$.

Silver chloride and bromide are dissolved according to the following reactions:

(9) $AgCl+2KCN=KAg(CN)_2+KCl$.

(10) $AgBr+2KCN=KAg(CN)_2+KBr$.

[1] *Journ. Chem. Met. and Min. Soc. of S. A.*, VIII, 267.
[2] C. M. Eye, *Min. and Sci. Press*, XCVIII, 82.
[3] *Ibid.*, XCVII, 420.

The presence of HCN in cyanide solution produced by the action of acid and acid sulphates in the ores on portion of KCN in the solution after the destruction of the protective alkali, which condition may arise when insufficient protective alkali is present in solution, causes a decided decrease in the extraction of the silver, the same as is the case with gold, in most cases.

There was considerable discussion during 1908 as to the effect of the addition of mercury salts to cyanide solutions in the treatment of silver ores. Hobson[1] states that when Hg_2Cl_2 (calomel) is added to cyanide solution it forms $KHg(CN)_2$—potassium mercurous cyanide—and that this has a powerful solvent action on the minerals—pyrargyrite, proustite, and stephanite—in the presence of free alkaline hydrates. He states that the potassium mercurous cyanide is an unstable salt, and in order to preserve it it is necessary to have present reducing agents in the solution such as potassium ferrocyanide, etc. Certain of these facts are disputed by Bertram Hunt,[2] W. J. Sharwood, and W. A. Caldecott. It is shown that mercurous cyanide does not exist and that only mercuric cyanide, $K_2Hg(CN)_4$, is known. Sharwood[3] states that the use of potassium mercuric cyanide as a solvent for gold has been known many years, having been mentioned by Skey as early as 1876. He sums up the effect of mercury salts in the persence of cyanide on gold and silver as follows: "The effect of the salt is probably partly due to replacement of one atom of mercury in the $K_2Hg(CN)_4$ by two atoms of gold or silver, which replacement is independent of the presence of oxygen and partly, if air is accessible to the action of the free KCN, which is accelerated at first by the influence of the mercury gold couple thus formed.

"When Ag_2S is similarly dissolved, there is an exchange of one atom of Hg for two atoms of Ag, the HgS formed being insoluble and therefore without influence in retarding the further dissolving of portions of silver. There is an obvious economy in the mercury lost, when all possible extraction is effected by the aid of simple cyanide with air oxidation, before resorting to the use of mercuric salts. Thus, with $K_2Hg(CN)_4$ and an equal weight of KCN the action on gold and silver at first is very rapid, then the action becomes slower as a coating of mercury is deposited and amalgam forms."

Cyanide Costs.

Below follow some costs of cyanidation in different parts of the world.

[1] *Loc. cit.*
[2] *Min. and Sci. Press*, XCVII, 286.
[3] *Ibid.*, XCVII, 420.

COST OF CYANIDE MILLING AT MILLS OF GUANAJUATO DEVELOPMENT COMPANY'S MILLS. (a)

Name of Mill.	Period 1908.	Wet tons milled per mo.	Crushing.			Concentration.	Classifying.	Cyanide treatment.	
			Coarse.	Fine.	Regrinding.			Sand.	Slime.
Peregrina	Average of 4 mos.	12,356	$0.079	$0.395	$0.185	$0.061	$0.011	$0.346	$0.253
	Minimum mo. tonnage	11,324							
	Maximum " "	13,593							
Pinguico	Average of 8 mos.	6,188	0.075	0.305	0.235	0.080	0.020	0.460	0.405
	Minimum	5,820							
	Maximum	6,168							
San Prospero	Average of 7 mos.	4,014	0.055	0.340	0.100	0.050	0.025	0.390	0.360
	Minimum	5,820							
	Maximum	6,168							
Nayal	Average of 6 mos.	1,347	0.115	0.600	....	0.135	0.205	0.595	0.430
	Minimum	1,168							
	Maximum	1,477							

Name of Mill.	Period 1908.	Filtering and Precipitation.	Pumping Solutions.	Water Supply and Surface Expense.	All Costs.				
					Labor & Bosses.	Supplies.	Power.	Miscellaneous.	Grand Total.
Peregrina	Average of 4 mos.	$0.189	$0.067	$0.042	$0.453	$0.748	$0.415	$0.003	$1.619
	Minimum mo. tonnage				0.485	0.910	0.425	0.005	1.825
	Maximum " "				0.440	0.665	0.380	....	1.485
Pinguico	Average of 8 mos.	0.415	0.050	0.120	0.720	1.090	0.355		2.165
	Minimum				0.825	1.140	0.365		2.330
	Maximum				0.675	0.975	0.325		1.975
San Prospero	Average of 7 mos.	0.310	0.065	0.055	0.470	0.875	0.395	0.010	1.750
	Minimum				0.510	1.100	0.470		2.080
	Maximum				0.465	0.860	0.335		1.660
Nayal	Average of 6 mos.	0.290	0.045	0.080	1.020	1.045	0.420	0.010	2.495
	Minimum				1.065	1.350	0.535		2.950
	Maximum				0.950	0.915	0.285		2.150

(a) C. T. Rice, *Eng. and Min. Journ.*, LXXXVI, 1001.

Cost of Precipitation and Refining at the Liberty Bell Mine, Telluride, Colo.[1]—March, 1908: Ore milled, 10,548 tons; solution through zinc boxes, 24,510 tons; precipitate recovered, washed and dried, 1275.5 lb.; bullion from the same, 16,016.8 oz.; metal in precipitate, 86.1 per cent.; bullion fineness, gold and silver, 950. Precipitation: Zinc, 4700 lb. at $0.0897 per lb., $421.60; cutting the same, $65; labor, $81.55; power and supplies, $26.65; total, $594.80. Clean-up and filter pressing: Labor of cleaning and repacking zinc boxes, etc., 1200 cu.ft. zinc, $43.50. Refining: Acid treatment and washing, 1050 lb. H_2SO_4 at $0.046 per lb., $48.30; labor, $55.30; power and repairs, $8.45, total, $112.05. Drying and melting: Coal, 750 lb. at $6 per ton, $2.25; coke, 2500 lb. at $17 per ton, $21.25; flux, $32.75; crucibles, $36; labor, $29; repairs and sundries, $3.05; total, $124.30. Grand total cost of operation, $874.65. Cost

[1] W. A. Moulton, *Min. and Sci. Press*, XCVI, 803.

per ton of solution: For precipitation, $0.0243; for clean-up, etc., $0.0018; for refining, $0.0096; total, $0.0357. Cost of refining a troy ounce bullion, $0.0148.

PRECIPITATION, MELTING, REFINING.(a)

Name of Mill.	Liberty Bell.	Desert.	Pinguico.
Ore milled, tons of 2000 lb.	10,548	13,830	6,927
Solution through boxes, tons	24,510	81,000	51,150
Precipitate recovered, lb.	1,275	27,947	11,026.4
Bullion from precipitate, oz.	16,016.8	291,412	123,359.5
Metal in precipitate, per cent	86.1	71.5	76.3
Bullion, fineness, Ag. and Au	980	972.6	925.55
Precipitation—			
Zinc and supplies	$594.80	$2899.71	$1180.75
Clean-up and filter-pressing—			
Labor on boxes	43.50	519.45	77.56
Refining—			
Acid and labor	112.05	.....	165.37
Drying and melting	124.30	1928.32	606.12
Totals	$874.63	$5347.48	$2029.80
Precipitation, per ton of solution	$0.0243	$0.0358	$0.0230
Clean-up and filter-pressing, per ton of solution	0.0018	0.0064	0.0015
Refining, per ton of solution	0.0096	0.0238	0.0150
Totals, per ton of solution	$0.0357	$0.0660	$0.0395
Refining cost, per oz., bullion	$0.0148	$0.0066	$0.00625
Total cost of precipitation, melting, etc., per oz. bullion	$0.0545	$0.0183	$0.0164

(a) *Min. and Sci. Press.* C. E. Rhodes, XCVII, p. 523; Parsons, XCVII, p. 46; W. A. Moulton, XCVI, p. 803.

Crown Deep, Ltd., Transvaal.—200-stamp mill, crushing 6.183 tons per diem for 322½ days, milling 399,130 tons of a value of 9.729 dwt. gold per ton. Recovery, 94.836 per cent. Tailings assayed 55c. per ton. Recovered by amalgamation, 65.597 per cent.; by sand cyanidation, 22.551 per cent.; by slime cyanidation, 6.688 per cent.; extraction from sands, 86.982 per cent.; from slimes, 85.313 per cent. The ore treatment expenses were as follows: Ore sorting, 4.6c.; crushing, 4.6c.; haulage, 1.9c.; stamp milling, 35.4c.; tube milling, 13.1c.; conveyance, classification of pulp and filling tanks, 5c.; cyaniding, 31.5c.; disposal of tailings, 13.2c.; grand total, $1.106. Total receipts were $9.29 per ton; total costs mining, milling, etc., $5.42; working profit, $3.87 per ton.[1]

Consolidated Mercur Gold Mines Company, Mercur, Utah.—Dry crushing plant. Average value of ore treated, $3.77; average tailings value, 92c. per ton; cost of milling, $1.26 per ton; cost of mining, $1.65; total cost, $2.91. Consumption of cyanide, 0.86 lb. per ton; lime, 6.85 lb. per ton.[2]

[1] Official report.
[2] *Eng. and Min. Journ.*, LXXXVI, 386.

Slime Treatment at the Homestake Mine, Lead, S. D.—The cost by Merrill filter presses, for March, 1908, 49,946 tons being treated, is given in the accompanying table:[1]

SLIMES TREATMENT AT HOMESTAKE MILL (a).

Item.	Labor.	Electric Power & Lighting.	Chemicals.		Other Supplies.	Total Cost.
			Item.	Cost.		
Thickening pulp	$0.0037				$0.00006	$0.00379
Transportation					0.00030	0.00030
Neutralization	0.00625	$0.00116	Lime 4.476 lb.	$0.02236	0.00006	0.02985
Filling and discharging presses	0.0104	0.00347			0.00239	0.01626
Dissolving and washing in presses	0.02468	0.01502	Cyanide 0.31 lb.	0.0620	0.00014	
			HCl	0.02324	0.010	0.13508
Precipitation	(triangular presses)				0.00057	
Dust	0.00281		Zinc 0.127 lb.	0.00762	0.00026	0.01126
Heating	0.00218				0.00537	0.00755
Assay Office	0.00348				0.00518	0.00866
Superintendence	0.00911				0.00043	0.00954
Miscellaneous	0.01272				0.00117	0.01389
Fire Protection					0.00054	0.00054
Refining Bullion					0.00864	0.00864
Exp. and Mint Charges						
Totals	$0.07533	$0.01965		$0.11524	$0.03511	$0.24533

(a) Cost of supplies:—HCl, $4.20 per carboy; cyanide, 20c. per lb.; lime, 0.5c. per lb.; zinc, 6c. per lb.; labor, $3.00 per 8 hour shift; power, $7.50 per mechanical horse power per month; filter cloth, 1c. per ton of slime treated.

Kalgoorlie, W. A.—The Associated Northern mine has been retreating waste slimes, 2700 tons per month, by mixing, agitation and filter pressing at a cost of 60c. per ton. The Lake View Consols mine has been retreating 9000 tons per month by the Cassel basket vacuum filter at a cost of 62c. per ton, the extraction in both cases being the same.[2]

Miscellaneous.

Amalgamation in Cyanide Solution.—With the constant increase in the application of crushing in dilute cyanide solution this question was discussed considerably during 1908. At the Liberty Bell mine,[3] at Telluride, Colo., the ore is crushed in a solution containing 1.5 lb. KCN per ton. The plate below each battery is 4 ft. 11 in. wide and 16 ft. long. Some of the plates have four drops in this length and others three and two. The plates are kept soft by ample mercury, as it is noted that the cyanide in solution tends to harden the amalgam in spots, making the amalgamation less efficient. The pulp from the plate is classified and the sands reground in tube mills, and the whole product as slimes passed over an additional 8 ft. of amalgamated plates. No particular difficulty is experienced in amalgamation.

[1] For sand treatment costs, see *Mineral Industry*, XVI.
[2] *Min. and Sci. Press*, XCVII, 766.
[3] T. T. Read, *Min. and Sci. Press*, XCVII, 670.

Strong cyanide solution will harden amalgam, the quicksilver coming out in globules and running down the plate.[1] Amalgamation in cyanide solutions of such strengths as is usual, presents no difficulty, except that plates rarely last more than six to nine months with solution of 1 to 1.5 lb. KCN per ton. A field for investigation is offered here, for with plates (silvered copper) costing approximately $2 per sq.ft. the item of renewal is a considerable one. When amalgamating in cyanide solution the plates will be eaten through in spots, and as it may not be desirable to renew the plates at once, the plate tables should be built in such a way as to prevent the mercury from working through. When drops are used between plates, the edges of plates should not project beyond the backing, as they are gradually eaten down to dangerous knife edges.

Sometimes the cyanide solution,[2] if strong enough, will soften the mercury and amalgam and cause it to flow, effecting a hardening of the plate in spots. In this way it has the same effect as if the amalgamation were carried on in hot water. Gold in the form of amalgam is more readily attacked by KCN than free gold; there are, therefore, when amalgamating in KCN solution, two opposing actions continually going on, viz., (1) the mercury taking free gold from the ore; and (2), the cyanide solution dissolving gold from the amalgam. But as the action of cyanide is slow and that of amalgamation fast, the amalgam increases and accumulates on the plates. It is also very probable that the cyanide solution prepares some of the gold for amalgamation which might escape the plates when crushing in water. This seems to be borne out by the fact that when amalgamating in solution, amalgam will form the entire length of the plate. At the Liberty Bell mill, above mentioned, it is almost as heavy at the end of the 16 ft. of plate as at the head.

Precipitation.—Zinc dust precipitation is coming more and more into use in plants that have large amounts of solution to precipitate. It is at present being installed in a number of plants in Mexico treating silver ores. When it is considered that the consumption of zinc is appreciably less with zinc dust than with zinc shavings, and that in treating silver ores comparatively large amounts of solution must be handled, zinc dust precipitation becomes a factor. At the Desert mill, Nevada, treating silver ores, the zinc consumption is 1.71 lb. per ton at a cost of 18c., or 0.8c. per oz. bullion recovered. It has been pointed out in a previous volume of THE MINERAL INDUSTRY[3] that the zinc consumption is a function more of the amount of solution to be precipitated than of anything else. It

[1] H. W. McFarren, *ibid.*, XCVII, 815.
[2] J. H. Haynes, *West. Chem. and Met.*, IV, 233.
[3] Vol. XIV, 283.

will usually amount to from 0.20 to 0.4 lb. of zinc per ton of solution precipitated when shavings are used, irrespective of whether gold or silver ores are treated. The figures for zinc dust are appreciably less than this as will be seen from the results of the Homestake mills given in the accompanying table:

ZINC DUST PRECIPITATION AS COMPARED TO ZINC SHAVINGS AT THE HOMESTAKE MILLS, LEAD, S. D. (a)

Monthly Averages of Results and Costs.

Year.	1906.	1908.
Months averaged	12	9
How precipitated	Shavings	Dust
Cubic contents, boxes	142 cu. ft.	.....
Filtering surface, press	.....	230 sq. ft.
Maximum flow per hour, tons	12	14
Minimum strength of solution precipitated	0.03%	0.015%
Average strength	0.03%	0.0242%
If below minimum strength, strengthened by addition of	Barren works solution, of strength 0.07%.	The lowest strength encountered, .015% is successfully precipitated.
Tons precipitated	5909	5585
Dollars recovered, total	$1513.00	$1513.00
Dollars recovered, per ton solution	0.2560	0.3060
Average barren assay	0.0214	0.0178
Per cent. precipitated	91.7%	94.4%
Pounds of precipitant used		
Total	1019	718
per ton solution	0.1724	0.1285
per dollar recovered	0.6735	0.4192
Value of precipitate per pound	$2.35	$4.03
Cost of precipitant	$129.75	$51.15
" " labor	97.50	19.75
" " miscellaneous supplies	16.03	6.03
All costs, total	$243.28	$76.93
" " per ton solution	.0412	.0138
" " per dollar recovered	.1608	.0449
Pounds cyanide, in solution, wasted after precipitation	3545	2703
Value of same	$709.00	$540.60

(a) Allan J. Clark, *Journ. Chem. Met. and Min. Soc.*, S. A., IX, 222.

SUMMARY OF HOMESTAKE RESULTS.

	Shavings.	Dust.	Saving by Dust.
All costs, labor and supplies	$243.28	$76.93	$166.35
Gold discharged in barren solution	126.45	99.41	27.04
Value of cyanide in wasted barren solution	709.00	540.60	168.40
Totals	$1078.73	$716.94	$361.79

The equipment provided at Sand Plant No. 1 for which the above figures are valid, consists of two 70-ton tanks and a Merrill triangular precipitation press of 16 frames, placed about 15 ft. lower than the tanks, the solution flowing to the press by gravity. An ingenious device is used to feed the zinc dust in exact proportion to the amount of solution,[1] which

[1] R. L. Herrick, *Mines and Minerals*, XXVIII, 432.

is very essential. The above figures apply only to weak solutions, and those as low as 0.015 per cent. KCN are successfully precipitated by zinc dust, with from 0.3 to 0.5 lb. CaO protective alkalinity per ton. It has always been considered practically impossible to precipitate satisfactorily solutions below about 0.07 per cent. KCN by means of zinc dust, but the above solutions are very much lower than this.

Zinc Dust Precipitation at the Cerro Prieto Plant.—At the works of the Black Mountain Mining Company, near Magdalena, Sonora, Mexico,[1] the ore is crushed in weak KCN solution in a 120-stamp mill. Coarse sands are reground in tube mills, and the whole product is classified into sands and slimes, the sands treated by percolation and the slimes by decantation and filtering in vacuum filters. The gold solution carries generally 2 to 3 dwt. in gold and four times as much silver per ton. They contain ordinarily from 0.07 to 0.1 per cent. KCN and 0.2 per cent. alkalinity (protective). A sand filter removes all suspended slimes from solution before precipitation (this is essential in zinc dust precipitation). The solution precipitation tank is also provided with a sand filter, to insure absolute freedom from suspended slimes. The gold solution is drawn off from the bottom of the precipitating tank through a 6-in. pipe, connected to the precipitating press. Zinc dust in the form of an emulsion is added at the point where the solution leaves the tank. The emulsion is prepared as follows: A small quantity of solution is lifted by compressed air through a 1-in. pipe and discharged into a cone at the top of the tank. Zinc dust is fed into this cone at regular intervals and thoroughly mixed with the solution by compressed air through a 0.75-in. pipe reaching nearly to the bottom of the cone. The emulsion overflows into a 3-in. pipe leading into the 6-in. outlet pipe and is added to the overflowing gold solution and becomes thoroughly mixed with it in flowing down to the zinc press. This apparatus maintains very satisfactorily the essential regularity of the zinc dust feed. The zinc press is of the Dehne type, 35 in. square on the outside of the frames, having 24 leaves, with 4-in. frames intervening. The total filtering surface is 367 sq.ft. Cotton sheeting, 72 in. wide, is used for filter cloths. Two cloths are used over the leaves. Each time the press is cleaned new cloths are put on next to the leaves, the inner cloth put on over them, and the outer cloth burned. The press stands 31 ft. below the bottom of the precipitating tank, affording a gravity pressure for filling the press of from 31 to 47 ft., or 13.4 to 20.4 lb. per sq.in. No pump is used in connection with the press.

The resistance or back pressure of the press varies from 0 to 14 lb. as it begins to fill up to the time it is ready to be discharged. After cleaning the press and commencing a new precipitation, an excess of zinc dust is

[1] Robert Linton, *Journ. Chem. Met. and Min. Soc.*, S. A., IX, 74.

fed into the feed cone, and the mixture allowed to flow into the press slowly in order to form a thin film of zinc dust on every filter cloth. The high rate of zinc dust feed is kept up for six hours, the quantity of solution fed being meanwhile gradually increased. The rate of feeding zinc dust is then cut down, and gradually lowered until the press is cleaned again. Before starting the press the gold solution is always titrated for KCN, and when necessary enough fresh cyanide added to insure the immediate setting up of precipitation. After precipitation is once started, comparatively weak solutions (as low as 0.07 per cent. KCN) are precipitated without difficulty. The precipitation is very rapid and complete. As much as 1200 tons of solution has been passed through the press in 24 hours, with an average value left in the effluent solution of less than 1 grain of gold and a mere trace of silver per ton. The rate of flow corresponds to 3.2 tons of solution per square foot of filter surface per 24 hours. Lead acetate is added in suitable proportions to the zinc dust to aid precipitation. The silver content of the solution doubtless aids the rapidity of precipitation. The average flow through the press is 900 tons per 24 hours. The consumption of zinc dust is 0.4 lb. per oz. of gold and silver precipitated.

Success in zinc dust precipitation requires careful supervision and a thorough working out of questions of feed of zinc to the solution and of zinc and solution to the press. The aim is to obtain a granular product, which when it forms a coating on the cloths or settles to the bottom of the frames, shall offer the least obstruction to the flow of solution through it. If the product is very fine, or contains slime carried in from unclarified solutions, the coating on the cloths soon becomes dense and impervious to the solution flowing through the press, and the capacity of the press is much diminished. The refining of the zinc dust precipitate involves more labor and expense than the refining of the zinc shavings product. It is practically essential to refine zinc dust precipitates by smelting and cupellation as described for the Homestake company in Vol. XVI of THE MINERAL INDUSTRY. The additional cost of refining has proved a serious factor in many cases. On the other hand, there is the saving of all the trouble and expense connected with the zinc boxes, and a saving in zinc, and if the refining is done efficiently, the extra cost of refining may be practically balanced in the saving made in the precipitation and in the ability to work up the by-products of the clean-up. There is further a very decided advantage, in that, as the press is locked and secure, there is an increased security against theft.

Smelting of Cyanide Precipitates.—As practiced at certain mills on the Rand,[1] the precipitates from the zinc presses are not acid-treated,

[1] A. Thomas, *Journ. Chem. Met. and Min. Soc.*, S. A., IX, 6.

but are put into white-washed trays, 48 in. long, 26 in. wide, 8 in. deep, made of ⅛-in. soft steel. The white-wash prolongs the life of the trays and prevents the precipitates when calcined from adhering to the bottom. These trays are placed on special rectangular fire boxes for calcining. The temperature during calcination is kept low at first and gradually raised as calcination proceeds. The operation usually takes three hours. The loss in weight is about 40 to 45 per cent.; most of it, 38 per cent., is due to moisture and the remainder probably to carbonaceous matter. The calcined precipitates are of a purple color. When calcined and cold they are taken to an adjoining room where they are weighed and fluxed. In order to avoid dusting, there should be no unnecessary handling. The fluxing is carried out as follows: The calcined precipitates, which are more or less caked, are weighed direct from the trays without any grinding. Plumbago crucibles, No. 100, with clay crucible liners are used. One hundred parts of precipitates are placed into the crucibles with 7.5 to 10 parts of MnO_2, filling the crucible to within 4.5 to 5 in. from the top, while the rest of the space is used for the fluxes, made up of crushed old liners from previous melts for the silica 25 parts, and fused borax from 20 to 25 parts. The flux is varied to suit the composition of the precipitate. The buttons of gold obtained from the fusion are usually of high grade and ready for casting into bars. Occasionally a thin skin of matte may appear on the buttons. When this is the case, these buttons in the final melting are put together with the richest buttons and the fineness of the bullion thus averaged up. When there is more matte, it is skimmed in the final melt of all the buttons and treated with the by-products. Matte generally occurs when the lining crucible cracks and the slag comes in contact with the reducing walls of the graphite crucible. Good and thorough calcination of the precipitates to oxidize the base metals is essential to produce a fine bullion. It cuts down the bulk of the precipitates, and does away with the necessity of adding large amounts of oxidizing agents, as MnO_2, for the oxidation of the base metals, the addition of which causes rather high slag losses, particularly in silver. The bullion is cast into bars of about 700 oz. weight.

Treatment of the By-products of the Refining.—Most of the mines employ the usual type of reverberatory pan furnace, as is used in the Tavenner process. The pan is made of 0.5-in. cast iron, is 9x6 ft. in size and 2 ft. deep. Size of grate in the fire box is 3.5x3 ft. Size of hearth in iron pan is 7.5x3.5 ft.; center of hearth to roof, 2.5 ft.; lowest point of roof to hearth, 1 ft. 7 in.; uptake flue of furnace 2.5x0.75 ft.; capacity of furnace for lead bullion, 4200 lb. The by-products consist of the slag from the smelting of the precipitates, ground-up crucibles and liners, and rich pan furnace slags from previous runs. In smelting this

material such other refuse is added, as may be at hand, viz., bullion furnace ashes, precipitation house sweepings, old cupels, bricks from furnace bottoms, calcining trays, old battery screens, assay slag, etc. In one instance a mixture as above was made into 14 charges of 900 lb. each containing 9 per cent. PbO, and 5.5 per cent. metallic iron for matte and reducing agent. In starting the melt a few bars of lead are placed in the hearth of the furnace to form a bath for the charge to rest on. Average value of slag from this melt was 0.2 oz. gold and 16.8 oz. Ag per ton. When the melt is finished the slag is skimmed from the charge and the lead bullion is drossed to eliminate some copper. The lead bullion is then cupelled in an English cupelling furnace. In the above run 19,143 oz. troy of lead bullion yielded 840 oz. of bullion, of the fineness, gold 477.5 and Ag, 471.5.

Cyanidation of Concentrates.—With the increased application of cyanidation to silver ores, in the course of which concentrates carrying a large part of the ore value are produced, the question of the cyanidation of concentrates is again very prominent and some success has been achieved. The essential points of treatment as outlined for arsenopyrite concentrates carrying gold by the Oriental Cons. Mining Company, at Chittabalbie, Korea, are as follows:[1]

1.—The concentrates must be treated fresh, and reground in tube mills with cyanide solution of not less than 0.4 per cent. KCN when new and up to 0.6 per cent. total cyanides after use for some time. Materials must be ground so that 90 per cent. will pass a 200-mesh screen. For fresh concentrates 2 to 3 lb. of lime per ton are sufficient.

2.—The concentrate charge is to be mechanically agitated and aerated for 15 to 20 hours.

3.—The cyanide solutions must not be permitted to become foul from too much use. A solution free from reducing agents (sulphides) should be used, i.e., the solution must be cleansed frequently. Thorough aeration will keep the reducing agents in the solution down. Solutions should be renewed every six months.

4.—The author recommends a speciál combined agitator and vacuum filter in the treatment of concentrates. The concentrates after agitation are washed with two washes of weak cyanide solution, 0.25 per cent. KCN and 0.1 per cent. KCN respectively, and then with water. Precipitation of solutions is carried out in the ordinary manner.

At the Grand Junction mine, Waihi, New Zealand,[2] concentrates consisting of pyrite, galena, and blende, amounting to 8 per cent. of the weight of the ore, and of a value of $53 per ton, already fine enough to

[1] A. E. Drucker, *Min. and Sci. Press*, XCVI, 458.
[2] F. C. Brown, *Min. and Sci. Press*, XCVII, 424.

pass a 200-mesh screen, are again reground in tube mills, with 25 per cent. of 10-mesh sands, so that the final product will pass through a silk cloth corresponding to what would be a 400-mesh screen. In order to grind to this fineness, it is necessary to add the sands above mentioned. The reground concentrates are then agitated in Pachuca tanks, of 40 tons capacity, 37 ft. high and 7.5 ft. in diameter, for 12 days with 0.4 per cent. KCN solution. They are then vacuum-filtered. The extraction by this method is 92.3 per cent. The ratio of Au to Ag in the original ore is 1: 6.

In Mexico, the problem of treating concentrates is being widely taken up, particularly at Guanajuato, where the Mexican Milling and Transportation Company has been experimenting with a small plant, at the San Prospero mill. The method of treatment is similar to that described for the Grand Junction mine, except the concentrates will be filter-pressed, and precipitation of gold and silver will be carried out by zinc dust in Merrill precipitation presses.

The Goldfield Consolidated Company, Nevada, is also working out a process of treating concentrates, details of which are not as yet available.

Crushing.—Direct crushing of ore in cyanide solution is receiving wider application every year. There are of course, certain conditions, e.g., high acidity of ore, etc., where the method is not practicable, but wherever it can be applied it seems to be the favorite method. The weight of stamps has again been increased on the Rand, but a notable conservatism in this direction is displayed in the United States and Mexico. On account of the inaccessibility of many Mexican mines, and also the high first cost for the installation of stamps, millmen in that country are looking to other devices for medium crushing; at the Hacienda de San Francisco near Pachuca, large roller Chilean mills are used in place of stamps with good success. The new mill of the Esperanza Company will crush sulphide ores in Huntington mills in place of stamps.[1] The oxidized ores will be crushed by stamps. For regrinding, tube mills are used almost exclusively and the notable feature of tube milling for 1908 was the employment of hard local ore in place of pebbles wherever this is possible. The saving due to this substitution is very considerable. An excellent paper on tube mill economics has been published by H. W. Fox.[2] The experimental regrinding plant of the Homestake company at Lead, S. D., was in operation during 1908, with the result that the tube mill shows certain economies over the pans, which will probably lead to the adoption of the first named machine.

Use of Lime.—W. J. Sharwood[3] has investigated the relative efficiency

[1] C. T. Rice, *Eng. and Min. Journ.*, LXXXVI, 760.
[2] *Mines and Minerals*, XXVIII, 537.
[3] *Journ. Chem. Met. and Min. Soc.*, S. A., VIII, 203.

of coarse and fine lime in cyanidation of sands by percolation. He shows that under certain conditions if slacked lime is used, or unslacked lime, of particles finer than 100 mesh, that this yields about 90 per cent. of its available lime in five days leaching. But the yield is at a rate not sufficiently uniform. The conditions at the Homestake sand plant No. 1 are the use of 3.5 lb. of lime per ton sands, through which solution percolates at the rate of one ton of solution in five days per ton of sands. The sands leached contain pyrite and pyrrhotite (5 to 10 per cent.), and the acidity is generated progressively, viz., the acidity is not all present at the beginning of the treatment, but forms continuously during the treatment. If now the lime used be slacked, or be too fine if unslacked, too much goes into solution during the early stages of the leaching with not enough left for the latter stages.

The best size for the lime is found to be when the greater number of particles are between 40 and 60 mesh. Particles larger than 20-mesh are not dissolved completely during the time of treatment and are discharged with the tailings, hence represent a waste in lime. The manner of adding lime at the plant is to crush the burnt lime in a jaw crusher set to a 1.5-in. opening. Two hours supply is weighed out and this is gradually delivered by a Challenge feeder to a single stamp battery with a small stream of water. The screen of the battery discharges into a launder which joins the launder carrying the classified sands to the leaching tanks. A further addition of lime is made during the progress of the leaching, by spreading 200 lb. over the surface of a 600-ton charge. If, however, the alkalinity of the earlier effluent solution is unduly high, in comparison with the percentage of cyanide (viz., if the protective alkali is excessive) this top lime is omitted. It is found at this plant that if the "protective alkali" is higher than 0.3 lb. CaO per ton of solution the extraction of gold is impaired.

The alkalinity of solutions is indicated by what are termed alkalinity numbers. The alkalinity number corresponds to the number of cubic centimeters of decinormal acid required to neutralize 100 c.c. of effluent solution. The alkalinity refers to total alkalinity, and not to the alkalinity of any particular compound. It can be translated into terms of percentage of any given alkaline substance by multiplying by the equivalent weight of the compound. Thus:

1 unit number=1 c.c. N/10 acid=	.0028% CaO	=0.056 lb.	CaO per ton.
	.0056% KOH	=0.112 "	KOH " "
	.0040% NaOH	=0.080 "	NaOH " "
	.0065% KCN	=0.13 "	KCN " "

A saturated solution of lime in water has an alkalinity number of 45. The ordinary cyanide solutions have about the same solvent powers for CaO that water has.

In a cyanide solution of known percentage of KCN and alkalinity estimated by the above standard, the protective alkali in pounds per ton can be calculated by the formula: $0.056a—8.6K$=pounds of CaO per ton, in which a=the alkalinity by the above standard, and K=the percentage of KCN. Synthetic tests have shown that this method, while not by any means strictly accurate in the presence of double zinc cyanide, is liable to no greater errors than the methods based on titration after precipitation with silver nitrate.

In the cyanidation of material containing $FeSO_4$ it was found that the addition of NaOH to solutions for protective alkali caused a very marked falling off in extraction.[1] This is explained by the formation of $Fe(OH)_2$, which is a very powerful reducing agent, robbing the solution almost immediately of its oxygen and thus preventing the solution of the gold. The only remedy for this condition seems to be to remove the $FeSO_4$ by means of water washes preliminary to cyanide treatment, and cyanide with solutions of as little protective alkalinity as possible, even at the expense of increased cyanide consumption.

"Passivity of Gold" in Cyanide Solutions.—In KCN solutions gold dissolves at the anode as a univalent ion. Passivity of gold (non-solution) develops in commercial KCN solution at a small concentration, a brown color due to paracyanogen being noticed in the solution; but in pure KCN a much higher concentration is necessary to induce passivity.[2] The presence of NaCN in the commercial KCN is responsible for this difference and the cause of passivity appears to be the formation of an invisible protective coating of sodium auro-cyanide on the anode, this salt being much less soluble than that of potassium. This explains the superiority of KCN over NaCN as a gold solvent.[3] This explanation of passivity may be applied to all known instances of this phenomenon. Thus, when in the neighborhood of the anode the solubility product of the ions of the protecting substance is reached (hastened in this case by the presence of more sodium ions) the protective coating is formed, rendering the anode passive.

Deterrents to the Dissolution of Free Gold in Cyanide Solutions.—It was found that oil and grease in the ore will act to prevent the solution of gold by KCN, and also that the precipitation of $CaCO_3$ in the sands, in highly alkaline solutions, will have a similar effect.[4]

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[1] B. L. Gardiner, *Journ. Chamber of Mines*, West Australia, Oct., 1907, p. 915.
[2] A. Coehn and C. L. Jacobson, *Zeit. der Anorgan. Chemie*. V. 55-321-55.
[3] Know that at one large mill the relative efficiencies of KCN and NaCN were fully tried out on a large scale and that KCN (ordinary commercial salt) proved to be superior, but only so to a very slight extent. C. H. F.
[4] Duncan Simpson, Bulletins Nos. 41 and 42, Inst. Min. and Met., March 12 and Feb. 13, 1908.

Progress in Gold Milling in 1908.

By Robert H. Richards and Charles E. Locke.

Mill Design, Construction and Equipment.

Screen Frame.[1]—At the Ida H. Gold Mining Company, West Australia, instead of tacking the stamp screen to the inside of a wooden frame, each panel of the frame has a rabbet ½ in. deep and ¾ in. wide all around its edge and each screen panel fits into this recess and is held in place by an iron frame secured by buttons. This arrangement facilitates the changing of screens and avoids the necessity of throwing away the whole screen when one panel is broken.

Repairing Wooden Mortar Blocks.[2]—At the Treasury mine the wooden mortar blocks decay around the outside where they extend above the ground. A complete description is given of the repair of such a block, the decayed portion being cut away and new wood fitted in its place. Three days and 21 hours elapsed from the time the stamps were hung up until they were dropping again. The total cost including labor was $202.50.

Reinforced Concrete Foundations for Stamp Batteries.[3]—This is a good example of the extension of the use of concrete for mortar blocks. The old wooden mortar blocks for the stamps at Redjang, Lebong, Sumatra, had become decayed and were replaced at the top by reinforced concrete. The article contains a full description of the operation, together with detailed costs.

Feeding Water to Stamps.[4]—Water fed in beside the stamp stems strikes on top of the bosses and falls as rain. Water fed with the ore shoots over the dies against the bottom of the screen frames and makes a back wash. Water let in horizontally at a level with the top of the dies forces uncrushed rock to bank up against the screens. Water let in through five pipes two at each side of the feed chute and one under the feed chute so as to strike the back plate vertically and strike the back end of the dies at an angle of about 50 deg., has sufficient force to wash off the crushed rock without moving the coarse particles. The last scheme is best as shown by the accompanying table. Hight of discharge is best kept constant by varying the hight of the chuck block.

The table shows the capacity per stamp per 24 hours. The screened rock was through 1½-in. screen except in trial III, where it was all fines. In trial I the stamps weighed 1150 lb. and dropped 7 in. 105 times per

[1] G. W. Williams, *Journ. Chem. Met. and Min. Soc.*, South Africa, VIII (1908), p. 198.
[2] A. Richardson, ibid., IX (1908), p. 24.
[3] S. J. Truscott and J. P. Fuller. *Trans.*, I. M. M. (1908); *Canadian Min. Journ.*, Vol. XXIX (1908), p. 586; *Min. Journ.*, Vol. LXXXIV (1908), p. 511.
[4] H. T. Pitt, *Journ. Chem. Met. and Min. Society*, South Africa, Vol. VIII (1908), p. 373; *Eng. and Min. Journ.*, LXXXVI (1908), p. 386.

minute. The screen was 500 light mesh and the screen area was 520 sq.in. Water supply was let into the top of the mortar box. Discharge

FEEDING WATER TO STAMPS.

	Trial I Tons.	Trial II Tons.	Trial III Tons.	Trial IV Tons.	Trial V Tons.	Trial VI Tons.
Coarse rock	6.153	5.723	5.740	5.941	5.771	6.630
Screened rock	6.253	5.797	8.878	6.034	6.390	6.627

was level with the top of dies. Dies were packed up on two false bottoms. The chuck blocks decreased in hight as dies wore. In trial II the conditions were the same, except the discharge was 1 in. above top of dies. In trial III the conditions were like II except the water was let in through four 1½-in. holes level with the ore feed chute at the back of the mortar. Trial IV was like III except that discharge was level with top of dies. Trial V was like IV except that the screen instead of being straight was made to give 173 sq.in. more area than the straight screen. In trial VI the conditions were like I except that the water was led in through five 1¼-in. holes in the back of mortar level with top of dies and discharge on mortar lip.

There are several mills on the Rand now equipped with back water feed. It effects a saving in water.

Cushioning Vibrations of Cam-Shafts.[1]—The dap or gain in the battery post on which the chair, holding the cam-shaft bearing sets, may be cut in an inch or two and filled with pieces of thick belting, or other cushioning material, for the cam-shaft bearing, which does not touch the chair except on the sides, the bottom of the chair being open.

Amalgamated Plates.[2]—The best conditions of plates for catching gold are as little slope as possible, as thin pulp and as small a rate of flow as possible. Solid particles should not be allowed to come to rest on the plate. Vibration helps but it has a tendency to form hard amalgam and is more expensive. Best results are obtained when the amalgam forms a rough and almost fibrous nape on the plates with a slight excess of mercury; the dressing, sprinkling and cleaning up should be carried out with this point in view. After cleaning and dressing the amalgam should be left on the plate in cross ridges made by the brush. Starting from this time the nape begins to form and continues until the amalgam is hard. Mercury is then sprinkled on, but no brush is used. Results at the Tucker mill in Gilpin county, Colo., obtained by taking samples of the pulp at the mortar lip and at the tail of the plate and calculating the extraction show that the extraction is about 40 per cent. just after

[1] Algernon Del Mar, *Min. and Sci. Press*, XCVII (1908), p. 877.
[2] J. H. Haynes, *West. Chem. and Met.*, IV (1908), p. 233.

clean-up of the plate, and increases to a maximum of 86 per cent. 15 hours after clean-up. From here it decreases to 65 per cent. at the end of 24 hours, when the plate is again cleaned. Dressing at intervals of eight hours had the effect of dropping the extraction from 70 or 80 per cent. down to 40 per cent. Where only coarse gold is present in the ore the maximum extraction was obtained just after clean-up. In this case the best extraction was obtained by using more mercury and having clean-up as often as possible.

Where stamping is carried on in cyanide solution there is a dissolving action on the gold on the plate; also the mercury and amalgam are softened by cyanide and flow off leaving a hard plate. Here the extraction is a maximum just after clean up and decreases from this point on, owing to the fact that gold in the form of amalgam is more readily attacked by cyanide than in the free state.

Silver Coating of Amalgamated Plates.[1]—To prevent the loss of gold through the inefficiency of a new plate the use of pure silver amalgam is recommended extending over a period of two or three weeks before the plate is used. To make the amalgam dissolve silver coin (about ¼ oz. per square foot of surface of plate) in dilute nitric acid using gentle heat and a porcelain dish. Evaporate to dryness over water bath. Heat until mass begins to fuse and all the bluish tinge is changed to grayish black (cupric nitrate changed to insoluble cupric oxide). Dissolve in water. Filter. Add pure mercury to the weight of about three times that of silver used, a few drops of nitric acid and a few pieces of bright iron floated on the mercury. The silver is precipitated and absorbed by the mercury, forming a pasty amalgam.

Accumulation and Absorption of Gold Amalgam by Copper Plates.[2]—Edward Halse gives the results of experiments showing that actual absorption of gold into the copper is never more than a fraction of a grain per ton of ore milled. W. F. A. Thomae expresses a judgment in full accord with this, setting the limit of absorption at not more than 10 oz. to a plate 4.5x12 ft. At the Hidden Treasure mill, at Black Hawk, Colo., Arthur L. Collins found that plates which had been in use 10 years yielded 8 oz. of fine gold after being thoroughly scoured. Copper plate will absorb mercury to saturation, which point is reached at about 1 per cent. in approximately 16 days. The gold, instead of penetrating the copper, tends to accumulate in the form of hard scale upon its surface. The effect of silver plating is to restrain absorption at first, since the mercury has to diffuse through the silver, but eventually the absorption reaches saturation as in the case of plain copper.

[1] W. A. Caldecott, *Journ. Chem. Met. and Min. Society*, South Africa, IX (1908), p. 142.

[2] Edward Halse. W. F. A. Thomae. *Min. and Sci. Press*, XCVI (1908), p. 806; *Trans.* I. M. M., XVII (1908), pp. 482, 486, 492; *Eng. and Min. Journ.*, LXXXVI (1908), pp. 709, 996; *Min. Journ.*, LXXXIV (1908), p. 36; *Journ. Chem. Met. and Min. Soc.*, South Africa, IX (1908), pp. 62, 214; *Electrochem. and Met. Ind.*, VI (1908), p. 375.

The rate and amount of accumulation of hard scale (not absorption) varies extremely under different conditions. The Drumlummon mill, in Montana, and the Sucre mill, in Colombia, show such wide discrepancies as 0.499 and 1.721 dwt. fine gold per ton of ore milled at the two places through periods of about two years. The gold recovered by regular clean-ups at these two mills was respectively 4.65 and 9.49 dwt. per ton of ore. The rate of accumulation roughly follows the grade of the ore and it is found that the proportion of silver present materially affects the rapidity with which the scale will form.

The Pierce Amalgamator.[1]—The Pierce amalgamator is designed to recover very fine or rusty gold from mill pulp or more especially from placers. It is also claimed that this device may be used with advantage as a substitute for plates in front of stamp screens or even as a supplementary amalgamating device after the plates and quicksilver traps.

Home-Made Arrastra.[2]—A very neatly built arrastra is described in this article which will handle 20 tons of ore a day and the total cost of erection, at a mine in Central America, 65 miles from the coast, where the only communication is by means of pack mules, was $2400 exclusive of superintendence. This cost was made up as follows: tools and supplies brought from the United States, $650; lumber, whip-sawed by contract at a cost of $28 per thousand board feet, $225; labor, skilled at $1 and unskilled at 40 to 50c. per day, $1050; freight, $175; and miscellaneous, $300.

Amalgamation Methods.[3]—If we allow a flake of gold to pass the plates and go to the cyanide plant, we can get only 95 per cent. of it, as against the total we should have if caught on the plates. A striking difference is found in the condition of the plates as maintained by different amalgamators. Some keep the apron-plate quite hard while others keep the amalgam soft and plastic. A coarse, easily amalgamable gold will be readily caught on a hard plate, but fine gold requires a soft, plastic bed of amalgam. Less attention is required when running the plates hard.

Careful amalgamators who use considerable refinement in their methods, dress the apron-plates with a rag, taking a little amalgam off daily with the rag, entirely eschewing the use of a rubber, and taking no more on "clean-up" day than at any other time. They keep a good bed of amalgam on the first plate, and less on the second; after rubbing the quicksilver well in and getting an even texture across the plate, with any surplus amalgam pushed up to the head, they smooth down the amalgam by riffling or roaching across at right angles to the flow of the

[1] *Eng. and Min. Journ.*, LXXXV (1908), p. 112.
[2] Teodoro Köhncke, *Min. and Sci. Press*, XCVII (1908), p. 185.
[3] H. W. MacFarren, *Min. and Sci. Press*, XCVII (1908), p. 814.

pulp, with a whisk-broom. They aim to feed quicksilver hourly through the mortar, in such a quantity that very little will be needed in dressing the plates, and yet not enough to cause the brush lines to rub or disappear. The amalgam on the upper apron-plate should be of such a consistency that it can be pushed up with the finger and remain without flattening out. In appearance, this amalgam should look neither hard and dead, nor like a mirror, but should have a white frosted surface. The lower plates should be watched. The quicksilver and amalgam should not run down upon the lower plates. When any gold is caught on the last apron-plate, investigation is necessary. The practice of riffling or roaching across the soft amalgam has been condemned, on the ground that these riffles catch the fine iron and steel and the sulphides. The fact that these tiny grooves do this speaks well for their function in catching gold. However, these riffles can be avoided by smoothing down the amalgam with a fine-haired paint or kalsomining brush. Care should be used that all the particles of amalgam are bedded down.

Chisels, scrapers, and pumice stone should not be used on apron-plates unless the amalgam becomes so hard that it cannot be removed in any other way, and it is well to avoid the amalgam becoming as hard as that. Sweating plates is dangerous on account of salivating the workmen. Turn up the edges of the apron-plates and slip the ends well under one another, and the quicksilver will not work through the table to the floor. Drops between plates are in favor, as the amalgam seems to pile up at the drop; this is mainly due to too much quicksilver being fed, or the plate being cleaned so close that there is no amalgam remaining to act as a "binder" for the quicksilver. There should be a drop between the first and second plate, but others are superfluous, since but little should get below these plates. These drops interfere with quick dressing of the plates. A drop of $\frac{1}{2}$ in. is sufficient; more than $\frac{3}{4}$ in. is liable to scour. Foundations for the plate tables should be carried up from the ground, so as to be as independent of the mill-jar as possible, as it causes the quicksilver to separate out of the amalgam. In starting a new plate, run some low-grade coarse gold ore through until the plates take on a layer of amalgam. New plates washed with quicksilver do poor work, and consequently "skin-tight" clean-ups should be avoided, unless some of the amalgam is returned to the plates.

Use as few chemicals as possible. A little lye to cut the grease is good, also nitric acid solution for pickling the quicksilver. The quicksilver is sometimes "loaded" by first freeing from all adhering moisture and then adding metallic sodium in small lumps until the quicksilver just commences to amalgamate a bright nail. It can be used to advantage in starting new plates or in covering bare spots. Bare spots are

often treated by softening with strong cyanide solution and then rubbing amalgam in, but in most cases the spots are soon bare again. It appears better to cover the bare places by coaxing the deposition of amalgam from the edges toward the center. To do this, brighten the copper by burnishing with a little fine grit and then pushing the adjacent amalgam over and rubbing it well in.

The use of inside plates is a matter of individual preference. Some claim they should not be used; that they place the amalgam where it can easily be scoured off and lost, should the mortar overfeed. Others claim that in some cases fine gold can be caught by their use, that cannot otherwise be saved. All are agreed they should not be used if the same saving can be made without them. It is customary with many amalgamators to use an inside plate when running on rich rock, with the idea that the less the amount of amalgam to be handled on the plate, the less the loss will be. Inside plates must be carefully watched, especially on starting, and no bare spots should be allowed; they tend to spread like a rotten spot in an apple. At all times the feed must be kept just right. They should be kept hard, using the lip-plate as the indicator in feeding quicksilver.

The lip-plate should be wide, 12 or 15 in. if possible. Splash and other outside mortar-plates are a matter of individual opinion. There should be as large a surface here as possible. The usual conditions under which work is done today are with a fine-grained gold, a low-grade ore, and a large capacity—necessitating a narrow mortar and a low discharge, thus prohibiting the successful use of inside plates, or of catching much gold in the mortar-sands. This leaves only the outside mortar or lip-plates, and the apron-plates. The amalgam on the mortar-plates is in a hard form, not likely to run or slough off, and here is the place to hold it—a better place than on the inside plates, but not as good as the mortar-sands. The apron-plates are reserved for catching the gold that has escaped higher up. These mortar or lip-plates are scraped at intervals varying from a few days to a month, according to the richness of the rock.

The mill-man may experiment with a plate in the tail-box, the launder, and on the concentrators, but they must be dressed occasionally. Patent amalgamators to follow plates may be of advantage in some cases, but usually not, as it is the amalgamator's business to see that there is nothing left that can be caught by amalgamation, consequently they should only be put in on trial.

The practice of cleaning the removed amalgam on one of the apron-plates is bad for various reasons, although when the "treasure box" is used, the loss of gold is small. The easiest way to clean the amalgam is

to place it in a large gold-pan with an amalgamated bottom, on removing it from the plates; then, when at leisure, work it up on this amalgamated surface at the clean-up sink, catching the refuse and periodically grinding it up with quicksilver.

Where difficulty is experienced in amalgamating the gold, it is well to increase the grade of the apron-plates and use less water in the mortar. For a coarse, easily amalgamated gold, long apron-plates are unnecessary, though they may be required with fine gold or where an excess of water is used. Given a long plate, where the gold is practically all caught on the first few feet, there is on a large part of the plate a condition akin to that when running through a large tonnage of poor ore, where it appears that there is not sufficient metal plated to prevent loss, probably by attrition. No exact reason can be given for this poor work with extremely low-grade ore. It is analogous to what occurs in the zinc-boxes when a large volume of solution of low-gold content is allowed to run through. This condition may justify using a little rich ore to sweeten the mill-feed. Long plates may be of advantage where a distributing or collecting box at the head of the apron-plate is utilized to turn the pulp down one-half of the plate while the other half is being dressed, thus saving the labor involved in hanging-up and the loss of duty from stopping the battery. Every mortar should be built so that one of these boxes can be bolted to it, if desired, as they are convenient for using a wide lip-plate. They distribute the pulp evenly across the plate in case of an irregular discharge from the mortar due to the bad order of drop or to other cause. They also form a projecting lip over the plate-table, preventing leakage, and they reduce jarring of the table by close contact with the mortar. Where no distributing-box can be bolted to the mortar, the plate-table can be set a few inches below the mortar-lip to permit a wooden box being inserted for catching the flow. At the Empire mill, at Grass Valley, the tables are fitted to the mortars in this way, and when dressing a plate, a trough of sufficient length to carry the flowing pulp to the plate of an adjacent battery is used. These methods should only be practised if tests show that the tailing is only a little higher while the plate is carrying an overload.

With individual stamps, outside amalgamation is usual, although quicksilver has been successfully fed to the mortars. As these stamps require much water, a half plate should be fitted to each stamp, or a full-width plate to two stamps built together. The placing of two plate-tables to take the pulp from three individual stamps built together has not been a success on account of the inability to distribute the pulp evenly across the width of the plates. Too often in outside amalgamation the plates are dressed too wet, and then allowed to become hard before

dressing again. This trouble can be avoided by sprinkling a little quicksilver from a bag or bottle upon the dry spots at the head of the plate, without stopping the battery, but it must be done carefully. Outside amalgamation is interesting and offers excellent opportunity to study plate-work. On many ores it will be found possible to stop nearly all the gold on the first 12 inches of the upper plate, if quicksilver is dropped upon it as needed. When quicksilver is not supplied as required, and the plate becomes hard and coated, as when treating rich ore, the gold will tend to slip over and be caught farther down, while if too much quicksilver be used, the amalgam will run down, which is probably worse.

Amalgamation in cyanide solution presents no difficulties within the requirements made of it. As the solution usually does not exceed 1 lb. per ton in cyanide strength, it does not cause the plates any harm. Neither does it injure the hands of the workmen, though it may make them rough at times. A good remedy for this is to rub Albany grease into clean hands, and to work with gloves so as to allow the grease to remain on as long as possible. As the life of plates in solution is limited to from six to nine months, the devising of means for prolonging their life offers an excellent field for investigation. With silvered copper plates costing approximately $2 per square foot, the item for renewals is important. When amalgamating in cyanide solution, the plates will be eaten through in spots, and yet it may not be convenient to put in a new set; consequently it is necessary to build plate-tables that will prevent the quicksilver from working through. This can be best accomplished by using 2x4- or 4x4-in. planed, well-seasoned lumber, the length of the table, put together with tar of paraffine paint, and bolted tightly across the width of plate every 3 ft. If drops are used, do not allow the edge of the plates to project beyond their wooden backing, as they are gradually eaten down to dangerous knife-edges.

Number of Stamps in a Mortar.[1]—Comparative tests on a mortar with four stamps and one with five stamps showed a very slight advantage of the latter which was probably due to the extra agitation in the mortar caused by the extra stamp. The results tend to discredit the theory that five stamps do not give a proper cycle in crushing.

Milling Practice on the Rand.[2]—The majority of stamps dropping on the Witwatersrand at the present day weigh 1250 lb. each. The tendency to increase the weight is resulting in heavier stamps being installed in all new mills and in additions to old mills. The 300 stamps of the joint Simmer Deep-Jupiter mill will weigh 1700 lb. each. The 20 new stamps for the New Kleinfontein are to weigh 1635 lb., those of the

[1] J. Kennedy, *Journ. Chem., Met. and Min. Soc.*, South Africa, IX (1908), p. 46.
[2] *Min. and Sci. Press*, XCVI (1908), pp. 286 and 821.

Cinderella Deep 1680 lb., and those of the Wit Deep, 1400 lb. each. At the Luipaardsvlei Estate, Jack East, and joint Knights Deep-Simmer mills stamps weighing 1550 lb. have been in use for a considerable time. There are 73 tube-mills in use on the Rand, in 28 different mills, and the employment of them for regrinding the pulp issuing from the mortars has permitted an increase in stamp-duty, so that in 10 mills it is in excess of seven tons, the maximum being 9.2 tons per stamp per day in the Luipaardsvlei mill.

Improvements in Stamp Milling in South Africa.[1]—In the new 300-stamp mill for the Simmer Deep several innovations are being introduced. The bin framing and roof trusses will be of steel, but the battery framing, bin lining, floors and wall framing will be made of timber. The mortars rest directly on the concrete foundation, so doing away with cast-iron mortar blocks. The stamps will be the heaviest yet installed on the Rand, weighing 1670 lb. They are expected to give an output of eight tons per stamp per day. The mortars are being made wide enough so that the weight of the stamps can be increased to 1800 lb. The stamps will be in units of 10, each unit being driven by a separate motor.

New Mills on the Rand.[2]—The following table shows the details of new equipment being installed on the Rand.

NEW MILLS ON THE RAND.

Name of Company.	Number of Stamps.	Weight of Stamps, Pounds.	Number of Tube Mills.	Size of Tube Mills.	Tons per Month.	Duty per Stamp per Day, Tons.	Remarks.
Bantjes Consolidated	35	1,650	1	22 ft. x 5 ft. 6 in.	10,000	10	Electrically driven mill. Company purpose putting in their own prime movers.
Cinderella Deep.	50	1,700	2	16 ft. 6 in. x 4 ft.	12,000	8	Electrically driven mill.
Geduld Prop...	50	1,650	.....	.....	.....	7	Steam driven.
Jupiter (a).....	100	1,700	.....	.....	.....	.....	.....
Knight Central.	100	1,700	3	22 ft. x 5 ft. 6 in.	24,000	8	Mill will be steam driven.
Main Reef West	80	.....	.....	.....	.....	.....	Stamps are being erected at the back of the Con. Main Reef Mill.
Simmer Deep...	200	1,670	4	22 ft. x 5 ft. 6 in.	.....	.....	Electrically driven mill.
Vogelstruis Con. Deep...	40	1,650	.....	.....	10,000	.....	.....
West Rand Con.	20	1,700	2	.....	16,000	.....	Steam driven.
West Rand Con.	60	1,250	(b)	.....	.....	.....	.....

(a) Joint mill with Simmer Deep. (b) Old stamps.

The Transvaal—Costs and Recovery.[3]—The accompanying table shows the costs and recoveries made by the Consolidated Goldfields Company for 12 months ending Aug. 31, in each year.

[1] Eustace M. Weston, *Eng. and Min. Journ.*, LXXXV (1908), p. 355.
[2] *South African Min. Journ.*, Sept. 19, 1908, p. 802; *Journ. Chem., Met. and Min. Soc.*, South Africa, IX (1908), p. 216
[3] W. Fischer Wilkinson, *Eng. and Min. Journ.*, LXXXV (1908), p. 53.

CONSOLIDATED GOLDFIELD COMPANY.

		Tons Milled.	Stamps.	Tube Mills.	Per cent. Recovery.	Value of Yield.	Working Costs.	Working Profit.	Total Working Profit.
Simmer & Jack	1906	742,257	320.0	.....	91.6	$7.72	$4.90	$2.82	$2,095,243.20
	1907	844,404	320.0	4 (c)	89.9	7.78	4.50	3.28	2,777,047.74
Robinson Deep	1906	412,524	203.2	2	92.7	10.62	5.23	5.39	2,235,128.58
	1907	519,475	281.1	2	94.9	9.83	5.31	4.52	2,350,028.70
Knights Deep	1906	321,648	119.0	3 (d)	91.3	7.21	5.75	1.46	470,258.46
	1907	453,017	160.7	3	90.8	6.95	5.00	1.95	875,987.84
Simmer East	1906 (a)	252,649	118.2	.....	88.8	6.18	5.86	0.32	77,619.06
	1907	496,021	234.6	.....	89.2	5.94	5.21	0.73	368,558.10
Luipaards Vlei	1906 (b)	52,083	60.0	2 (e)	90.5	6.04	5.89	0.15	7,100.46
	1907	197,544	60.0	2	92.1	7.11	4.90	2.21	436,875.12

(a) 13.463 months. (b) 4.286 months. (c) Started June 1907. (d) Started September, 1905. (e) Started August, 1906.

West Australian Wet Crushing Plants.[1]—The accompanying table gives the details of the larger mills. The word slimes in West Australia

WEST AUSTRALIAN WET CRUSHING PLANTS.

Mine.	Number of Stamps.	Monthly Tonnage.	Weight of Stamps, Pounds.	Duty Tons.	Wear of Shoes and Dies.	Discharge, Inches.	Drops per Minute.	Hight Drop, Inches.	Screen.	District.
Ivanhoe	100	17,000	1200	5.5	2.25 d. per ton	2	104	7½	15 x 15	Kalgoorlie
Oroya-Brownhill	60	9 000	1100	6.48	4.26 oz. Shoes 4.56 oz. Dies	2	108	7¾	10 x 10	Kalgoorlie
Sons of Gwalia	50	10,000	1000	6.68	.....	3.5	108	8	20 x 20	Leonora
Lake View Consols	75	11,000	1200	5.5	4.3 oz. Shoes 2.5 oz. Dies	2	102	8	300 mesh	Kalgoorlie
Golden Horseshoe	150	24,500	1270	5.5	2.14 oz. Shoes 3.04 oz. Dies	3	104	8	15 x 15	Kalgoorlie
Great Fingall	100	22,500	1150	7.0	.....	3	106	8	12 x 12	Murchison

is used to indicate material which will yield an economic extraction within 24 hours' agitation with cyanide and which is sufficiently homogeneous to permit of filter-pressing. Depending on the nature of the ore the maximum grain in slimes varies from 100-mesh size to 200-mesh size. The ores are quartzose containing pyrite, pyrrhotite, and tellurides of gold, silver and mercury. There are two classes of mills, viz., the dry-crushing, all-roasting plants and the wet-crushing, concentrating plants. Stamps are used as primary crushers and fine grinding in the wet-crushing plants is accomplished by pans and tube mills. In wet crushing amalgamation is practised either on plates or in the grinding pans. Concentrators are used in wet-crushing plants after the stamps, after the grinding pans and after the tube mills. At the Oroya Brownhill the battery pulp is run over a shaking screen and the oversize is ground in pans before concentration.

The grinding pans are of the Forwood-Down type. Comparing pans and tube mills the former are superior for reducing coarse sands to

[1] G. W. Williams, *Eng. and Min. Journ.* LXXXV (1908), p. 345. *Journ Chem. Met. and Min. Soc* South Africa VIII (1908), pp. 232, 383. *Min. Journ.*, LXXXIII (1908), pp. 438, 467.

fine, but for reducing fine sands to below 150 mesh their superiority has not been so clearly demonstrated. In a test at the Ivanhoe mill the battery product from 20 stamps with 15x15-screen, containing 56 per cent. coarser than 60-mesh was reduced to a final product containing 95 per cent. finer than 150-mesh by two pans working in tandem, the slimes being removed from the product of the first pan before feeding to the second. The cost was 55c. per ton. The cost for a tube mill doing the same work was 61c. per ton. The pans were 5 ft. in diameter, driven at 57 r.p.m., and required 6.5 h.p. The tube mill was a Krupp, 13 ft. x 3 ft. 3 in., driven at 31 r.p.m. and requiring 20 h.p. In regular work at the Ivanhoe mill 5-ft. pan making 57 r.p.m. and using 6.5 h.p., grinds 10 tons per day. Sizing tests are as follows: on 40 mesh before grinding, 31.4 per cent., and after grinding, 0.6 per cent.; through 40 on 60 mesh, before, 21.8 per cent., after, 6.2 per cent.; through 60 on 100 mesh, before, 17.6 per cent., after, 28.7 per cent.; through 100 on 150 mesh, before, 6.1 per cent., after, 8 per cent.; through 150 mesh, before, 23.1 per cent., after, 56.5 per cent. The grinding efficiency is proportional to the circulation. For any given arrangement of shoes and dies the grinding efficiency is proportional, within limits, to the expenditure of power. Coarse sands grind best at high speed, fine sands at medium speed. A small increase in speed involves a large increase in power. The grinding efficiency is proportional to shoe area under equal conditions of circulation. Pans yield best results when working on a uniform product. The pulp usually fed to pans contains 25 per cent. by weight of sands.

The tube mill is an impact grinder. On the Oroya-Brownhill the tube mills are 13 ft. 7 in. x 3 ft. 8 in., make 32 r.p.m., require 21 h.p., and each contains 2.5 tons of flints. Chilled cast-iron liners 1 in. thick wear at the rate of 470 lb. per mill per month. Each mill grinds 27 tons per day through 150 mesh.

On the Golden Horseshoe and Lake View the tube mills are 16 ft. 4 in. x 4 ft. At 29 r.p.m. and using 30 h.p., each mill crushes about 33 tons of sand per day through 200 mesh. On the Lake View manganese steel liners wear at the rate of 250 lb. per month. The consumption

GOLDEN HORSESHOE; SIZING TESTS.

	Battery Pulp. 15x15 Mesh Screen. 22 I. S. W. G.	Feed to Tube Mill.	Discharge of Tube Mill.
On 50 mesh	27.0%	10.0%	3.7%
Through 50 on 100 mesh	18.0	52.0	46.0
Through 100 on 150 mesh	4.0	9.2	10.8
Through 150 on 200 mesh	1.0	5.5	3.5
Through 200 mesh	50.0	23.3	36.0

of flints averages 1000 lb. On the Golden Horseshoe a set of chilled steel liners 1 in. thick lasts about six months. Silex liners are not used in Kalgoorlie. The accompanying sizing tests are on Golden Horseshoe products.

Individual Mills Described.

Stamp-Mill Practice on the Mother Lode.[1]—The Lightner mine is about in the center, north and south, of the Mother Lode, and the ore is typical of this great belt. In 1905 fire destroyed what was thought to be the most modern and successful stamp-mill in California. In 1906 a new 40-stamp mill was erected along the same lines. This mill averaged five tons per stamp-day and did equally as good work as the older mill had done. Development increased the orebody to such an extent that 20 stamps were added in 1907. It was soon found that the newest stamps were in every way superior to the older ones; the loss in the tailings was no greater and the amalgam recovery inside the mortars was larger than in the old mill. The new 20 stamps require no more water for battery or for concentrators, and the life of both the shoes and dies is longer. This new plant is crushing from 10 to 15 per cent. more ore than the old mill, with no more expenditure per stamp for power, water, and labor, and with less cost for wear and tear.

Some of the details of the new mill are as follows: The stamps weigh about 1000 lb. each, and are given a drop of 7 in., at a speed of 106 drops per minute. The hight of discharge above the die is 7½ in.; punched screens of 30-mesh are used, and each battery has 308 sq.in. of clear screen-opening. The shoes are made of chrome steel and Pennington hammered steel, weighing, when new, 181 lb., and when worn out, from 25 to 30 lb. They last about 10 months. Semi-steel dies are used, 5½ in. high, weighing 82 lb. when new, and about 30 lb. when worn out, after a service of eight months. The mortars are known as the "close pattern," and there is no surplus room, either between the stamps themselves, or between the stamps and the linings of the mortar. The stems are held rigidly in place and in alinement, and are made more efficient as crushers by a patented iron guide known as the Pacific Battery Stem Guide. The mortar-blocks are of sugar pine, resting on concrete foundation.

Each battery is equipped with 88 sq.ft. of copper plate, 2½ oz. silver per square foot. These plates are arranged in lines 2 ft. wide by 22 ft. long, all on a grade of 2 in. per foot. Three 4-ft. Frue vanners take the pulp from each five stamps. Each battery requires a water supply of 8 gal. per minute and each concentrator a water supply of 1 gal. per

[1] Alex. Chalmers, *Min. and Sci. Press*, XCVII (1908), p. 785.

minute. The mill is driven by a 50-h.p. induction motor. The 20 stamps, 12 Frue vanners, 36 ft. of 4-in. mill line-shafting, and 34 ft. of 2-in. concentrator counter-shafting consume 40 h.p.

The tailings are sampled automatically, showing an average loss, on ore ranging from $3 to 6 per ton, of 20 to 25c. per ton of ore crushed. This loss is practically the same in the old and the new mill. Inside the mortars of the old mill the recovery was about 20 per cent., while in the new mill it never drops below 25 per cent. of the assay-value. Ordinarily it requires but six men to look after the entire 60-stamp mill. There are times when chips are coming freely from the mine; then two additional boys are required to keep the mortars free from these wood particles. It may be mentioned that the screen-frames are all equipped with a pocket, provided with a little hinged door, so that the mill-man can get his hand inside the mortar without removing the screen-frame. This device saves time on account of there being so many chips in the ore. The labor cost for milling averaged about 22½c. per ton with the old 40 stamps, while with the entire 60 stamps this cost ranges from 20 to 15c. per ton.

Treating Gold Ore in Rhodesia.[1]—At the Giant mine in the Hartley district the ore is a heavily mineralized chloritic schist mixed with iron oxides and quartz. It contains about 32 per cent. iron oxide and magnetite, 15 per cent. pyrite, 46 per cent. insoluble and 6 per cent. alumina. About 30 per cent. of the gold is bright and is readily caught with mercury; the remainder is rusty gold. The ore passes over grizzlies and the oversize is crushed in two rock breakers. There are 15 stamps each weighing 1250 lb., crushing 15 tons per stamp per 24 hours, dropping 7½ in. 103 times per minute. The wire screens on two batteries have ⅜-in.

SIZING TESTS OF TWO BATTERIES AT GIANT MINE.

	⅜-Inch Screen.	¼-Inch Screen.
On ⅜ inch	.....	3.12%
Through ⅜ on ¼ inch	2.50%	7.22
Through ¼ on 14 inch	7.38	10.22
Through 14 on ⅜ inch	6.15	8.00
Through ⅜ on 30 mesh	25.00	25.00
Through 30 on 60 mesh	12.50	12.45
Through 60 on 90 mesh	8.19	7.75
Through 90 on 120 mesh	5.00	5.21
Through 120 mesh	30.78	20.00

holes and on one battery ½-in. holes. The hight of discharge is 3 in. Screens with $\frac{1}{16}$-in. slotted holes crushed 10.5 tons per day. Wire screens with ⅛-in. holes gave trouble from clogging.

[1] R. C. H. Cooke, *Journ. Chem., Met. and Min. Soc.*, South Africa, IX (1908), p. 152.

The stamp pulp passes over an amalgamated plate 4 ft. long. The big tonnage and coarse pulp injures the plate in no way. The amalgam traps consist of a few ¾-in. pebbles laid in the bottom of the launder and overlaid with ⅜-in. screen. After leaving the traps the pulp passes to a spitzkasten 4 ft. square. The spigot product with a ratio of 0.8 to 1 part water to one part of dry sand goes to three tube mills (one spare) while the overflow with added water joins the tube mill product. The tube mills are 25 ft. long and 5 ft. diameter, lined with silex blocks. Liners last 31 weeks. French imported pebbles are consumed at the rate of 300 lb. daily. The charge of pebbles is 12 tons and the speed is 28 r.p.m.

SIZING TESTS OF TUBE MILLS AT GIANT MINE.

	Tube Mill Feed.	Tube Mill Product.
On ½ inch	1.02%	.....
Through ½ on ¼ inch	3.65	.....
Through ¼ on ⅛ inch	6.97	.....
Through ⅛ on ¼ inch	6.00	0.00%
Through ¼ on 30 mesh	23.83	1.56
Through 30 on 60 mesh	13.54	2.00
Through 60 on 90 mesh	8.33	8.00
Through 90 on 120 mesh	5.20	11.12
Through 120 mesh	30.79	76.00

After the tube mill come the 12 blanket tables (two spare), 12 ft. long and 2.5 ft. wide, with 12.5 per cent. slope. The feed has seven parts water to one part sand. The blanketing is special make, similar to the ordinary military rug, and is put on in strips 4½ ft. long, each strip overlapping 6 in. on the next strip below. Each blanket is removed and washed every half hour in a vat 5.5 ft. in diameter and 2.5 ft. deep. The yield of concentrates is about four to five tons per day. These are further concentrated on an elevated blanket table into about half a ton. The tailings (4.5 tons) join the tube mill product going to the first blanket tables. The half ton of concentrates is charged into a revolving amalgamating barrel with 300 c.c. of concentrated nitric acid. After four hours mercury is added and the treatment continued four hours more. The product is finally discharged on an ordinary mechanical batea and the amalgam recovered in the usual way.

The tailings from the first blanket tables flow over five narrow auxiliary blanket tables, whose concentrates are treated in the same way as the preceding and whose tailings go to a two-compartment spitzkasten, each compartment being 3.5 ft. square. The spigot product is elevated by two Frenier sand pumps back to the spitzkasten before the tube mills. The overflow with a ratio of 10 parts water to one part dry slime is distributed to five canvas tables each 20 ft. long and 2.5 ft. wide, covered with the

best fine white duck. The concentrates are recovered by washing in the same way as on the blanket tables and amount to five tons in 15 days. They are reconcentrated twice a month on an auxiliary canvas table.

The canvas table tailings are unwatered in two spitzkasten 11 ft. square, each placed in the center of a slime collecting vat 30 ft. in diameter. The spitzkasten spigot (about 180 tons dry ore) flows over a canvas table 100 ft. long. This pulp has a ratio of water to sand of $2\frac{1}{2}:1$. After the canvas table it goes to a spitzkasten 8 ft. square whose spigot product with one part sand to one part water flows away to a dam while the overflow goes to 40-ft. slime collecting vat. The slimes settling in the two 30-ft. vats and the 40-ft. vat are periodically discharged to waste while the peripheral overflows are run through filters and pumped back to be used again.

The original ore assays 0.40 oz. gold per ton. The amalgamated plates save 30 per cent. and the blanket tables 55 per cent. About 0.01 oz. per ton is saved on the first canvas tables and 0.10 oz. on the final canvas table. The tailings assay about 0.05 oz. The total extraction is 87.7 per cent.

Metallurgical Development at Goldfield, Nevada.[1]—Early in 1907 the Combination mill had settled down to the following method of treatment: The ore, which is essentially a silicified dacite enriched with gold, contains 50 per cent. quartz, 24 per cent. kaolinite, 15 per cent. alunite, 7 per cent. pyrite, 2.5 per cent. water, is delivered to (1).

1. Twenty stamps each weighing 1350 pounds and crushing through 12-mesh screens. To (2).
2. Four amalgamating plates. Amalgam to retort, pulp to (3).
3. Four classifiers. Spigots to (4) and overflow to (6).
4. One tube mill. To (5).
5. Amalgamating plates. Amalgam to retort; pulp to (6).
6. Cone classifiers. From (3) and (5); deliver spigots to (7) and overflow to (8).
7. One Deister table. Feed is all finer than 200-mesh. Concentrates to smelter and tailings to cyanide plant.
8. Six vanners. From (6); delivers concentrates to smelter and tailings to cyanide plant.

At the Nevada Gold Reduction Company's mill 450 tons per day are handled in the sampling department and 100 tons per day in the stamping and cyanide mills. One hundred tons of ore per day is delivered to (1) and the remainder is sold to smelters.

1. Twenty stamps each weighing 1250 pounds and crushing the ore through 12-mesh screens in a 0.5 to 1.25 pound cyanide solution. To (2).
2. Four amalgamating plates. Amalgam to retort; pulp to (3).
3. Eight Wilfley tables. Concentrates, assaying from $200 to $1000 per ton go to the smelter; sand to (4); slimes to (6).
4. Allis-Chalmers tube mill, 5 by 22 ft., grinding to a screen size of less than 100 apertures per linear inch. To (5).
5. Amalgamating plates. Amalgam to retort; pulp to (6).
6. Classifiers. From (3) and (5); deliver spigots to (7) and the overflow to the cyanide plant.
7. One Wilfley table. Concentrates to smelter and tailings, after being reground, go to the cyanide plant.

The Goldfield Consolidated Mill at Goldfield, Nevada.[2]—In this new 600-ton mill the silicious sulphide ore comes in railroad cars to (1).

[1] T. A. Rickard, *Min. and Sci. Press*, XCVI (1908), p. 840.
[2] P. E. Barbour, *Eng. and Min. Journ.*, LXXXVI (1908), pp. 416, 467; M. R. Lamb, *Mines and Minerals*, XXIX (1908) p. 209; *Mining Science*, LVII (1908), p. 443.

1. Bin holding 850 tons. By three shaking feeders to (2).
2. One No. 7½ K Gates breaker crushing to 2½ in. To (3).
3. Trommel 45-in. diameter, 14 ft. long, with 1½-in. holes in ¾-in. manganese steel plate. Oversize to (4); undersize to (5).
4. Two No. 4 K Gates breakers. To (5).
5. Belt conveyor, 26 in. wide, 370 ft. long, traveling 350 ft. per minute, elevating 125 ft. To (6).
6. One No. 8 Blake Denison weighing machine. To (7).
7. One No. 3 Vezin sampler, making a 5 per cent. cut. Sample to (8); reject to (12).
8. One set 26 x 15-in. rolls, crushing to ¾ in. To (9).
9. One No. 2 Vezin sampler making 5 per cent cut. Sample to (10); reject to (11).
10. Bin holding six tons. Once a day the sample is run through a feeder, No. 1 Vezin sampler making 5 per cent. cut, feeder, first sample grinder, second sample grinder (two passes) yielding a final sample of which 1 lb. represents 64,000 lb. of ore. All rejects go to (11). Each sample grinder makes a 50 per cent. cut.
11. Bucket elevator lifting 51 ft. with 6x4-in. buckets, 12 in. apart, running 300 ft. per minute. To (12) or (13).
12. One 26-in. distributing conveyer. To (13). Up to this point the plant runs eight hours per day and will handle 600 tons in that time.
13. Battery ore bins 18 ft. 8 in. wide, 20 ft. 6 in. deep and 187 ft. long, built of steel, lined with plank, capacity 4000 tons. By 20 Gates and Challenge feeders to (14).
14. One hundred stamps weighing 1050 lb. each, making 108 drops per minute, screen 16 mesh No. 21 steel wire, modified Homestake mortars with false bottoms. To (15).
15. Twenty amalgamated plates, 5x16 ft., sloping 1½ in. per foot, each divided into four sections 5x4 ft. x ¼ in. thick, coated with 1 oz. of silver per square foot. To (16).
16. Twenty cast iron mercury traps. To (17).
17. Twenty 24-in. double-cone hydraulic classifiers. Spigots between 16 and 150 mesh (65 per cent. of total) to (18); overflow to (24), or partly to (20) as hydraulic water.
18. Six Dorr classifiers. Settlings with 46 per cent moisture to (19); overflow to cyanide department.
19. Six Allis-Chalmers tube mills 5 x 22 ft. To (20).
20. Six 48-in. double cone hydraulic classifiers. Spigot to (21); overflow to (22).
21. Four 10x54-in. Frenier sand pumps. To (18).
22. Fourteen amalgamating plates 5x16 ft., sloping 2 in. per foot, each plate in four sections 5x4 ft. with 2 oz. silver per square foot. To (23).
23. Two amalgam traps made of 12-in. pipe. To (24).
24. Thirty 8-ft. Callow tanks. Spigots to (25); overflow to cyanide department.
25. Sixty Deister No. 3 tables. Concentrates; middlings, to (26); tailings to cyanide department.
26. Ten Deister No. 3 tables. Concentrates; tailings to cyanide department.

The mill is built of steel covered with asbestos-covered corrugated steel. The foundations, mortar blocks, etc., are of concrete. Electric power is used with individual motors for different groups of machines. The ore yields about 3 to 5 per cent. of concentrates so that the cyanide department has to handle 570 tons of dry slime per day. This mill represents the most advanced ideas in the milling of the Goldfield ores.

Pan Amalgamation at Zacatecas, Mexico.[1]—At the Bote mill the Boss continuous process is used. The ore goes to (1).

1. Gyratory breaker crushing to 1½ in. To (2).
2. Rolls, crushing to ¼ in. To (3).
3. Six 8-foot Chile mills, crushing to about 55 mesh (60 per cent. through 200-mesh). To (4).
4. Dewatering tanks. Thickened pulp with 50 per cent. moisture to (5).
5. Eighteen pans in series, followed by four settlers in series. Amalgam to retort; pulp to (6).
6. Seven 6-ft. Frue vanners and two Standard tables. Concentrates; tailings.

The pans are 5 ft. 4 in. diameter and 3 ft. deep. The muller makes 65 r.p.m. No grinding shoe is used. The pulp is heated to 150 deg. F. before going to the pans. In the first six pans, 300 kg. of mercury are charged; in the second six pans, 200 kg.; and in the last six pans, 100 kg. Six kilograms of copper sulphate and 30 kg. of salt are added per ton of ore. Nine or 10 kilos of this mixture are added every five minutes. Lime is added at the last pan to precipitate as much of the dissolved silver and gold as possible. The settlers are 8 ft. diameter and 3 ft. deep, with wooden shoes on the mullers, running at 14 r.p.m. The mill treats 2200 tons per month. The ore must assay 800 or 900 grams of silver per metric ton to pay for mining and milling. An extraction of 80 per cent. is claimed.

[1] C. T. Rice, *Eng. and Min. Journ.*, LXXXVI (1908), p. 401.

Treatment of Black Sands.

Treatment of Black Sand.[1]—The black sand concentrates from a season's run of the Cracker Jack mine, Douglas county, Oregon, amounted to 10 tons. To recover the gold and platinum these were first run over a screen with ½-in. holes, the quicksilver and amalgam being caught in a trap beneath the screen. The oversize was waste; the undersize was twice jigged in a hand jig of large capacity. The jig concentrates weighed 1500 lb. and were rich in black sand. The jig tailings carried some very fine black sand (about 10 lb. per ton), but this had practically no value. The 1500 pounds were washed over a series of steep mercury troughs to remove rusty gold and the last of the quicksilver, and next passed through a small hand jig, 10x12 in., which yielded 400 lb. of pure black sand. This was panned twice and yielded platinum concentrates and tailings free from platinum.

Mining and Separation of Black Sands.[2]—Arthur R. Townsend divides the operations attending black sand enterprises into three classes. First, the mining of the deposit; second, the separation of the heavy from the light sands; and third, the separation of the valuable constituents of the heavy sands from the valueless ones. The first operation is taken care of by hydraulic mining or dredging, and the third undoubtedly may be taken care of by some of the methods that have been worked out by the U. S. Geological Survey or by some modification of these methods. The second operation presents the greatest difficulty, involving as it does a concentration of from 33:1 to 50:1. Existing stationary devices for saving placer gold are not capable of saving all the product and doing close work. In saving all the gold they save a tremendous volume of sand and if one tried to save all the black sand a large quantity of white sand would likewise be saved with it. The amount of black sand saved at present in hydraulic and dredge riffles is extremely small, amounting to only a few thousandths of a per cent. of the gravel washed. A mill designed to wash and exclude pebbles and boulders, classify the sand and distribute it to percussion tables would undoubtedly be able to make a good saving, but would be unprofitable because of the enormous tonnages that must be treated in order to obtain a moderate tonnage of black sand. Mr. Townsend suggests some adaptation of the sluice and riffles, or dredge riffles, making a dirty concentrate perhaps 15 per cent. of the original material and containing 90 per cent. of the black sand mixed with a quantity of light sands. This concentrate could then be cleaned up by a properly designed mill yielding a clean black-sand product for further separation.

[1] H. B. Pulsifer, *Eng. and Min. Journ.*, LXXXVI (1908), p. 1003.
[2] *Eng. and Min. Journ.*, LXXXV (1908), p. 307.

GRAPHITE.

The United States produces approximately 20 per cent. of the world's output of graphite. This is insufficient to supply the domestic demand and considerable quantities are imported. Imports come chiefly from the island of Ceylon, where are situated the most extensive graphite deposits of the world. The accompanying table shows the production, imports and consumption of graphite for a period of years.

STATISTICS OF GRAPHITE IN THE UNITED STATES.

Year.	Refined Crystalline Graphite.						Amorphous Graphite. Production.		Artificial Graphite. Production.	
	Production.		Imports.		Consumption (c)					
	Pounds.	Value (b)	Pounds.	Value.	Pounds.	Value	Tons 2000 lb.	Value.	Pounds.	Value
1897	993,138	$ 44,691	19,113,920	$ 270,952	20,107,058	$ 315,643	1,200	$11,400	162,382	$10,149
1898	1,647,679	82,385	30,199,680	743,820	31,847,359	826,205	1,200	11,400	185,647	11,603
1899	3,632,608	145,304	41,586,000	1,990,649	45,218,608	2,135,953	1,030	8,240	405,870	32,475
1900	4,103,052	164,122	32,298,560	1,389,117	36,401,612	1,553,239	1,045	8,640	860,750	68,860
1901	3,967,612	135,914	32,029,760	895,010	36,997,372	1,067,921	809	31,800	2,500,000	119,000
1902	4,176,824	153,147	40,857,600	1,168,554	45,034,424	1,322,401	4,739	55,964	2,358,828	110,700
1903	4,525,700	164,247	32,012,000	1,207,700	36,537,700	1,371,947	16,591	71,384	2,620,000	178,670
1904	4,357,927	162,332	25,350,000	905,581	29,707,927	1,067,913	19,115	102,925	3,248,000	217,790
1905	4,260,656	170,426	34,914,611	983,034	39,175,267	1,153,460	d21,953	80,639	4,595,500	313,979
1906	4,894,483	170,866	50,974,336	1,554,212	55,868,819	1,725,098	d16,853	(e)	4,868,000	312,764
1907	4,586,149	149,548	40,962,000	1,777,389	45,548,149	1,926,937	d26,962	138,381	6,924,000	483,717
1908	3,433,039	149,763	22,912,714	762,267	26,345,753	876,030	d1,443	75,250	7,385,511	502,667

(b) Nominal. (c) Neglecting the small re-exports of foreign product. The exports of graphite from the United States were valued at $324 in 1901, $365 in 1902, $4,220 in 1903, $8,455 in 1904, and $91 in 1905. (d) Statistics of U. S. Geological Survey. (e) Not reported separately.

GRAPHITE IN FOREIGN COUNTRIES.

Africa.—The discovery of an extensive deposit of graphite is reported in the Pietersburg district, in the Transvaal. Two beds, one 7 ft. and the other 4 ft. in width, have been uncovered, the mineral occurring between crystalline schists and gray granite. The Transvaal Graphite Mining and Manufacturing Company, Limited, has been formed at Johannesburg to work the property. The office of the company is at 21-24 Bettelheim Buildings, Simmonds street, Johannesburg.

In addition to the deposits on the South coast, Cape Colony, graphite-bearing lodes have been explored in Zululand. The Nondweni Graphite Syndicate, Limited, has been organized for the purpose of developing these deposits.

Austria.—At Wranowitz, Bohemia, the discovery of a thick bed of graphite was reported. It is said that the graphite mine at Leims, belonging to Baron Meyr von Melnhof, was finally closed Dec. 31, 1908.

Canada.—The production of graphite in Canada during 1908 was 251 tons, valued at $5565. The deposits which have been chiefly developed occur in the counties of Labelle and Argenteuil, in the Province of Quebec. In the former county the graphite is found most prominently in a more or less wide band of gneiss. In Argenteuil county it occurs almost invariably in the limestones, which are strongly developed in the township of Grenville and the townships to the north.

Ceylon.—This country is the world's chief producer of graphite. During 1908 the industry suffered greatly on account of the general financial depression. Many of the mines were closed, and at present only those are worked in which it is necessary to maintain operations to prevent flooding.

Korea.—An American syndicate has recently been granted a concession of graphite mines in southeastern Korea, which are believed to promise profitable returns.

Mexico.—About 4000 tons of graphite were mined in Sonora in 1908. This was shipped to the United States for grinding and milling.

New Zealand.—The discovery of a promising graphite deposit is reported at Cuttle Cove, Preservation inlet.

WORLD'S PRODUCTION OF GRAPHITE.
(In metric tons.)

Year	Austria	Canada	Ceylon (d)	Germany	India	Italy	Japan	Mexico (f)	Sweden	United States (b)	Totals.
1897.....	33,504	305	19,275	3,861	61	5,650	204	759	99	450	69,163
1898.....	33,062	1,107	78,509	4,593	61	346	346	1,857	50	824	125,006
1899.....	31,819	1,188	29,037	5,196	1 548	9,990	55	2,305	35	1,648	80,962
1900.....	33,663	1,743	19,168	9,248	1,858	9,720	942	2,561	84	1,799	81,938
1901.....	29,992	2,004	22,707	4,435	2,530	10,313	88	762	(e)56	1,800	74,688
1902.....	29,527	993	25,593	5,023	4,648	9,210	97	1,434	63	1,895	78,371
1903.....	29,590	660	24,492	3,720	3,448	7,920	114	1,404	25	2,053	73,436
1904.....	28,620	410	26,478	3,784	3,800	9,765	216	970	55	2,045	76,143
1905.....	34,416	491	31,134	4,921	2,324	10,572	209	970	40	1,933	87,010
1906.....	38,117	405	36,578	4,055	2,642	10,805	177	3,915	(c)	2,220	98,872
1907.....	40,425	525	(c)	4,033	2,472	9,260	72	3,202	(c)	2,080	(c)
1908.....	(c)	227	(c)	4,844	(c)	(c)	(c)	1,076	(c)	1,557	.....

(b) Crystalline graphite. (c) Statistics not yet available. (d) The figures for 1897 and 1899 are exports; the enormous production in 1898 as reported in official government publications is not reflected in the exports for that year, which amounted to 24,349 metric tons. (e) The production of crude graphite in 1901 was 1727 tons. (f) Exports.

GRAPHITE MINING IN THE UNITED STATES.

BY F. W. IHNE.

Graphite has a dark steel-gray, somewhat silvery, color with metallic luster. Its hardness is only 1.15 to 2, and its specific gravity 2.25 to 2.50. Its streak is shiny, grayish-black, and it has an unctuous

feel. It is insoluble in acids and incombustible at any degree of heat below that generated in an electrical furnace or by oxygen gas. Exposed to such very high heat it does not burn like coal, but decomposes into carbon dioxide like the diamond. Graphite crystallizes in the hexagonal system, in prisms and tables of six sides. Perfect crystals, always small, are rare; at least they are very seldom obtained in perfect condition on account of their fragile nature, and the fact that they are often enclosed in hard rock, or hidden in soft, friable, granular graphite. The mineralogical occurrence of graphite is in laminated or scaly, columnar, granular, crystalline and massive forms, and in an earthy, amorphous form. Molybdenite is often mistaken for graphite, but a distinction can easily be made by comparing the relative weights, molybdenite being about twice as heavy as graphite, while its streak and color are more dull and lead-gray. Finally the blow-pipe will show that graphite is infusible, while molybdenite gives a sulphur reaction. The distinction can also be made with nitric acid, which dissolves molybdenite, but has not the slightest effect on graphite.

The occurrences of graphite in the United States are numerous. It is spread over the entire country and is found in nearly all formations, from the old Laurentian to the younger coal measures, although the most and best deposits occur in the older crystalline and non-fossiliferous rocks, and the ancient metamorphic schists. But frequent as are the occurrences of graphite, the number of deposits that can be worked with a prospect of profit is small, most of them suffering from lack of sufficiency in quality or quantity, or both.

The trade has divided graphite into two general classes, viz., the crystalline and the amorphous. Some scientists have recently attempted to introduce a new classification, based on the behavior with hot nitric acid and potassium chlorate. On the basis of the yielding or indifference of graphite to that acid reaction these scientists established a rule for the division of all graphites into two classes, calling the ones which yield, "graphite," and those which do not, "graphitite." According to this rule, the graphites from Ticonderoga, N. Y., and Clay county, Ala., are real graphites, while the graphite from Raton, Colfax county, N. M., and the artificial graphite from Niagara Falls are graphitites. Interesting as this new classification may be, it is of no practical value to the trade, and even Sir Benjamin Brodie's process of making very fine pure graphite is too expensive to be of any practical use. Therefore, I still hold to the old well-established and universally adopted classification, viz., crystalline and amorphous. Although both kinds of graphite are frequently found in the United States the deposits of the amorphous far

exceed those of the crystalline class in quantity, if not in quality and value.

Crystalline Graphite.

Crystalline graphite occurs in the form of small aggregations, scales and grains (rarely as regular prismatic crystals) intermingled with other minerals in veins or beds, deposited between the joints of schists and layers of rocks, or disseminated through granite, gneiss, crystalline metamorphic schists, crystalline limestone and quartzite. No graphite is found in a chemically pure state; even the cleanest crystals contain a small fraction of foreign matter, and a graphite containing 99.7 per cent. of carbon seems to be the purest that nature has produced as a mineral.

Alabama.—This State has, especially in Clay, Chilton and Coosa counties, large deposits of rock containing flake graphite. In Clay county, there is a zone of soft, mica-free weathered granite, containing graphite, about 10 miles long, many hundred feet wide, and of unknown but apparently great depth, excavations to a depth of 75 ft. having been made without disclosing any change in the formation, except that the flakes of graphite increased somewhat in quantity and size. This graphite is disseminated with great uniformity and I believe that this deposit is the largest and most regular of any known in the United States. In the upper part of the deposit there is a small amount of sesquioxide of iron, but otherwise there are no objectionable impurities, the absence of mica being in particular a great advantage. I have taken a large number of samples from many different places in this zone, which have shown a graphite content of 3.5 to 5.25 per cent., the graphite being wholly of the flake variety. Dr. Edgard Everhart, of the Geological Survey of Georgia, who examined the easterly portion of this zone says: "In no place where an exposure had been made was there found anything but decomposed granite with flake graphite. No pockets of graphite were observed, but it seemed to be equally disseminated through the mass of the rock. Throughout the decomposed granite, there was found only the graphite that is known as flake. Samples of ore were taken at different points and at different depths. In every instance, the characteristic graphite-bearing granite was found. The average yield of graphite is over 4.5 per cent., and the analyses show that the percentage increases with the depth."

This deposit is favorably situated for mining and milling, much of the ground being capable of extraction by open cuts and adit levels, while several creeks afford good water supply. So far the deposit has been exploited by only two concerns, viz.; the Allen Graphite Company, about nine miles west of Ashland, Clay county, and the Redding Graphite

Company, operating a smaller plant, 2½ miles east of the Allen company. The Allen Graphite Company gets its ore from a wide open cut, 55 ft. deep, which from top to bottom shows a uniform quality of material, yielding somewhat over 4 per cent. of graphite. Near the cut is a crusher and dryer, whence a narrow gage railroad leads to the concentrating plant about 1½ miles distant. The extraction of graphite is about 2 to 2.5 per cent.; the loss of approximately 50 per cent. can be reduced considerably by providing some auxiliary machinery, which probably will be done. With the help of a new, but simple machine, I have obtained over 4 per cent. of good, flake graphite, assaying over 90 per cent. carbon, from ore containing 4.5 per cent. carbon. The output of the Allen Graphite Company is 1800 to 2800 lb. per day. Nearer to Ashland, in the same zone, is the property of the Victoria Flake Graphite Company, on which many openings have been made, but a concentrating plant has not yet been erected. The above statement made by Dr. Everhart has reference to the Victoria property.

In Chilton county, there is another zone of graphite, in micaceous schist. I traced this zone for a length of about four miles, and a width of several hundred feet, beginning about two miles east of the Mountain Creek station on the Louisville & Nashville railway, and trending from there in a northeasterly direction. In this district, Dr. Fitzpatrick of Montgomery, Ala., about six years ago erected a concentrating plant, which after some trials and tribulations is now turning out about 3000 lb. of good flake graphite per day at a satisfactory profit. The graphite-bearing formation at this place is at least 200 ft. in thickness. Another milling plant has been built between the Fitzpatrick plant and Mountain Creek station, but has not yet been operated. In Coosa county some mining for flake graphite has been done, but details are not yet known to me.

Colorado.—A large deposit of good flake graphite is said to have been discovered in this State, but I have not been able to obtain details respecting it.

Montana.—On the northeastern slope of the Ruby mountains in Madison county I observed a rather small but good-looking deposit of foliated graphite. At Dillon, Beaverhead county, the Crystal Graphite Company has produced some fine flake graphite from a mine in that vicinity, but this company does not appear to be operating at present.

New Mexico.—There is an outcrop of good foliated graphite, a few miles east of Las Palomas, Sierra county, but remoteness from railway checks development.

New Jersey.—In New Jersey some graphite mining is done by the Ashbury Graphite Mills in Warren county and the Raritan Graphite

Company, at High Bridge in Hunterdon county. The crude material is crystalline, found in beds of partly decomposed gneiss, with a tenor of 3 to 6 per cent. graphite.

New York.—The most productive deposits of crystalline graphite known in the United States are found in the northern part of New York, almost exclusively in the counties of Essex, Warren, Washington and Saratoga and in the southeastern portion of Pennsylvania. Near Ticonderoga, Crown Point, Rock Pond and vicinity in Essex county, and near Graphite and Hague in Warren county, N. Y., are the celebrated mines of the Joseph Dixon Crucible Company and those of the Crown Point Graphite Company, the Champlain Graphite Company, the Empire Graphite Company, the Glen Falls Graphite Company, the International Graphite Company and others. The occurrence of graphite in this region is variable. The Dixon Crucible Company has in its mine near Graphite a fissure vein of small thickness, but great length and depth, cutting through gneiss. There are others which are opened on small and short veinlets, pockets and beds in quartzite and crystalline limestone, or in shales and schists. The principal impurities are quartz, feldspar, pyrites, calcite and mica. The graphite content of the rock that comes to the mills seems to be about 4 to 6 per cent. and the average recovery of saleable graphite about 3 per cent., though in a few cases, for instance in the American mine near Hague, the carbon content may go as high as 10 or 12 per cent.

The refining of the New York graphite is difficult and subject to a high percentage of loss. A published statement of the Dixon Crucible Company, a few years ago, gave the graphite content of the rock in its mines as 10 per cent., of which one-half was recovered in the mills. Since then the content must have decreased considerably, but if it be correct that the available content has sunk to 3 per cent. or even as low as 2½ per cent., the operating of graphite mines would, notwithstanding the complicated and expensive process of concentrating, still be a paying business, if there be enough material to warrant operation on a large scale.

The amount of graphite in the rock is often less important than some other factors. A great deal depends on the kind of graphite and the nature of the impurities mingled with it. If 20 or 30 per cent. of finely granular or amorphous graphite of good quality is intimately mixed with quartz, iron ore, pyrites and mica, as frequently is the case of graphite occurring in metamorphic mica schists, or if such a percentage of amorphous graphite occurs in an earthy material, the deposit might be workable with some profit, but it is certainly of much less value than an occurrence of a large mass of granite, gneiss, or quartzite containing only 2 to 3 per cent. of flake graphite. The ease of cleaning, and the market

value of the product means often much more than the graphite tenor of the crude material.

The graphite of the Adirondack district, though generally of the crystalline class, is not always flaky, being frequently microcrystalline and containing sometimes a considerable percentage of a very fine dense graphite that stands much nearer to the amorphous than to the crystalline class. The flaky graphite from Ticonderoga (American mine) and from some other mines in that section, if well prepared, is of the best kind, being useful for crucibles and other refractory articles and of particular fitness as a lubricant.

The concentrating and refining of the graphite in the Adirondack region is to a great extent done as first developed at the Dixon works, using partly a wet and partly a dry process, the latter being especially applied to the finishing. After crushing (at a few places by stamps), log-washers, screens and stationary buddles are used for wet concentrating, while dryers, air-jigs, underrunner buhr mills of the Munson type, and the Abbé pebble mills (Dixon's have four of the former and five of the latter), Hooper pneumatic machines and bolting chests are used for the finishing. The Dixon company, which is the only one in the district to use buddles, as far as I know, operates two plants, one at Graphite, near Hague, for concentrating and the other for finishing at Ticonderoga. The Dixon company has equipped its mills with excellent machinery and turns out mineral of the best quality, for which it obtains 6 to 30c. per lb. I do not believe that it sells much of the flake as low as 6c. a lb. and think that will be only for "paint stock" and a low grade of foundry facing, as I have samples of the flake, far from the best, which sold for 12c. per lb. The prices from 20c. upward, of course, are paid only for a limited quantity used for special purposes. The Dixon company utilizes even its "dust," an "omnium-gatherum" of very small flakes and powder mixed with all kinds of impurities, very difficult to separate and therefore usually considered as being nearly worthless and rather an unavoidable obstacle. There is not much demand for this kind of graphite, although often quite high in carbon, and for the small amount that can be disposed of, no higher price can be obtained than from 1 to 1½c. per lb. However, the Dixon company has the advantage over many other refiners, of being able to use a great deal of this dust by mixing it with imported graphite and working it into crucibles.

North Carolina.—This State is rich in amorphous, granular and mixed graphite, also there are many occurrences of flaky graphite, some of which have been developed. In the western portion of the State, the Southern Graphite Company has a great deal of amorphous and granular graphite. Veins of flaky graphite have been found in Macon, Cleveland, Ruther-

ford, Yancey, Mitchel, and Alexander and other counties, pieces of the granular mineral ranging from the size of a hickory nut to that of a cocoanut. Such pieces are found loose in and upon the soil, and in irregular beds and pockets in the sub-soil, or in small veinlets in weathered parts of the micaceous schist. Probably hundreds of tons could be picked up from the plowed lands in these counties, the mineral being sometimes mixed with amorphous.

Pennsylvania.—After New York, Pennsylvania is the largest producer of crystalline graphite. Nearly all of its mines are situated in the counties of Chester and Berks in the southeastern portion of the State, especially in Chester county. Near Chester Springs there are nine mills built by as many companies of which a few recently have been consolidated. All of them get their raw material from deposits of moderate size in mica schist in the vicinity of Chester Springs, which on an average may run 3 per cent. in graphite, although that of the Chester Graphite Company and the United States Graphite Company is somewhat higher. Besides these companies are the Federal Graphite Company, the Sterling Graphite Company, the Parker Graphite Company, the National Graphite Company and others. The Federal Graphite Company is going soon to commence operations again; also the National Graphite Company; while a new mill is in course of erection about three miles from Chester Springs.

The machinery used for crushing, separating and refining in this district is largely the same as in the Adirondack region, but the Chester county graphite mills, while having often more pneumatic machines than they need, should be better equipped with machinery for converting the "dust" into a higher grade of graphite. The refining of the graphite by means of magnetic separators and the treatment with hydrochloric acid, tried once by the Federal Graphite Company, has not proved to be successful.

The product which this district turns out is of a high grade and brings a good price. However, the output is not yet very large. For instance, the Chester Graphite Company, the only one that uses, like the Dixon company, buddles for the first concentrating, produces only 1200 to 1800 lb. of flakes per day, making three grades, which it sells at 8, 8½ and 9c. per lb. The output of the United States Graphite Company is about 4000 lb. per day; that of the Sterling Graphite Company, 800 to 1200 lb. The total production of crystalline graphite in Pennsylvania in 1908 was probably in the neighborhood of 800,000 pounds.

South Carolina.—In Cherokee county of this State I have seen a deposit of fine crystalline graphite, 6 to 8 in. wide in a hard, schistose rock. This has not yet been developed.

South Dakota.—A few deposits of good, crystalline graphite occur in Custer and Pennington counties, but the larger of them are remote from railways, which has so far prevented exploitation.

Virginia.—A few deposits of crystalline graphite have been opened in Albemarle and Orange counties, in the foot hills of the Blue Ridge. The Naylor-Bruce Graphite Company, of Charlottesville, is working there and is reported to have developed a few good deposits. The occurrence in this region, according to E. K. Judd, is in veins of 13 in. to 8 ft. in width in gneiss and syenite.

Wyoming.—Deposits of impure flaky graphite occur in Halleck and Sibyl cañons, about 30 miles north of Laramie.

Amorphous Graphite.

Occurrences of amorphous graphite, especially those where a small amount of the crystalline mineral is mixed with the amorphous, or where a little graphite is disseminated through a large mass of earthy, argillaceous, or carboniferous material, giving the whole a slightly graphitic character, are very common in the United States. Chief in importance among the occurrences are the graphitic, semi-anthracite deposits of Rhode Island, the Baraga carbonaceous schists of Michigan, and the deposits in Bartow county, Ga. The entire output of Rhode Island and Michigan graphite is consumed in the manufacture of paint and foundry facings. The Georgia graphite is employed exclusively as a filler for fertilizer, which it improves in color and increases in weight.

Large deposits of such amorphous graphite occur in Portage county, Wis.; in Wake, Catawba, Macon and Hayward counties in North Carolina; in Pennington county, S. D.; in Halleck Cañon, Albany county, Wyo.; and near Raton, N. M. Nearly all of these deposits have been worked at intervals, but none to any considerable extent except that in Wisconsin, the raw material of which has been ground and to a certain degree refined in a factory at Stevenspoint, Wis., where it has been worked into paint and foundry facings that yield a fair profit.

At many places in the vast area of mica schists and other crystalline rocks in North Carolina and Georgia, there are graphitic strata. In these a part of the mica appears to be left out, and to be replaced by graphite. The deposits graphitized in this way are at many localities of great length, width and depth, containing in the aggregate a large amount of good, amorphous graphite, but its separation is no easy task. Nevertheless it can be done; I have extracted from such rock as much as 3 per cent. of very fine graphite, suitable for the manufacture of lead pencils and good enough to be shipped abroad. A large zone of such graphitized mica schist is exposed in MacDowell county, N. C., a few hundred yards

north of Graphiteville, a station on the Southern Railway. This deposit is known for a length of about four miles and a width of nearly 300 ft. and is so situated as to be capable of attack by quarrying. From samples of this deposit I have obtained 5 to 8 per cent. of amorphous and 2 to 3 per cent. of microcrystalline graphite of a quality that would command a good price.

Wisconsin. (By W. O. Hotchkiss.)—Graphitic schist has been mined in a small way at Junction City on the Chicago, Milwaukee & St. Paul and Wisconsin Central lines in the central part of the State for several years. The operation is conducted by the Pioneer Graphite Company, which has a plant for grinding paint. Both graphite and iron ore paint are made. About 170 tons of graphitic material were mined in 1908, and it is planned to take out about 200 tons in 1909. A new shaft was sunk in 1908 and a better quality of material was found in a lense that stopped about 40 ft. from the surface. The shaft is about 70 ft. deep and tunnels are driven from the bottom along the strike of the schist.

The Market for Graphite.

By Reginald Meeks.

Because of the wide variation in quality in the numerous grades, graphite is a complex commodity. It is stated by dealers that practically no two consignments of the same "grade" are identical and moreover a lump, weighing perhaps 1 lb., will often yield five or six distinct qualities. Hence there is usually a wide range in quotations for each of the several classes or grades upon the market. In the United States the consumption of plumbago is confined to domestic, Mexican and Ceylon varieties; the last is by far the most important.

Domestic Plumbago.

The domestic plumbago produced and marketed in the United States is classified as No. 1 and No. 2 lump, powdered and dust. No. 1 lump is a fine selected grade containing about 90 per cent. carbon and is worth 3c. per lb. No. 2 is an inferior grade containing 80 to 85 per cent. carbon, the remainder being slate, quartz, etc. The price varies from $45 to $52.50 per 2000 lb. Powdered domestic graphite is prepared by grinding and is frequently mixed with talc. No. 1 powdered is quoted at 4¼c. per lb. and No. 2 at 3½c. per lb. Dust fetches as low as $22 per 2000 lb. Domestic grades are prepared in the United States and are marketed both in this country and in Germany. They are used for foundry facings, paints, lubricants, and many minor purposes, but seldom for crucibles and high-grade stove polish. The product of the Dixon Crucible

Company of Jersey City, N. J., is probably the only exception; its New York deposits are of sufficiently high grade for the manufacture of crucibles as well as stove polish, pencils, lubricants, etc. The United States Graphite Company operates mines of amorphous graphite in Sonora, Mexico, about 250 miles south of the international boundary line and 60 miles east of the Gulf of California. There is a 30-mile haul by mule team to Torres whence the graphite is shipped to Saginaw, Mich., for treatment. The average analysis of the crude material is as follows: Graphitic carbon, 84.66 per cent.; combined moisture, 1.36; insoluble SiO_2, 4.22; soluble SiO_2, 4.05; Al_2O_3, 4.31; Fe_2O_3, 0.36; CaO, 0.65; MgO, 0.39 per cent. After cleaning, sorting, grinding and floating in air the company prepares the graphite for the manufacture of pencils, paint, stove polish, foundry facings, lubricants and greases, pipe-joint compound, electrotyping graphite, for use in powder glazing and shot polishing, for rheostats, brushes, batteries, packing, bearings, etc.

Manufacturers of lubricants from amorphous graphite claim that this variety is superior to flake graphite for their purposes because of its greater spreading capacity and because the infinitesimal particles enter and fill the microscopic holes or pores in the surface of the bearing, whereas the flakes are too large to accomplish that result. However a discussion of the relative merits of the two forms of plumbago does not come within the scope of this article.

Although the United States produces annually about 20 per cent. of the world's supply of graphite, it has been necessary to import an average of about 36,000,000 lb. per year during the last 10 years.

Grades and Prices of Ceylon Graphite.

The trade divides Ceylon plumbago into four general classifications known as lump (large and ordinary), chips, dust and flying dust. These names refer to the size of the product and in each size the quality varies from the highest to the lowest, each having a price according to the percentage of carbon, silica and iron and also depending upon the color and texture.

The large lumps, as they come from the mine are broken by hand and sorted into smaller lumps each having the fibers or flakes running in one direction. During this operation much fine material is produced and those particles which are too small to be classed as lump are separated into a class known as chips. The still finer particles are screened and sized and marketed as dust. Some fine dust floats in the air and settles on tarpaulins. This product is known as "flying dust." During the monsoons, foreign matter is carried by the wind and finds its way into the

dust, flying dust and even chip. Hence the quality of these grades is never so high as lump.

The accompanying table gives quotations for the commercial grades of graphite in New York and in Colombo, Ceylon:

PRICES OF COMMERCIAL GRAPHITE, OCTOBER, 1908.

Grade.	New York cents per pound.	Ceylon rupees per long ton. (b)
Lump	Best 9@9½ Med. 7½@8 Poor 4@5	Large 225@525 Ordinary 200@550
Chips	Best 6@6½ Med. 4½@5½ Poor 3½@4 (a)	150@375
Dust	Best 4@4½ Med. 2½@3½ Poor 2½	50@300
Flying Dust	Very poor 1½@2 Best 3½@3½ Med. 2½ Poor 1½@1½	40@130

(a) Difficult to market. (b) Current rates of freight, Colombo to New York, 27s. 6d. @ 30s. per long ton. Note—One rupee = $0.324.

The quality of plumbago is the principal factor which determines the price of the various grades or sizes. The chief impurities are iron and silica and of the two iron is the more objectionable. Too much iron renders graphite unfit for crucibles or other refractories and lessens its value for stove polish. In all plumbago the desideratum is a high carbon content. Fancy grades contain 95 per cent. or more; high grades, over 90 per cent.; good grades, 85 to 90; medium grades, 80 to 85; and poor grades 70 to 80 per cent. Graphite with less than 70 per cent. carbon is rarely marketable at any price. Fancy grades are too scarce to be regularly quoted and are used to bring up the quality of a consignment. A nominal quotation is 12c. per lb. Quality depends on: (1) carbon content; (2) relative amount of iron and silica; (3) texture; (4) color; (5) brightness.

Uses.

The highest grades of Ceylon graphite are used for the manufacture of crucibles for making crucible-steel and for melting metals. The annual consumption for this purpose is estimated at 21,000,000 lb. The chief requisites for crucible manufacture are high carbon and low iron contents. Iron, if in great excess, fuses with the clay and the crucible quickly "burns out."

Next in importance is the manufacture of stove polish. Carbon should be about 85 to 90 per cent. and iron low, but the grade need not be so pure as for crucible use. Large quantities of graphite, both imported and

domestic, are used for foundry facings and for this purpose there has never been found a substitute. Silicious material is frequently an advantage because it increases the adhesion of the graphite to the mold. Hence poorer grades may be used. It is estimated that 12,500,000 lb. are used annually for stove polish and foundry facing. Besides these three uses Ceylon graphite is used for lubricants, pencils, certain kinds of paint, for dynamo brushes, for electrotyping and for various minor uses.

Importation by Dealers.

According to Robert Crooks, 138 Front street, New York, the importation of Ceylon graphite by New York dealers during the period from 1899 to 1906 inclusive was as follows:

IMPORTS OF PLUMBAGO AT NEW YORK.
(In barrels of 500 lb.)

Importers.	Lumps.	Chips.	Dust.	Total.
Robert Crooks & Co.	42,086	29,892	48,519	120,497
Paterson Boardman & Co.	37,825	24,395	36,138	98,358
H J. Baker & Bro.	13,702	11,577	20,160	45,439
Pettinos Bros.	923	753	16,879	18,555
McCullough Dalzell Cruc. Co.	16,189	4,293	5,750	26,232
H. W. Peabody & Co.	4,902	4,024	8,785	17,711
H. W. Everett & Co.	715	496	0	1,211
Smith & Schipper	59	137	629	825
Various	1,944	1,866	5,676	9,486
Grand Totals	118,345	77,433	142,536	338,314

These figures do not include direct importation, by crucible-steel manufacturers, which is estimated to be upward of 15,000,000 lb. annually.

GYPSUM.

The industrial depression seriously affected the gypsum industry. The chief producing States in 1908 were New York, Michigan, Iowa, Oklahoma, Kansas, Ohio, Virginia and Texas. A few other States figured to a small extent. Statistics of gypsum for a period of years are given in the accompanying tables:

STATISTICS OF GYPSUM IN THE UNITED STATES.
(In tons of 2240 lb.)

Year.	Production.		Imports.					
			Crude.		Ground or Calcined.		Plaster of Paris.	Total Value of Imports.
	Quantity. (b)	Value. (c)	Quantity.	Value.	Quantity.	Value.	Value.	
1896	195,553	$583,130	180,269	$193,544	3,292	$21,982	$11,722	$ 227,248
1897	268,187	889,177	163,201	178,686	2,664	17,028	16,715	212,429
1898	281,130	864,415	166,066	181,364	2,973	18,501	40,979	240,844
1899	376,840	1,155,581	196,579	220,603	3,265	19,250	58,073	297,926
1900	432,323	1,316,255	209,881	229,878	3,109	19,179	66,473	315,530
1901	568,961	1,577,493	235,204	238,440	3,106	19,627	68,603	326,670
1902	(a) 728,998	2,089,341	305,367	284,942	3,647	23,225	52,533	360,700
1903	(a) 930,093	3,792,943	265,958	301,379	3,526	22,784	54,434	378,597
1904	(a) 840,104	2,784,325	294,238	321,306	3,278	11,276	23,819	356,401
1905	(a) 931,475	(d) 821,967	356,457	402,378	3,471	20,883	22,959	446,220
1906	(a) 1,375,588	3,857,975	390,178	464,724	3,203	22,821	21,297	508,842
1907	(a) 1,564,061	4,942,264	405,278	486,205	1,767	12,825	(e)38,920	537,950
1908	(a) 1,537,346	4,138,560	267,968	314,845	2,296	16,093	(f)26,733	357,671

(a) Statistics of the U. S. Geological Survey. (b) Represents the amount of crude gypsum quarried. (c) Represents the value of the marketed gypsum, including its various finished forms. (d) Value of crude material. (e) Includes $1,392 in gypsum manufactures. (f) All other manufacturers of gypsum, including $8381 in gypsum for pearl hardening for paper makers' use.

PRODUCTION OF CRUDE GYPSUM IN THE UNITED STATES.
(In tons of 2000 lb.)

States.	1905 (a)		1906. (a)		1907. (a)		1908.	
	Tons.	Value. (g)	Tons.	Value. (b)	Tons.	Value. (b)	Tons.	Value.
Cal., Ohio and Va.	147,136	$188,974	(e)202,376	$536,940	(e)283,132	$963,583	(e)272,698	$851,743
Colo. and Wyo.	26,880	26,930	(f)	.....	(f)	.....	(f)	.....
Iowa, Kan. and Tex.	(c)375,230	296,247	(d)639,885	1,546,188	690,315	1,969,266	370,454	846,984
Michigan	299,585	143,597	341,716	753,878	317,261	681,351	327,810	491,928
New York	153,367	151,272	288,631	749,896	324,507	800,225	318,046	760,759
Oklahoma	(d)	.....	(d)	.....	(d)	.....	(h)272,193	599,862
Other States	40,995	14,947	67,977	251,073	136,533	527,839	160,628	587,284
Total	1,043,202	$821,967	1,540,585	$3,837,975	1,751,748	$4,982,264	1,721,829	$4,138,560

(a) Statistics of the U. S. Geological Survey. (b) Value includes that of prepared products. (c) Includes Oklahoma. (d) Included with Iowa, Kansas and Texas. (e) Includes Nevada and Oregon. (f) Included in "Other States." (g) Value as mined. (h) Includes Texas.

According to testimony given at a hearing before the Ways and Means Committee, imported gypsum is whiter in color than the domestic

PRODUCTION OF GYPSUM IN THE PRINCIPAL COUNTRIES. (a)
(In metric tons.)

Year.	Algeria. (b)	Canada.	France. (b)	Germany. (c)		Greece.	India.	United Kingdom.	United States.
				Baden.	Bavaria.				
1896	37,512	187,778	2,051,124	32,801	28,799	120	7,605	196,404	201,205
1897	36,750	217,340	2,004,339	40,702	26,153	51	8,187	184,287	272,493
1898	37,337	196,864	2,115,361	28,037	25,688	83	8,300	199,174	285,644
1899	39,950	221,821	1,807,454	29,419	29,727	81	6,546	215,974	382,891
1900	42,237	228,656	1,774,492	26,381	35,484	129	4,415	211,436	439,265
1901	44,025	266,476	2,385,633	28,183	3,581	671	(d)	204,045	598,529
1902	44,975	301,165	2,185,346	33,150	31,701	Nil	(d)	228,264	740,906
1903	41,550	285,242	1,998,304	29,423	30,894	94	(d)	223,426	945,285
1904	48,375	309,133	1,957,302	26,984	22,766	393	3,937	237,749	853,546
1905	34,743	395,341	1,378,145	28,823	46,247	185	4,877	259,596	982,626
1906	27,950	378,904	1,377,429	25,643	50,763	70	(e)5 000	228,627	1,397,480
1907	26,400	431,286	1,316,567	29,153	48,975	70	(e)5,000	247,537	1,564,061
1908	(d)	346,418	(d)	35,217	(d)	(d)	(e)5,000	193,543	1,694,155

(a) From official reports of the respective countries, except the statistics for the United States. (b) A part of the product is reported as plaster of paris. In converting this to crude gypsum it has been assumed that the loss by calcination is 20 per cent. (c) Prussia is a large producer of gypsum, but there are no complete statistics available. (d) Statistics not yet available. (e) Estimated.

product, and is used in the manufacture of the finer grade of plaster of paris. It costs $1.50 per ton to carry crude gypsum by sea from the mines of Nova Scotia to the seaboard of the United States, which with the duty of 50c. a ton brings the cost of imported gypsum up to $4.05 per ton to the manufacturer. There are nine manufacturers using imported rock, all on the Atlantic coast, and four of them in New York. These manufacturers are unable to sell their finished product for less than double the price asked by the manufacturers who use domestic gypsum. There is no competition between imported and domestic gypsum. Crude gypsum is exported to Canada from the mines in the State of New York. According to other testimony, there is competition between domestic and imported gypsum.

Gypsum in the United States.

Iowa. (By James H. Lees.)—The gypsum industry in 1908 showed in some respects a similarity to that of 1907, while in others there was a variation. The crude gypsum mined was almost the same as the amount reported for 1907 but the amount sold as crude was greater. The amount of finished product sold was also above that sold during 1907, although its total value was less, due to the lower prices prevailing. These averaged about $3.25@3.50 per ton, dropping as low as $2 in December. After the beginning of 1909, however, prices rose somewhat. Taken altogether, 1908 was not so favorable to the industry as was 1907. The outlook for 1909 is very good, at least as regards tonnage output. The statistics of the industry for 1907 (as given by the U. S. Geological Survey) together with those for 1908, are shown in an accompanying table.

The gypsum deposits of Iowa are situated in Webster county, in the vicinity of Fort Dodge, a little to the northwest of the center of the State.

They are considered by Dr. F. A. Wilder to belong to the Permian system, the correlation being based upon lithologic and climatic data. There are probably between 60 and 70 square miles underlaid by gypsum, of which at least 40 square miles will be available for economic purposes. The

STATISTICS OF GYPSUM IN IOWA.

	1907		1908	
	Short Tons.	Value.	Short Tons.	Value.
Crude gypsum mined	251,874	.....	252,270	$316,394
Sold crude—				
To Portland cement mills	14,295	$21,860	15,960	21,409
As land plaster	1,562	4,278	8,985	11,593
Miscellaneous	2,977	2,977	.....	.....
Sold burned for—				
Wall plaster, etc	153,965	656,268	182,996	612,078
Other purposes	9.000	45,000		50,878
Total	181,799	$730,383	207,941	$695,958

valuable part of the deposits varies in thickness from 10 to 30 ft., and is remarkably pure in character, analyzing from 97 to over 99 per cent. fine. The beds are accompanied by red shales and sandstones which, with the gypsum, immediately underlie the mantle of drift, here from 50 to 130 ft. thick. The series is crossed about the middle by the Des Moines river, which has cut through to the underlying beds of the Lower Carboniferous, and has exposed the beds of gypsum along its bluffs.

The first mill for the treatment of gypsum was erected in 1872 and since that time 10 other mills have been built and put in operation. The first four mills were built along the Des Moines river and a small tributary, south of Fort Dodge. Here the gypsum was quarried as it was exposed in the bluffs or uncovered by stripping. Of these four mills only one is now in use, the Blanden, owned by the United States Gypsum Company. As stripping became more difficult drift mining was resorted to in some cases. Later, when the extent of the deposits away from the river became better appreciated, mills were built upon the prairie and the gypsum was won by shaft mining. All the mills now in operation, with the exception of the Blanden mill, on the river, obtain their gypsum by shaft mining.

Of the 11 mills erected in this field one has burned down in recent years, one is entirely dismantled, after operating for only a year or so, and three others have been acquired and closed by the United States Gypsum Company, the last one in 1907. This company operated two mills in 1908, while of the other four which are active one each was owned by the Cardiff Gypsum and Plaster Company, the Plymouth Gypsum Company, the Iowa Hard Plaster Company and the American Independent Gypsum Company. The last two firms commenced work in 1907. The Acme Cement Plaster Company of St. Louis bought the American plant, and

will operate it as one of a series under the same control. The American Cement Company, of Lawrence, Kan., bought 100 acres of land near Fort Dodge and plans to build during 1909 a fireproof mill which will be the equal in capacity of any of those now on the field. The United States Gypsum Company supplies finished stucco to the Sackett Plaster Board Company, which has a factory near one of the mills of the former firm. The output of finished plaster board in1908 amounted to nearly 5,000,000 sq. ft. The finished product is resold to the United States Gypsum Company. The Plymouth Gypsum Company is beginning the manufacture of plaster board, although its work is still in the experimental stage.

In addition to its more common uses gypsum is also used by the Iowa Paint Company as a base for the cheaper grades of paint, such as dry pigments and barn paints. These paints have attained a wide reputation and have given excellent results. In preparing the gypsum for this purpose it is ground in specially designed machinery to such a fineness that 99.7 per cent. will pass through a 100-mesh sieve; this is much finer than gypsum is ground for plaster. It is then mixed crude with proper pigments and liquids for making the desired finished product.

New York. (By D. H. Newland.)—The most notable feature of the gypsum industry in New York during the past year or two has been the great activity manifested in the development of new mines and quarries and the erection of milling and calcining plants. Though the local deposits have long been worked commercially as a source of gypsum for agricultural purposes, strangely enough their utilization for the manufacture of calcined plaster did not attract much attention until recently, many years after that branch of this industry had been established in Michigan and even some of the western States. The explanation for this may be found, doubtless, in the darker color of the crude gypsum in most places, as compared with that from other sources of supply. Yet the gypsum, particularly in the western part of the New York belt, has been shown to yield on calcination a light and very good grade of material. With the growth of the calcined plaster trade, the production has increased in the last decade from an average of 30,000 tons annually to over 300,000 tons. Another important outlet for the product has developed lately among the portland cement mills. The mills in New York and Pennsylvania are largely supplied by the local mines, the gypsum being shipped in both crude and calcined condition.

Among the companies that have engaged in the industry recently are: The Empire Gypsum Company, at Garbutt; Akron Gypsum Company, and American Gypsum Company, at Akron; Niagara Gypsum Company, and Gypsum Products Company, at Oakfield; and Dewitt Mining Company, at Jamesville. The Victor Gypsum Company has explored a property at

Victor, finding two beds of workable size, but has not developed them. The trade in 1908 was somewhat depressed and the production fell off by about 10,000 tons from the total for 1907 which was 323,323 short tons. New York was the leading producer in 1907.

Oklahoma.—According to Peter Hanraty, chief inspector of mines for this State, the production of gypsum in 1908 was 44,362 tons.

(By Charles N. Gould.)—The gypsum area of which the Oklahoma beds form a part is the largest in the United States. The area extends practically uninterruptedly from southern Nebraska across Kansas, Oklahoma and Texas, nearly to the Pecos river. The line of outcrops from southern Nebraska to west central Texas is approximately 600 miles long. The width of the area containing gypsum varies from a few miles to more than 100 miles. Oklahoma is in the center of the region and the most extensive deposits are in this State. The amount of gypsum in Oklahoma is practically inexhaustible. With perhaps two exceptions, every county west of the main line of the Rock Island railroad contains enough gypsum to make plaster to last an indefinite length of time. For convenience all the deposits of the State may be roughly grouped under four general regions, as follows: (1) The Kay county region occupies the central part of Kay county. (2) The main line of gypsum hills extends from Canadian county northwest through Kingfisher, Blaine, Major, Woods, Harper and Woodward counties to the Kansas line. (3) The second line lies along a line parallel to the main range, and from 50 to 75 miles farther southwest and extends from the Keechi hills, in southeastern Caddo county, northwest through Caddo, Washita, Custer and Dewey, and into Woodward and Roger Mills counties. (4) The Greer County region occupies the greater part of western Greer county, as well as the southern part of Beckham and western part of Jackson county.

Classified by counties the approximate amount of gypsum in Oklahoma is as follows: (This classification refers to the old counties of Oklahoma, before the boundaries had been changed by the recent constitutional convention.) Canadian, 50,000,000 tons; Kingfisher, 50,000,000; Blaine, 2,500,000,000; Woods, 14,000,000,000; Woodward, 24,000,000,000; Comanche, 200,000,000; Caddo, 3,000,000,000; Washita, 20,000,000,000; Custer, 6,000,000,000; Dewey, 1,000,000,000; Day, 500,000,000; Roger Mills, 1,000,000,000; Greer, 53,000,000,000; total, 125,800,000,000 tons. Classified by regions the amounts are: Main line of gypsum hills, 40,200,000,000 tons; second line of gypsum hills, 31,600,000,000; Greer county region, 54,000,000,000; total, 125,800,000,000 tons.

There are at present 10 gypsum mills in western Oklahoma, situated as follows: One each at Okarche, Cement, Marlow, Bickford, Ferguson, Southard, El Dorado and Alva, and two at Watonga; there is one mill at

McAlester in the eastern part of the State. The capacity of the various mills ranges from 40 to 100 tons each per day. In the majority of cases the material used is gypsite, or dirt gypsum. Rock gypsum seems to be in bad repute among practical plaster men in Oklahoma. Coal is the fuel used, the greater part of which comes from the McAlester region, 200 to 250 miles distant. The price of coal at the mill ranges from $4 to 6.50 per ton. The market for gypsum products is largely to the east. Much of it goes directly to Kansas City and Memphis. For several years the demand has exceeded the supply.

There are two problems to be solved in connection with the gypsum plaster industry of Oklahoma, namely, the location of available gypsite deposits, and the securing of cheaper fuel. The plaster men believe that the greater part of the gypsite has been located and that the supply will soon be exhausted. There are geological reasons, however, for believing that there are vast undiscovered deposits of gypsite in each of half a dozen of the western counties, and that all that is needed is systematic prospecting. It is my conviction, based on 10 years of careful study of conditions, that at a conservative estimate not 10 per cent. of the available gypsite deposits have yet been found.

The fuel problem is more difficult. It is useless to look for coal anywhere in the gypsum region, and the geological structure precludes the probability of petroleum or natural gas being found in quantity. The nearest coal is 200 miles or more from the gypsum, and under existing conditions the railroads get the freight on both the coal and the finished plaster. A milling in transit rate to the gas fields of eastern Oklahoma has been suggested as the solution of the difficulty. The fact, however, that gypsum is more than one-fifth water of crystallization presents grave difficulties to this plan, even if a suitable rate could be obtained from the railroad. The last resort is to pipe the natural gas to the gypsum. Gas mains are already laid as far as Oklahoma City, and they will probably soon be extended to Elreno, which is not more than 15 miles from the southern end of the gypsum hills. Whether or not a rate can be secured which will enable the plaster men to utilize the gas remains to be seen. To my mind this is the most practicable solution of the problem.

Gypsum in Foreign Countries.

Canada.—Opportunities for developing large gypsum properties are not lacking in Canada. However, gypsum, as a mine product, will stand only a limited cost of transportation. As most of the New Brunswick and Nova Scotia quarries are on, or close to, tide water the cost of handling is reduced to a minimum, but were the gypsum reduced to the plaster of commerce and the arts at or near the quarry, the range of the market would

be extended. The necessary plant is cheap and installation of this sort would increase the value of the quarry immediately.

Nova Scotia is noted as a large producer of gypsum. Low mining costs, due to the occurrence of the deposits in thick beds, their situation within easy access of the sea, and the relative cheapness of labor are factors which, even considering the cost of transportation and the import duty, make it possible for the Canadian product to compete in the United States markets.

Persia.—Large deposits of gypsum are found along the Persian Gulf. The mineral occurs in at least three different formations. Great beds of rock-gypsum are met with at the base of the Fars series. This is the massive variety and consists of fine, almost microscopic, crystals of gypsum mingled with a very small amount of earthy matter. It is often translucent and attains a high measure of purity.

IODINE.

Iodine is manufactured chiefly in Chile, where it is obtained as a by-product of sodium nitrate. Iodine salts occur in all kinds of *caliche,* but especially in the variety known as *chancacado* or brown *caliche.* The iodine salts are dissolved with the sodium salt in the process of leaching, and are extracted from the solution, after the nitrate has crystallized. To effect this, the weak solutions of nitrate, density 70 deg. Tw., are pumped into the iodine house, from the sump, where they have collected. Here they are treated in long rectangular open tanks with dilute sulphuric acid. The hydro-iodates of potassium, etc., are decomposed by the acid, forming potassium sulphate and setting iodine free. The time necessary to effect this reaction in each 1000-gal. tank is two hours, during which time the liquid must be stirred

AVERAGE PRICE OF IODINE PER POUND.

Year.	London.	New York.	Year.	London.	New York.
1898...	10s. 0d.	$2.37	1904.....	10s. 2d.	$2.41
1899...	10 0	2.37	1905.....	11 2½	2.66
1900...	9 6	2.25	1906.....	8 0	1.90
1901...	8 1½	1.93	1907.....	8 0	1.90
1902...	8 0	1.90	1908.....	8 0	1.90
1903...	8 0	1.90			

constantly. The reaction having been accomplished, the liquid is allowed to stand for 24 hours in order to settle the iodine, potassium sulphate, etc., after which the clear solution is syphoned from the tank. The precipitate is taken out, washed immediately with fresh water, and pressed into cakes weighing 80 lb. each. These cakes are stored in a dry place, until a quantity has been collected. The iodine is then extracted by sublimation, which is effected in closed retorts, in batches of 300 to 400 lb., the retorts being charged every 10 or 12 hours. The operation of subliming 3000 lb. of iodine takes about 10 days. After the operation is over and the crystals of iodine in condensing chambers have cooled completely, the product is taken out, placed in wooden, pitch-lined kegs and shipped to England for sale.

The combination of Chilean iodine producers is operating under an agreement with Anthony Gibbs & Son, of London, by virtue of which that firm acts as sole agent for the sale of iodine there, the shipment

being sent to it on consignment. The monthly quantity sold is prorated among the producers, according to the amount of stock they may have on hand at the time of settlement.

EXPORTS OF IODINE.
(In metric tons.)

Year.	Chile.	Germany.	Norway.	Year.	Chile.	Germany.	Japan.	Norway.
1895	144	.....	3	1902	244	24	2	.....
1896	206	26	2	1903	387	29	14	11
1897	243	26	2	1904	461	30	31	9
1898	235	26	5	1905	564	27	26	12
1899	304	26	16	1906	331	44	11	13
1900	318	29	11	1907	290	50	.....	14
1901	385	27	10					

IRON AND STEEL.

By Frederick Hobart.

The record of the iron and steel industry in 1908 is one of general and extreme depression followed by a slow and gradual recovery. The short and sharp curtailment of the closing months of 1907 put the productive capacity of the United States on a basis lower than had been seen for at least eight years. The conditions were different from those of any previous period of depression. The dominating influence of the United States Steel Corporation had been exercised only in prosperous times, and it had now to face the reverses which had been brought about by a period of overtrading and too extended credits. The way in which this was done finds no precedent in former years. Through an adroit use of its influence in uniting with it the larger independent producers the Steel Corporation succeeded in holding the trade to a policy of maintaining prices in the face of a lessened demand. This defiance of the law of supply and demand was maintained throughout the year, only some small reductions being made in a grudging way and at a time when they were of the least effect. The result is apparent to those who study the operations of the year. Recovery, which has come slowly, might, it is believed by many, have been hastened by free concessions which would have encouraged the investment of capital in construction, by offering the advantages of cheaper material. It was in this way that recovery was started in former periods of depression with undoubted success.

Iron Ore.

The production and consumption of iron ore showed a very considerable decrease from the high level of 1907, as might have been expected. The cutting down of operations was sharp and sudden—more so, perhaps, than in any preceding period of depression, since a much larger proportion of the output is under a unified control than ever before. As in the previous year, Lake Superior ores furnished the raw material for 75 to 80 per cent. of the pig-iron production. The estimated output of iron ore is given in an accompanying table. In this statement the increase in stock is estimated, for the reason that it is impossible to obtain any exact figures of ore on hand. The stocks on Lake Erie docks at

IRON ORE MINED AND CONSUMED IN THE UNITED STATES.
(In tons of 2240 lb.)

District.	1902	1903	1904	1905	1906	1907	1908
Lake Superior	27,571,121	24,099,550	21,822,839	34,353,456	38,522,129	42,245,070	26,014,987
Southern States	4,850,000	5,889,000	5,456,000	7,175,000	7,450,000	7,585,000	5,900,000
Other States	2,215,000	2,483,000	2,190,000	3,050,000	3,265,000	3,125,000	1,875,000
Total	34,636,121	32,471,550	29,462,839	44,578,456	49,237,129	52,955,070	33,789,987
Add decrease in stocks	.....	703,169	.....	.....	.....	.....	.....
Add imports	1,165,470	980,440	487,613	845,651	1,060,390	1,329,168	776,898
Total	35,801,591	34,155,159	29,950,452	45,424,107	50,297,519	54,284,238	34,566,885
Increase in stocks	1,214,591	.....	.....	.....	.....	3,750,000	750,000
Deduct exports	88,445	80,611	213,865	208,058	265,240	278,208	309,099
Total consumption	34,499,555	34,074,548	29,736,587	45,216,049	50,032,279	50,256,030	33,507,786

the close of navigation in 1908 showed an increase of 1,055,000 tons over the preceding year; but a considerable part of this was under contract and awaiting transportation. Stocks in furnace yards were smaller than they were a year before. In 1907, owing to the sudden stoppage of many furnaces, there was a large accumulation of ore at furnace stockyards; in 1908 the piling up of stock was at the Lake docks, as a provision made against a possible increase in furnace activity before the opening of navigation in the following spring. Allowing for the probable increase in stocks the consumption of ore per ton of pig iron made in 1908 was approximately 1.98 tons, or a little higher than in previous years.

Lake Superior Iron Ore Shipments.—The records of iron ore shipments have been closely kept and recorded for a series of years, chiefly by the enterprise of the Cleveland *Iron Trade Review*, to which we are indebted for the figures in the accompanying tables. The first of these shows the shipments by ranges.

SHIPMENTS OF IRON ORE FROM LAKE SUPERIOR.
(In tons of 2240 lb.)

Range.	1904	1905	1906	1907	1908
	Tons.	Tons.	Tons.	Tons.	Tons.
Marquette	2,843,703	4,210,522	4,057,187	4,388,073	2,414,632
Menominee	3,074,848	4,495,451	5,109,088	4,964,728	2,679,156
Gogebic	2,398,287	3,705,207	3,643,514	3,637,907	2,699,856
Vermilion	1,283,513	1,677,185	1,792,355	1,685,267	841,544
Mesabi	12,152,008	20,153,699	23,792,553	27,492,949	17,257,350
Baraboo	67,480	111,391	128,742	76,146	122,449

PRODUCTION OF IRON ORE.
(In tons of 2240 lb.)

Year.	Tonnage.	Year.	Tonnage.	Year.	Tonnage.	Year.	Tonnage.	Year.	Tonnage.
1855	1,449	1875	881,166	1895	10,429,037	1904	21,822,839	1908	26,014,987
1860	114,401	1880	1,948,334	1900	19,059,393	1905	34,353,456		
1865	193,758	1885	2,466,642	1902	27,562,566	1906	38,522,239		
1870	859,507	1890	9,003,725	1902	24,289,674	1907	42,266,668		

LAKE SUPERIOR ORE SHIPMENTS TO END OF 1908.

(In tons of 2240 lb.)

Range.	Tons.	Per Cent.	Range.	Tons.	Per Cent.	Range.	Tons.	Per Cent.
Marquette.....	87,647,819	21.5	Gogebic.....	56,732,446	14.0	Mesabi.....	167,527,143	41.2
Menominee....	66,337,670	16.3	Vermillion....	28,017,170	6.8	Baraboo.....	797,868	0.2

RECEIPTS AND STOCKS AT LAKE PORTS.

(In tons of 2240 lb.)

Ports.	Receipts.		Stocks.	
	1907	1908	1907	1908
Toledo.....	1,314,140	680,553	518,645	590,925
Sandusky.....	83,043	.....	44,546	36,079
Huron.....	971,430	213,377	415,730	458,158
Lorain.....	2,621,025	2,286,388	366,271	426,274
Cleveland.....	6,495,998	4,240,816	1,281,335	1,458,392
Fairport.....	2,437,649	1,518,961	523,981	835,821
Ashtabula.....	7,521,859	3,012,064	2,056,820	2,293,531
Conneaut.....	5,875,937	4,798,631	1,090,774	1,296,675
Erie.....	2,294,239	828,602	652,219	730,530
Buffalo.....	5,580,438	2,835,099	435,407	315,148
Total.....	35,195,758	20,414,491	7,385,728	8,441,533

The second shows the total shipments over a series of years, at five year intervals up to 1900, and yearly from that date. The third table shows the total shipments of ore from the region up to the end of 1908; the total to that date having been 407,060,116 tons. The fourth shows the season receipts and shipments at Lake Erie ports—about 80 per cent. of the total—for 1907 and 1908, with the stocks at the close of navigation.

The shipments do not include those from the Michipicoten range in Canada, which were 148,000 tons in 1908, all from the Helen mine.

On the old ranges the Gogebic shows a smaller proportional decrease in 1908 than the others. The Vermillion range had the largest proportional decrease, which was chiefly due to the abandonment of the Chandler mine, which was the chief producer up to two years ago.

The larger productions on the Marquette range in 1908 were 438,379 tons from the Cleveland Cliffs and 278,366 from the Hartford; on the Menominee, 391,620 tons from the Chapin, and 365,361 from the Pewabic. On the Gogebic the leading producers were the Norrie, with 773,243 and the Newport with 579,390 tons. The Vermilion had only one great shipper in 1908, the Pioneer, with 477,506 tons. The Mesabi leaders were the Hull-Rust, 2,026,683; the Burt, 1,460,998; the Fayal, 1,439,879; the Adams, 765,592; the Virginia, 661,329; and the Mahoning, 611,592 tons.

The Oliver Iron Mining Company—the mining department of the Steel Corporation—in 1908 shipped 14,123,957 tons, or 54.3 per cent. of the total. Only a small part of this came from the Hill lands, though a good deal of development and exploration work was done there.

Notwithstanding the dull year, shipments were made from six new mines—one on the Marquette, two on the Menominee, and three on the Mesabi range. Shipments are also reported from four old mines, which had been closed for several years previously—two on the Menominee and one each on the Marquette and the Mesabi.

The 20 per cent. of Lake ore which does not find its way to Lake Erie ports, goes chiefly to Chicago and vicinity for the furnaces of the Illinois Steel Company and others, and lately for the great new furnaces of the Indiana Steel Company at Gary.

Other Ore Production.—The production in the Southern States is the most important after that of the Lake Superior region. It is, as a rule, closely regulated by the activity of the furnaces. The control of the mines by the iron-making companies and the proximity of the mines to the furnaces makes it unnecessary to accumulate stocks, as is done with the Lake ores, which have to be carried from 500 to 1000 miles by water, over routes which are closed by ice for five months of the year. In the East, New York, Pennsylvania and New Jersey are the larger producers, in the order given. In all of them there was a considerable reduction in output. The same condition prevailed in the widely scattered iron-ore regions of the West.

General Conditions.—The depression of 1908 did not stop preparations for an increase in the production of iron ore. Extensive operations were carried on in stripping and preparing for mining on the lands of the United States Steel Corporation on the Mesabi range in Minnesota. In the Southern field all the large companies made arrangements to extend their mining operations, and to improve them by the use of new machinery and appliances. In New York several new mines were opened and some old mines reopened. The universal anticipation seems to be for a greater demand for ore, which must be met.

Limestone Flux.—The production of limestone and dolomite for use as flux in iron blast furnaces in 1908 is estimated at 8,658,558 long tons, an average of 12.7 lb. per ton of pig iron made. Dolomite is used chiefly at the Alabama furnaces, limestone being the usual flux elsewhere.

Pig Iron.

The statistics of pig iron and steel production are carefully kept by the American Iron and Steel Association, and its figures are used throughout this article. The production according to the grades of iron

made; according to the fuel used; by districts and by States; and the approximate consumption are shown in the accompanying tables. The complete statistics of production in 1908 show a reduction of 38 per cent., as compared with 1907, the heaviest proportional loss for many years. The production in 1908 was less than two-thirds of that of 1907, and was only a little above that of 1901; being less than that of

PIG IRON PRODUCTION OF THE UNITED STATES.

(In tons 2240 lb.)

Kind of Iron.	1902	1903	1904	1905	1906	1907	1908
Foundry and forge...	5,176,568	5,281,200	4,358,295	5,837,174	5,709,350	6,397,777	4,307,734
Bessemer pig.....	10,393,168	9,989,908	9,098,659	12,407,116	13,840,518	13,231,620	7,216,976
Basic pig.....	2,038,590	2,040,726	2,483,104	4,105,179	5,018,674	5,375,219	4,010,144
Charcoal.....	.....	504,757	337,529	352,928	433,007	437,397	249,146
Spiegel and ferro.....	212,981	192,661	219,446	289,983	300,500	339,348	152,018
Total.....	17,821,307	18,009,252	16,497,380	22,992,380	25,302,049	25,781,361	15,936,018

PIG IRON PRODUCTION ACCORDING TO THE FUEL USED.

(In tons of 2240 lb.)

Fuel used.	1902	1903	1904	1905	1906	1907	1908
Coke (a).....	16,315,891	15,592,221	14,931,364	20,964,937	23,313,498	23,972,410	15,331,863
Anthracite and coke .	1,096,040	1,911,347	1,228,140	1,674,515	1,535,614	1,335,286	318,741
Anthracite alone.....	19,207				25,072	36,268	36,268
Charcoal.....	378,504	504,757	337,529	352,928	433,007	(b) 437,397	(b)249,146
Charcoal and coke...	11,665	927					
Total.....	17,821,307	18,009,252	16,497,033	22,992,380	25,307,191	25,781,361	15,936,018

(a) Under coke furnaces are included the very few which use raw bituminous coal. It may be assumed that 99 per cent. of this class of iron was made with coke. (b) Includes a small quantity made by the electric furnace.

PRODUCTION OF PIG IRON BY DISTRICTS.

(In tons of 2240 lb.)

	1903	1904	1905	1906	1907	1908
N. England, N. Y. & N. J...	782,350	880,074	1,525,094	1,952,288	2,052,060	1,258,661
Pennsylvania.....	8,211,500	7,644,321	10,579,127	11,247,869	11,348,549	6,987,191
Ohio, Ill., Michigan, Wis. and Minnesota.....	5,508,034	5,077,549	7,260,712	8,226,778	8,467,045	5,050,303
Maryland.....	324,570	293,441	332,096	386,709	411,833	183,502
Southern States.....	2,912,509	2,449,872	2,887,577	3,090,507	3,033,388	2,143,290
West of the Mississippi.....	270,289	151,776	407,774	413,040	468,486	313,071
Total.....	18,009,252	16,497,033	22,992,380	25,307,191	25,781,361	15,936,018

any intervening year. The make was only about one-half of the total capacity of the furnaces.

Steel-making irons were 73.5 per cent. of the total in 1907, and 71.5 in 1908; indicating that the demand for foundry products decreased in a slightly less proportion than that for finished steel. The most

PRODUCTION OF PIG IRON BY STATES.

(In tons of 2240 lb.)

States.	1903	1904	1905	1906	1907	1908
Massachusetts	3,265	3,149	15,987	20,239	19,119	13,794
Connecticut	14,501	8,922				
New York	552,917	605,709	1,198,068	1,552,659	1,659,752	1,019,425
New Jersey	211,667	262,294	311,039	379,390	373,189	225,372
Pennsylvania	8,211,500	7,644,321	10,579,127	11,247,889	11,348,549	6,987,191
Maryland	324,570	293,441	322,096	386,709	411,833	183,502
Virginia	544,034	310,526	510,210	433,525	478,771	320,458
Alabama	1,561,398	1,453,513	1,604,062	1,674,848	1,686,674	1,397,014
N. Car. and Georgia	75,602	70,156	38,699	92,599	55,825	24,345
Texas	11,653	5,530				
West Virginia	199,013	270,945	298,179	304,534	291,066	65,551
Kentucky	102,441	37,106	63,735	98,127	127,946	45,096
Tennessee	418,368	302,096	372,992	426,874	393,106	290,826
Ohio	3,287,434	2,977,929	4,586,110	5,327,133	5,250,687	2,861,325
Illinois	1,692,375	1,655,991	2,034,483	2,156,866	2,457,768	1,691,944
Michigan	244,709	233,225	288,704	369,456	(a) 436,507	(a) 348,096
Wisconsin and Minn	283,536	210,404	351,415	373,323	322,083	148,938
West of Miss. Riv	270,289	151,776	407,774	413,040	468,486	313,071
Total	18,009,252	16,497,033	22,992,390	25,307,191	25,781,361	15,936,018

(a) Includes Indiana.

CONSUMPTION OF PIG IRON IN THE UNITED STATES.

(In tons of 2240 lb.)

	1905	1906	1907	1908
Production	22,992,390	25,307,091	25,781,361	15,936,018
Imports	212,455	379,828	489,440	92,202
Total	23,204,845	25,686,919	26,270,801	16,028,220
Exports	49,221	83,717	73,844	46,696
Approximate consumption	23,155,624	25,603,202	26,196,957	15,981,524

interesting point of comparison was the increased proportion of basic pig made in 1908.

In the table of production by districts, the classification is based chiefly on ores used. It is not possible to make an exact division in the report by States, but the errors will nearly balance. Thus the New York furnaces which use Lake ores offset those in eastern Pennsylvania which use local ores. Maryland production is largely from imported ores. The table shows that approximately 75 per cent. of the pig iron made in 1908 was from furnaces using Lake Superior ores, and 13.5 per cent. in Southern furnaces. Over 88.5 per cent. of the raw material came from those two districts.

Perhaps the only encouraging feature of the statement for 1908 is that the second half of the year showed a gain of 2,100,010 tons, or over 30 per cent., over the first half. This is a sign of improvement in business; but 1908 will stand out—for a long time, let us hope—as a pre-eminently lean year.

STEEL.

The production of steel ingots and castings in the United States in 1908 was less by a little over 40 per cent. than that of 1907, the decline having been rather greater than that in pig iron. The make of steel in 1908 was the smallest reported since 1901, with the sole exception of 1904, when the total was only 130,321 tons less than in 1908. The production and the geographical distribution of that production are shown in the accompanying tables. Aside from the great decrease, the notable feature in the year's production was that the falling off in bessemer output was proportionally much greater than in open-hearth, the result being that for the first time the latter formed the larger part of the steel made. The proportion of converter steel to the total in 1908 was 44 per cent., which compares with 50.2 in 1907, and with 72 per cent. 10 years ago. Moreover, the smallest loss was in the basic open-hearth process, so that acid steel was only 49 per cent. of the total and basic 51; the proportions in 1907 having been 55.7 and 44.3, while in 1898 they were 82.3 and 17.7 per cent. Basic bessemer steel was never made in this country on a large scale, and has not been made at all since 1897. All the bessemer metal reported in the accompanying tables was acid steel. In the second table the small quantity of crucible and special steels, which has never reached 1 per cent. of the total, is disregarded. The preponderance of the open-hearth process may be

PRODUCTION OF STEEL IN THE UNITED STATES.

(In tons of 2240 lb.)

Kinds.	1902	1903	1904	1905	1906	1907	1908
Bessemer	9,138,363	8,577,223	7,859,140	10,941,375	12,275,830	11,667,549	6,116,755
Open-hearth	5,687,729	5,837,789	5,908,166	8,971,376	10,980,413	11,549,088	7,836,729
Crucible and special	121,158	112,238	92,581	111,196	141,893	145,309	69,763
Total tons	14,947,250	14,527,255	13,859,887	20,023,947	23,398,136	23,361,946	14,023,247
Total metric tons	15,186,406	14,756,691	14,081,645	20,344,330	23,772,506	23,735,737	14,247,618

GEOGRAPHICAL DISTRIBUTION OF THE STEEL PRODUCTION IN 1907 AND 1908. (a)

	1907			1908		
	Bessemer.	Open-hearth.	Total.	Bessemer.	Open-hearth.	Total.
Pennsylvania	4,351,841	7,867,705	12,219,546	2,106,382	5,322,229	7,428,611
Ohio	3,636,679	819,642	4,456,321	1,955,446	525,171	2,480,617
Illinois	1,723,073	1,013,251	2,736,324	1,237,747	483,104	1,720,851
Other States	1,955,956	1,848,490	3,804,446	817,180	1,506,225	2,323,405
Total	11,667,549	11,549,088	23,216,637	6,116,755	7,836,729	13,953,484

(a) Disregarding the small quantity of crucible steel. In addition to the States named in the table, Massachusetts, Connecticut, New York, New Jersey, Delaware, Maryland, District of Columbia, Virginia, West Virginia, Kentucky, Michigan, Wisconsin, Minnesota, Missouri, Colorado, and Oregon made steel ingots or castings in 1908 by the standard bessemer process or by modified bessemer processes.

expected to continue permanently, since it is based on the actual substitution of one process for the other, and not on any temporary condition. Thus, for instance, the steel made in the great new works at Gary will all come from open-hearth furnaces; and such furnaces have lately replaced an important group of converters in the Carnegie works at Pittsburg. The bessemer converter has had an important part in industrial history, but it is gradually giving way to its later rival.

STEEL PRODUCTION FOR 11 YEARS.
(In tons of 2240 lb.)

	Acid.								Basic.		Total.
	Converter.		Open-hearth.		Crucible, etc.		Total		Open-hearth.		
	Tons.	P.c.	Tons.	P.c.	Tons.	P.c.	Tons.	P.c.	Tons.	P.c.	Tons.
1898	6,609,017	74.0	660,880	7.4	93,548	1.0	7,363,445	82.4	1,569,412	17.6	8,932,857
1899	7,586,354	71.3	866,890	8.1	106,187	1.0	8,559,431	80.4	2,080,426	19.6	10,639,857
1900	6,684,770	65.6	853,044	8.4	105,424	1.0	7,643,238	75.0	2,545,091	25.0	10,188,329
1901	8,713,302	64.7	1,037,316	7.7	103,984	0.7	9,854,602	73.1	3,618,993	26.9	13,473,595
1902	9,138,363	61.2	1,191,196	8.0	121,158	0.7	10,450,717	69.9	4,496,533	30.1	14,947,250
1903	8,592,829	59.1	1,094,998	7.5	112,238	0.8	9,800,065	67.4	4,734,913	32.6	14,534,978
1904	7,859,140	56.7	801,799	5.8	92,581	0.7	8,755,520	63.2	5,106,367	36.8	13,859,887
1905	10,941,375	54.6	1,155,648	5.8	111,196	0.6	12,208,219	61.0	7,815,728	39.0	20,023,947
1906	12,275,830	52.5	1,321,653	5.6	141,893	0.6	13,739,376	58.7	9,658,760	41.3	23,398,136
1907	11,667,549	49.9	1,269,773	5.5	145,309	0.6	13,082,631	56.0	10,279,315	44.0	23,361,946
1908	6,116,755	43.7	696,304	4.9	69,763	0.5	6,882,634	49.4	7,140,425	50.9	14,023,247

Steel Production of Leading Countries.

The United States and Germany are well established as the two leading steel producers of the world. In both production reached a maximum in 1907, and fell off heavily in 1908. The accompanying table shows the production for three years, divided into acid and basic steel. In the United States the decrease in pig iron last year was 9,845,343 tons, or 38.1 per cent.; in steel, 9,355,007 tons, or 40 per cent. In Germany the changes were much less marked, the decrease in pig iron having been 1,232,249 tons, or 9.4 per cent.; and in steel, 877,253 tons, or 7.8 per cent. These decreases are large for Germany, where the iron and steel production had increased steadily for a series of years; and where the changes in either direction are always much slower and less marked than in this country. The extent of the difference is shown by the statement that the German pig-iron production in 1907 was only 50.6 per cent. of that in the United States; in 1908 it equaled 74.1 per cent. In both countries by far the larger part of the pig iron made is converted into steel, from which most of the finished products are manufactured.

Of the steel made in the United States 49.1 per cent. was acid and 50.9 basic; the proportions in Germany being 6.3 per cent. acid and 93.7

basic. The basic converter, in which more than half the German steel is made, is not used in the United States at all. The German report includes, for the first time, electric steel; the quantity made in the electric furnace having been 19,536 tons.

STEEL PRODUCTION OF PRINCIPAL COUNTRIES.
(In metric tons.)

Kind of Steel.	1906		1907		1908	
	United States	Germany.	United States	Germany.	United States	Germany.
Acid converter	12,081,942	407,688	11,854,230	387,120	6,214,623	374,100
Basic converter	.....	6,772,804	.....	7,212,454	.....	6,510,754
Total converter	12,081,942	7,180,492	11,854,230	7,599,574	6,214,623	6,884,854
Acid open-hearth	1,300,800	230,668	1,290,089	212,620	707,445	224,211
Basic open-hearth	9,497,426	3,534,612	10,443,784	4,039,940	7,254,672	3,969,595
Total open-hearth	10,798,226	3,765,280	11,733,873	4,252,560	8,062,117	4,193,806
Crucible and special	116,633	189,313	147,634	211,498	70,879	(a)107,719
Total	22,996,801	11,135,085	23,735,737	12,063,632	14,247,619	11,186,379
Proportions steel to pig iron	92.3	89.2	90.6	92.5	88.0	94.6

(a) Includes 19,536 tons made in the electric furnace.

MAKE OF ACID AND BASIC STEEL.

	1906		1907		1908	
	Acid. Long Tons.	Basic. Long Tons.	Acid. Long Tons.	Basic. Long Tons.	Acid. Long Tons.	Basic. Long Tons.
United States	13,596,866	9,649,385	12,937,322	10,279,315	6,992,947	7,254,672
Germany	704,677	10,425,054	674,371	11,199,282	706,030	10,480,349
Total	14,301,543	20,074,439	13,611,693	21,478,597	7,698,977	17,735,019

Finished Iron and Steel.—The production of rails is shown in the accompanying table, which includes all classes of rails, rerolled, as well as those made from new steel. The rerolled rails in 1908 were about 71,000 tons. The production of bessemer-steel rails was the smallest reported since 1896. The output of open-hearth steel rails was more than doubled, which is one result of the sharp controversy over bessemer-steel rails for the past two years. The open-hearth rails were made chiefly by the Bethlehem Steel Company and the Tennessee Coal, Iron and Railroad Company.

The table gives also the production of all kinds of rails, according to the weight of the rails per yard. Street and trolley rails are included: The total decrease in 1908 was 47.1 per cent. The greatest proportional decrease was in rails between 45-lb. and 85-lb. section; the smallest was in light rails, under 45-lb. section.

Other finished materials in 1908, so far as reported by the American Iron and Steel Association, were in long tons: Structural shapes, 1,083,181; iron rods, 1,816,949. The total make of rolled iron and steel in all forms in 1908 was 11,828,789 long tons, a decrease of 8,036,033 tons, or 40.4 per cent., from 1907. The production of nails was 10,662,972 kegs of 100 lb. each, a decrease of 1,068,072 kegs, or 10 per cent. The output of tinplates in 1908 was 1,203,075,000 lb., an increase of 49,978,000 lb., or 4.1 per cent.

RAIL PRODUCTION IN THE UNITED STATES.

(In tons of 2240 lb.)

Material.	1907	1908	Section.	1907	1908
Bessemer Steel	3,380,025	1,354,236	Under 45 lb	295,838	183,869
Open-hearth Steel	252,704	567,304	45 to 85 lb	1,569,985	688,198
Iron	925	71	Over 85 lb	1,767,831	1,049,544
Total	3,633,654	1,921,611	Total	3,633,654	1,921,611

CHANGES AND CONSOLIDATIONS.

No considerable changes were made during 1908, and nothing so important as the transfer of the Tennessee Coal, Iron and Railroad Company in 1907 occurred. There was some exchange of properties among the subsidiaries of the Steel Corporation, but these did not change the general management. In December there was an important transfer of stocks of the Colorado Steel and Iron Company, which indicated that the Gould interests had parted with their control of that company. The new interest in the company is reputed to be either the United States Steel Corporation, or parties closely allied with it. These reports have not been confirmed, but are considered probable. Under present conditions it is not likely that the Steel Corporation would openly assume possession of this important Western concern. Its management on a friendly basis would, however, be a valuable asset.

Metallurgical changes during 1908 were comparatively few. The great Gary plant of the Steel Corporation began work, two of its blast furnaces having been started in December. The most important feature of this plant is the use of blast-furnace gases for the production of power on a scale much larger than has heretofore been attempted. Progress was made in the use of the Gayley dry-air blast, and an extended course of experiments proved that its advantages when applied to the bessemer converter were quite as great as with the blast furnace. The metallurgical changes are treated elsewhere.

Imports and Exports.

The values of the imports and exports of iron and steel, including machinery, in the United States for a series of years, with the principal items, are given in an accompanying table.

In 1908 the decrease in exports was in almost all branches, the structural steel and wire trades best maintaining their sales abroad. In imports the greatest decrease was in pig iron and in billets and other half-finished material, considerable quantities of which were brought in during 1907 under the stress of active demand.

At different times in the past we have heard a great deal of the possibilities of foreign trade in iron and steel products. This trade was to act as a balance wheel, maintaining production at an even level when the home demand was slack. Much was said of organizations promoted by the larger steel companies for the purpose of handling and extending sales abroad. In the present period of depression, however, these agencies did not work, exports having decreased to as great an extent as the home production.

The fact is that, in all the talk about foreign trade, no account was taken of the fact that under modern conditions a trade depression is not confined to one important country. It extends to others in greater or less degree, even spreading through the entire commercial world. When we have a large surplus to sell abroad, the demand there has fallen off, almost on parallel lines to our own. Other countries may be willing to buy our products at a low rate, but the purchasing and consuming capacity has been reduced. The balance-wheel theory failed entirely, when it came to be tested, because there was little foreign demand to operate it, and low prices offered could not start the wheel. Germany alone among the important iron-making nations had a small increase—7 per cent.—in its exports. This was gained at the expense of producers by the large syndicates, which sold abroad at such sacrifices that there were strong remonstrances from some of the constituent companies. Moreover, the gain in exports was chiefly in pig iron and in half-finished products. British, Belgian and French iron-masters took advantage of the syndicate policy to put in stocks of pig iron and steel billets at prices lower than their own cost of production. These crude products are finished in their own mills. Whether such sales of material are advantageous to German makers is an open question.

There is no wish to decry the value of foreign trade, which may be a valuable asset of manufacturers when well established. It cannot, however, be established on short notice, nor can it be relied on any more

than home trade in times of depression. And it may—as in the case of Germany—be acquired at a cost beyond its value.

IRON AND STEEL EXPORTS AND IMPORTS, UNITED STATES.
TOTAL VALUE. (a)

	1904	1905	1906	1907	1908
Exports	$128,455,613	$142,928,513	$172,555,588	$197,036,781	$151,118,114
Imports	21,621,970	26,392,728	34,827,132	38,789,992	19,957,261
Excess, exports	$106,833,643	$116,535,785	$137,728,456	$158,246,789	$131,155,853

(a) Including machinery.

UNITED STATES EXPORTS OF IRON AND STEEL.
(In tons of 2240 lb.)

	1901	1902	1903	1904	1905	1906	1907	1908
Pig iron	81,211	27,487	20,379	49,025	49,221	83,317	73,844	46,696
Billets, blooms, etc.	28,616	2,409	5,445	314,324	237,638	192,616	79,991	112,177
Bars	45,105	31,549	37,182	55,472	51,870	88,102	98,654	46,103
Rails	318,055	67,455	30,656	414,845	295,023	328,036	338,906	196,510
Sheets and plates	30,832	18,300	18,093	55,204	75,034	110,654	122,696	104,993
Structural steel	54,005	53,859	30,641	55,514	83,193	112,555	138,442	116,878
Wire	88,238	97,843	108,521	118,581	142,801	174,014	161,228	136,167
Wire-rods	8,165	24,613	22,360	20,073	6,514	5,896	10,653	7,412
Nails and spikes	29,881	35,994	42,664	45,112	47,756	59,491	56,826	38,906
Pipe and fitting							176,832	114,371

UNITED STATES IMPORTS OF IRON AND STEEL.
(In tons of 2240 lb.)

	1901	1902	1903	1904	1905	1906	1907	1908
Pig iron	62,930	625,383	599,574	79,500	212,465	379,828	489,440	92,202
Billets, blooms, etc.	28,164	289,318	261,570	10,807	14,637	21,337	19,334	12,112
Scrap iron and steel	20,130	109,510	82,921	13,461	23,731	19,091	27,687	5,090
Bars	20,792	28,844	43,393	20,905	37,298	35,793	39,746	19,672
Rails	1,905	63,522	95,555	37,776	17,278	4,943	3,752	1,719
Wire-rods	16,804	21,382	20,836	16,206	17,616	17,999	17,076	11,208
Tinplates	77,395	60,115	47,360	71,304	65,740	56,983	57,773	58,320

THE UNITED STATES STEEL CORPORATION.

The operations of the United States Steel Corporation, which controls about 60 per cent. of the finished steel production of the country, are summed up in the accompanying tables, which give the balance sheet as of Dec. 31, 1908; a summarized income account; and the production statement for two years. In this report no attempt is made to cover up the depression in business which marked 1908, or to disguise the losses of business in comparison with previous years. How great those losses were can be illustrated by a few figures. Thus the gross sales, while approaching very nearly $500,000,000 in amount, were less than those of 1907 by $274,706,928, or 36.3 per cent. The net earnings, after deducting all working and general expenses, depre-

ciation and ordinary renewals, decreased $69,116,963, or 42.9 per cent.; while the surplus over all charges—the final net profit—was diminished by $58,836,850, or 56.3 per cent. The higher proportion of loss in the final net surplus is explained by the fact that the interest and fixed charges are practically the same in a lean business year as in a good one. Perhaps the remarkable point is that with a loss of over one-third of the gross earnings the surplus for the year should have been sufficient to pay the usual dividends, without drawing upon the company's accumulated profits.

The loss in earnings was due entirely to smaller business, since the report states that there was little change in the prices realized. What the reduction was is shown by the table of production which notes decreases of 31.7 per cent. in the make of pig iron; 41.2 in steel ingots, and 41.3 in finished iron and steel. The tables are not carried far enough to make statements of average returns and costs practicable. A rough determination may be made, however, by taking the quantity of finished iron and steel products, which are the forms in which by far the greater part of the Corporation's output reaches the consumer. On this basis the gross earnings of 1907 show an average of $71.65 per ton, and the net earnings $15.24; the figures for 1908 being $77.71 and $14.80 per ton. These averages are, of course, too high, as they disregard the sales of steel billets and other half-finished forms, and of by-products, like copperas, cement, etc.; but they will serve some purposes of comparison.

As already noted, the final net earnings were sufficient to continue the payment of dividends at an unchanged rate; but the Corporation suspended for the year the appropriations for new construction—i.e., for actual additions of property, as distinguished from renewals and the making good of depreciation. Such appropriations in 1906 and 1907 amounted to $54,000,000 in each year. Since its organization the Corporation has spent for such additions a little over $348,000,000, the effect of which has been to double its pig-iron production, to increase its capacity for making steel about 80 per cent. and for finished forms 67 per cent. This has been a wise policy; indeed, a necessary one, if it was to maintain the position in the trade. After all, the additions did not do more than keep pace with the general growth in capacity for making iron and steel. In 1908 the proportion of the total output of the United States furnished by the Steel Corporation was very nearly the same as in 1907.

The report deals fully with the financial side of the year's operations, but has little to say of technical matters, or technical progress. One statement of interest, however, is that while the open-hearth steel

capacity was increased during 1908 by 3,052,000 tons yearly, the bessemer-steel capacity was decreased by 746,000 tons, owing to the substitution of open-hearth furnaces for converters.

Another proof of the diminished business is found in the fact that the average of 210,080 employees in 1907 was reduced by 44,969 to 165,211 men in 1908. The Corporation continued its policy of placing stock with its employees, and at the end of 1908 had 22,960 who were owners of preferred stock.

Very little is said in this latest report of iron-ore holdings, the subject being apparently avoided, notwithstanding—or perhaps because—of the recent active discussion of the question of iron-ore supplies and their ownership. The only reference is the statement that prospecting and preparation for increased production from the Mesabi properties were actively carried on, though the output last year was less than for three years preceding.

PRODUCTION OF THE U. S. STEEL CORPORATION.
Including Tennessee Company in 1907 and 1908

	1904	1905	1906	1907	1908
Iron Ore Mined—	*Tons.*	*Tons.*	*Tons.*	*Tons.*	*Tons.*
From Marquette Range	934,512	1,359,722	1,442,290	1,170,496	830,067
From Menominee Range	1,186,104	1,871,979	1,874,680	1,625,358	1,021,598
From Gogebic Range	1,271,831	1,671,747	1,465,375	1,425,457	1,078,025
From Vermilion Range	1,056,430	1,578,626	1,794,186	1,724,217	927,206
From Mesabi Range	6,054,210	12,004,482	14,068,617	16,458,273	11,272,397
In Southern Region	.....	.....	.....	1,576,757	1,533,402
Total	10,503,087	18,486,556	20,645,148	23,980,558	16,662,715
Coke Manufactured (a)	8,652,293	12,242,909	13,295,075	13,544,764	8,169,931
Coal Mined, not used in making coke	1,998,000	2,204,950	1,912,144	3,550,510	3,008,810
Limestone Quarried	1,393,149	1,967,355	2,227,436	3,201,222	2,186,007
Blast Furnace Products—					
Pig iron	7,210,248	9,940,799	11,058,526	11,234,447	6,810,831
Spiegel	100,025	158,071	150,044	130,554	74,716
Ferro-Manganese and Silicon	59,148	73,278	58,807	57,794	48,861
Total	7,369,421	10,172,148	11,267,377	11,422,795	6,934,408
Steel Ingot Production—					
Bessemer Ingots	5,427,979	7,379,188	8,072,655	7,556,460	4,055,275
Open-Hearth Ingots	2,978,399	4,616,051	5,438,494	5,786,532	3,783,438
Total	8,406,378	11,995,239	13,511,149	13,342,992	7,838,713
Rolled and Other Finished Products for Sale—					
Steel Rails	1,242,646	1,727,055	1,982,042	1,879,985	1,050,389
Blooms, Billets, Slabs, Tinplate Bars	932,029	1,253,682	1,096,727	761,195	551,106
Plates	404,422	780,717	836,399	894,364	312,470
Heavy Structural Shapes	313,779	484,048	620,823	587,954	313,733
Merchant Steel, Skelp, Hoops, Bands	577,384	962,782	1,240,548	1,338,833	577,591
Tubing and Pipe	710,765	911,346	1,025,913	1,174,629	654,428
Rods	84,934	84,049	111,488	126,095	93,406
Wire and Products of Wire	1,226,610	1,283,943	1,399,717	1,481,226	1,275,785
Sheets—Black, Galvanized and Tinplate	757,482	924,439	1,112,542	1,070,752	770,321
Finished Structural Work	357,488	404,732	643,622	719,887	403,832
Angle and Splice Bars and Joints	72,470	150,265	176,730	195,157	84,669
Spikes, Bolts, Nuts and Rivets	46,003	61,496	70,233	67,991	40,252
Axles	62,981	149,596	181,913	189,006	24,057
Sundry Iron and Steel Products	25,787	28,236	79,736	77,463	54,893
Total	6,792,780	9,226,386	10,578,433	10,564,537	6,206,932
Spelter	29,963	29,781	28,884	31,454	28,057
Copperas (Sulphate of Iron)	15,805	20,040	21,933	24,540	26,411
	Bbl.	*Bbl.*	*Bbl.*	*Bbl.*	*Bbl.*
Universal Portland Cement	539,951	1,735,343	2,076,000	2,129,700	4,535,300

(a) 1907 includes 828,751, and 1908, 578,869 tons made in by-product ovens.

U. S. STEEL CORPORATION: GENERAL BALANCE SHEET.

Liabilities.		Assets.	
Capital stock (a)	$869,223,912	Property accounts	$1,474,142,655
Bonds and debentures	593,231,209	Deferred charges	6,306,666
Mortgages and purchase debts	4,161,981	Outside investments	3,083,000
Current liabilities	43,031,854	Sinking fund assets	23,263,035
Sinking and reserve funds	64,879,380	Current assets	189,673,223
Bond sinking funds	38,074,012	Cash	49,548,953
Undivided surplus	133,415,214		
Total	$1,746,017,532	Total	$1,746,017,532

(a) Stock includes $360,281,100 preferred; $508,302,500 common stock; $640,302 stocks of subsidiary companies not owned by the Steel Corporation.

U. S. STEEL CORPORATION: SUMMARY OF INCOME ACCOUNT.

	1906	1907	1908		1906	1907	1908
Gross sales	$696,756,926	$757,014,768	$482,307,840	Interest, etc(a)	$29,742,414	$27,719,744	$29,153,815
Interest, etc	9,159,864	9,748,951	5,786,885	Depreciation	28,753,272	28,679,366	16,965,182
Total receipts	$705,916,790	$766,763,719	$488,094,725	Total	$58,495,686	$56,399,110	$46,118,997
Operating exp	$517,083,955	$564,166,767	$367,735,102	Surplus	$98,128,587	$104,565,564	$45,728,714
General expenses	22,907,679	25,395,379	21,001,037	Dividends	35,385,727	35,385,727	35,385,727
Sub. Co. accts	9,300,883	16,236,899	7,510,875	New Construct	50,000,000	54,000,000	.....
Total expense	$549,292,517	$605,799,045	$396,247,014	Total expense	$85,385,727	$89,385,727	$35,385,727
Net earnings	$156,624,273	$160,964,674	$91,847,711	Undivided pfts	$12,742,860	$15,179,837	$10,342,987

(a) Interest charges include sinking funds.

THE IRON AND STEEL MARKETS.

The markets for iron and steel at the beginning of 1908 were in a condition of deep depression. The shock of the financial depression which began in October, 1907, was still too recent for recovery to begin, and it was not for some months that the trade was on its feet, so to speak; and not until nearly the end of the year was there a pronounced tendency to better conditions. The most prominent feature of 1908 was the control over prices exercised by the large producers through the influence of the supervising committees instituted in December, 1907. The dominating influence was, of course, that of the Steel Corporation.

The only really open markets for the greater part of 1908 were those for pig iron, for iron bars and for a few minor products. Pig iron was badly demoralized for a considerable part of the year, and its recovery was gradual and irregular. The business was limited, notwithstanding low prices; but this was due to the fact that so large a proportion of the pig iron made enters into the production of steel, the quotations for which were not materially lowered. Bar iron was an open market, owing to the large independent interests in that branch, and the lowering of prices on that product did stimulate orders and force a reduction in the quotation for steel bars to meet the competition.

The only changes in price made were some small ones in January, and a general "readjustment" in June, which involved reductions of 10 to 15 per cent. in the leading articles of finished steel. These did not have the effect of stimulating trade in any marked degree; and thereafter no material changes were made. It was pointed out by many consumers that there was a wider spread between quotations of pig iron and finished material than had often existed; but these representations had no effect.

The railroads were light customers throughout the year, and their purchases of rails were smaller than for a number of years. In part this was the result of inability to purchase, owing to decreased earnings and difficulties in finance; in part of the still unsettled dispute over specifications; and in part over the question of price. Toward the end of the year some good orders were placed for locomotives and other equipment, mainly for steel cars. Some good orders for bridge work also came forward. The rail question, however, was still far from a real settlement; and the rail mills entered 1909 with a lighter business on their books than has been the case for several years. In light rails for mining and industrial purposes, on which the price restrictions were removed, owing to the competition of the rerolling mills, there was a good business.

Pittsburg. (By S. F. Luty.)—Mill operations in January were unsatisfactory except in the tinplate and wire branches. At the opening of the month the American Sheet and Tin Plate Company had but 52 tin mills on the active list, but before the close 154 mills were going. The American Steel and Wire Company operated its plants at an average of around 70 per cent. throughout the year, starting at over 60 per cent. and closing the year with nearly 90 per cent. of its capacity active. After the latter part of the first quarter all the large concerns reported a gradual increase in production, each month being a trifle better than the previous one. New business during 1908 was chiefly for small tonnages, showing that consumers bought only for immediate requirements. Agricultural-implement makers who had placed contracts for the year ending June 30 took all the material ordered and were buyers of large tonnages through the year. The farmers had a profitable season and did not hesitate to place orders for new implements. They were also heavy buyers of fence wire. Rail buying was disappointing. The tonnage placed in 1908 did not amount to much more than 1,500,000 tons against about 3,600,000 tons in 1907 and almost 4,000,000 tons in 1906. Difference with the railmakers over specifications and also a lack of funds are assigned as reasons for failure to buy more liberally. Early in the year the Pennsylvania Rail-

road reserved 147,000 tons and on Feb. 6 announced that it had placed 55,000 tons. The railmakers, however, rejected this order on account of the specifications. This road got through the year with less than 10,000 tons of rail which were purchased from the Cambria Steel Company and used for experimental purposes. In December this road entered the market with an order for 135,000 tons for 1909 delivery. Other lines placed orders for smaller tonnages. Late in the year railroads were large buyers of track material and bridge work; also rolling

AVERAGE PRICES AT PITTSBURG, 1908.

Month.	Pig Iron.			Ferro-Mang.	Steel.					Nails.	
	Bessemer.	No. 2 Foundry.	Gray Forge.		Bessemer Billets.	Rails.	Black Sheets No. 28.	Tank Plate.	Steel Bars.	Wire per Keg.	Cut per Keg.
	$	$	$	$	$	$	c.	c.	c.	$	$
January	19.15	17.90	16.90	49.00	28.00	28.00	2.50	1.70	1.60	2.05	2.00
February	18.15	16.90	15.90	47.00	28.00	28.00	2.50	1.70	1.60	2.05	2.00
March	18.15	16.90	15.90	46.00	28.00	28.00	2.50	1.70	1.60	2.05	1.90
April	17.65	16.40	15.40	45.00	28.00	28.00	2.50	1.70	1.60	2.05	1.90
May	17.15	15.90	14.90	46.00	28.00	28.00	2.50	1.70	1.60	2.05	1.85
June	17.15	15.90	14.90	46.50	25.00	28.00	2.50	1.60	1.40	1.95	1.75
July	16.90	15.90	14.90	46.50	25.00	28.00	2.50	1.60	1.40	1.95	1.75
August	16.25	15.65	14.65	46.00	25.00	28.00	2.50	1.60	1.40	1.95	1.80
September	16.00	15.40	14.40	45.50	25.00	28.00	2.50	1.60	1.40	1.95	1.80
October	15.65	15.40	14.40	46.00	25.00	28.00	2.50	1.60	1.40	1.95	1.75
November	16.90	16.40	15.40	46.50	25.00	28.00	2.50	1.60	1.40	1.95	1.75
December	17.65	16.65	15.65	46.50	25.00	28.00	2.50	1.60	1.40	1.95	1.75
Year	17.23	16.28	15.28	46.38	26.25	28.00	2.50	1.64	1.48	1.99	1.83

stock and other equipment. Structural steel contracts were confined to small tonnages chiefly, but there were a number of from 2000 to 3000 tons. The largest called for 13,500 tons for the Beaver bridge of the Pittsburg & Lake Erie Railroad and 12,000 for the new Oliver building in Pittsburg.

Work on the steel plant and blast furnaces being built at Aliquippa, a few miles below Pittsburg, on the Ohio river, by the Jones & Laughlin Steel Company, stopped in 1907, was not resumed. The new open-hearth steel department at the Ohio works, Youngstown, of the Carnegie Steel Company, was finished early in 1908 and was put in operation. The Youngstown Sheet and Tube Company's two new blast furnaces were completed and put in blast in November. The Pittsburg Steel Company, the large independent wire interest, completed eight open-hearth furnaces at its plant at Monessen, about 40 miles from Pittsburg, in June and four were started in the following month. This company's contract with the Carnegie Steel Company for billets expired on June 30.

The year opened with a weak pig-iron market and most of the merchant furnaces in the Valleys were closed on account of dull conditions. The highest price of the year was in January and the lowest in October. As soon as the result of the election was known producers advanced prices $1 and more a ton, and thereby checked what looked like a good buying movement. This did not deter producers from maintaining firm prices the rest of the year. Average prices for all sales by months are shown in the accompanying table. The official average of bessemer iron in which only sales of 1000 tons or more are considered was as follows, Pittsburg delivery: January, $19; February, $17.90; March, $17.86; April, $17.49; May, $16.92½; June, $16.90; July, $16.73; August, $16.11; September, $15.90; October, $15.48; November, $16.74; December, $17.40.

Merchant furnace interests demanded a reduction in ore prices, but at two meetings of ore producers, one on Jan. 29 and the other on Feb. 6 the appeal was ignored and old prices reaffirmed. The lake ore shipping season did not open until June 1 and a week later producers cut the price 50c. a ton, the season's new prices being as follows: Mesabi non-bessemer, $3.50; Old Range non-bessemer, $3.70; Mesabi bessemer, $4.25; Old Range bessemer, $4.50. These prices were guaranteed until May 1, 1909. The lake ore shipments for the short season of 1908 amounted to 26,014,987 tons.

When the Pittsburg Steel Company started four of its new open-hearth furnaces in July it bought 8000 tons of basic iron from the Midland furnace at a low price. Other small lots were purchased until Oct. 19 when it entered into a five-year contract with the Penn Iron and Coal Company, operating a furnace at Canal Dover, O., for 6000 tons a month, or a total of 360,000 tons. The price was on a sliding basis. The Standard Sanitary Manufacturing Company and the Westinghouse interests were large purchasers of pig-iron during the year.

The foundry pig-iron producers held several meetings early in the year for the purpose of devising means for preventing a decline in prices but all were unsuccessful. It was found to be impossible to maintain prices and late in April the pig-iron market was thrown wide open. While the movement of the United States Steel Corporation and other large steel interests for the maintenance of prices was not an entire success it had the effect of preventing general demoralization. The first readjustment of prices occurred on Jan. 6 when tinplate was cut from $3.90 to $3.70 a box; black sheets, $2 a ton from 2.60c. to 2.50c. for No. 28 gage; galvanized sheets, $4 a ton, from 3.75c. to 3.55c. for 28 gage; blue annealed sheets, 5c. per 100 lb., making new prices as follows: Nos. 10 and heavier, $1.80; Nos. 11 and 12, $1.85;

Nos. 13 and 14, $1.90; painted corrugated roofing 10c. a square, from $1.85 to $1.75; galvanized corrugated roofing, 15c. a square, from $3.25 to $3.10. It was explained that these cuts were not made to induce buying, but were on account of the reduced cost of raw material.

A meeting of the principal bar-iron producers was held in Pittsburg on Jan. 7 at which prices for Pittsburg and Eastern shipment were reduced $1 a ton to 1.55c. Pittsburg, and for Western shipment $4 a ton to 1.40c. Pittsburg. A practical reduction in the price of billets was made in March when the mills agreed to divide the freight of finishing mills located outside of Pittsburg when the rate was from $1@3 a ton.

At a meeting of steel interests held in New York on June 9 a readjustment of prices was decided upon. Billets were cut $3 a ton, from $28 to $25; sheet-bars and tinplate-bars, $2 a ton, from $29.50 to $27.50; structural material, a general cut of $2 a ton; shapes, from 1.70 to 1.60c.; plates, $2 a ton, from 1.70 to 1.60c.; wire products, 10c. per 100 lb.; wire nails, 10c. a keg, from $2.05@1.95; merchant-steel pipe, two points, or $4 a ton. Later wrought-iron pipe prices were reduced one point, or $2 a ton, by manufacturers who did not attend the steel meeting.

On June 1 the price of steel bars was cut $4 a ton, from 1.60c. which had been the price for two years, to 1.40c. On June 11 bar-iron prices were reduced to 1.40c. Pittsburg, and to 1.35c. for Western shipment. The season's cotton-tie price was fixed in April at 85c. a bundle, the rate that ruled in 1905 and 1906, and a cut of 10c. a bundle from the 1907 price. Over 2,000,000 bundles, or 40,000 tons, were bought soon after the price was arranged. On Jan. 22 the price of light rails was reduced $2 a ton but the new price was immediately shaded. Rerolling mills cut the price of the large producers from $2 to $3 a ton throughout the year. The established prices on finished material were not strictly adhered to at all times. In the summer plates were cut about $2 a ton. Sheets also were shaded in some extremely dull periods and in September and October tinplate was shaded from 5 to 10c. a box. In the last two months of the year prices in all lines were firmly held.

Although dull conditions prevailed throughout the year there were no particularly severe wage reductions. Outside of the mills governed by the annual wage scale of the Amalgamated Association of Iron, Steel and Tin Workers there was no concerted action in the matter of wages. In a few foundries and machine shops wages were readjusted and the reductions were accepted by the workmen without protest. A largely attended meeting of merchant blast-furnace interests was held at

Youngstown on Jan. 21 for the purpose of considering the advisability of reducing the wages of blast-furnace workers 20 per cent. After a thorough discussion it was decided to make no change as a cut might result in a further depression in pig-iron prices.

Alabama. (By L. W. Friedman.)—Alabama iron producers suffered during 1908 from the general depression of trade, and from the strike of the coal miners. The quotations for pig iron in Alabama ranged between $11 and $13 per ton, on No. 2 foundry basis. Some large-sized sales of iron were consummated during the low-price period, but care was taken by the producers in this section that this business should not interfere with general trade when the conditions improved. Alabama witnessed considerable improvements in blast furnaces. But one concern fell by the wayside, the Southern Steel Company, while some of the progressive companies made deals whereby they took in other properties. The Birmingham Coal and Iron Company acquired the properties of the Tecumseh Iron Company (an ore-mining company), and completed and put in blast its new furnace at Vanderbilt, near Birmingham. The Tennessee Coal, Iron and Railroad Company made extensive improvements about blast furnaces and started work on big improvements in the steel plant. The Woodstock Iron Company put in shape one iron-maker and repaired another, besides putting in operation mining and other properties. The Sloss-Sheffield Steel and Iron Company practically rebuilt one furnace and had a force of men improving another. Other concerns made improvements despite the depression. The depression, the placing of improvements and other reasons, affected steel production. The big plant of the Tennessee Coal, Iron and Railroad Company at Ensley was the only institution of the kind in the State that worked throughout the year.

Chicago. (By E. Morison.)—Except for two or three large transactions made at low points in the market, the history of the pig-iron market in Chicago for 1908 is a record of small buying for needs not more than three months away. The year began in depression. Prices fell gradually as it became apparent that no general revival of business was at hand. Sales of carload lots to 200 or 300 tons became the rule, and May saw iron decline from $18@18.25, the January price for Northern No. 2, to $17@17.50. Southern from $13@13.50 Birmingham ($17.35@17.85 Chicago), in January had dropped to $12 Birmingham, in the open local market. One large sale of about 50,000 tons was made in February at $11 Birmingham, and this transaction weakened the market greatly, the average melter concluding from it that general reduction in prices would follow.

The Spring depression lasted into June, when another sale of nearly as much tonnage was made of Southern at $11. There was a wave of buying for a brief time, and the market strengthened slightly. Southern was prevented from sagging below $11.50 in the open market and continued to rise slowly until December, when it brought $13@13.50 ($17.35@17.85 Chicago). Northern continued depressed, and in September, October and November sold at $16.50@17, or for less than Southern. This exception to the ordinary relations of the two kinds was largely due to the selling of considerable iron from Ohio furnaces throughout the months named at $16.50@17, meeting the needs of users of Northern to the disadvantage of local furnaces.

It was the hope of many in the iron business that the election in November would bring a decided change in the condition of the market.

IRON AND STEEL PRICES AT CHICAGO.

Material.	1906		1907		1908	
	Highest.	Lowest.	Highest.	Lowest.	Highest.	Lowest.
Lake Superior Charcoal	$26.50	$19.00	$28.00	$25.00	$24.00	$19.50
Northern No. 2 Foundry	27.00	18.00	27.00	19.00	18.50	16.50
Southern No. 2 Foundry	26.90	16.90	27.35	19.35	17.85	15.35
Connellsville Coke	6.90	5.40	5.15	5.50	.....	.....
Bar Iron	1.85c	1.665c	1.865c	1.75c	1.65c	1.50c
Structural Material (a)	1.865c	1.865c	1.88c	1.865c	1.88c	1.78c

(a) Beams and channels, 3 in. to 15 in., and angles 3 in. to 6 in. x ¼ in. or heavier.

Immediately preceding the election the market was very weak, but there was no decided change from the election or from any perceptible cause. Very slowly the trend upward became perceptible, however. The last half of the month of December saw a lull in business nearly always experienced in that month, but peculiarly disappointing because of the hopes raised by the gradual strengthening previously of the market.

There was little buying for more than three months ahead of needs at any time during the year and contract-making became restricted to the needs for the current year just before the election. The expansion into more general buying for the future in the last two months of the year rather than any increase in the immediate needs of buyers accounted for the general betterment of the market.

For iron and steel products the market was in general quiet, despite reductions in June from 1.65 to 1.50c. on iron bars and from 1.88 to 1.78c. on structural material. In summer the Western railroads began buying on a large scale in comparison with their purchases in the first half of the year, and they continued to make purchases throughout the

last six months. Building projects became important, and the end of the year saw a large tonnage needed for bridge and building operations in sight.

Seaboard.—For the greater part of 1908 foundrymen in Eastern territory bought on the hand-to-mouth plan, taking from time to time only such small lots as served their immediate needs. For these orders there was sharp competition among such furnaces as were kept in blast. It was a time when it was almost impossible to follow prices; cuts of 25 and 50c. were made from the nominal quotations, and these were kept quiet as far as possible. For several months there was considerable accumulation of unsold pig iron, and a natural disposition to try and work this off at almost any price was observed. About September a gradual stiffening began. The cast-iron pipe makers were taking a good deal of iron, and other foundries were increasing their requirements. Moreover, there was some rather active inquiry for basic pig, indicating some revival of business among Eastern steel makers. In October this business quieted down and the markets were dull until near the middle of November. Another revival set in, and by the beginning of December unsold iron accumulations were pretty well worked off. It was noticeable, however, that consumers continued the policy of buying for short deliveries only, and there was much reluctance about placing contracts for 1909. Very little business was done beyond the first quarter..

Through all the period of dullness Southern iron had the call in seaboard territory. Sales were larger, and the Southern furnaces kept in more active operation than the Northern stacks during the entire year. At the close of the year the market showed more activity and greater firmness, the prices of $13@13.50 Birmingham for Southern No. 2 foundry, and about $17 seaboard for Northern No. 2 being fairly well established.

In finished material—with the exception of structural steel—the markets were dull throughout the year. Early in the year there was an expectation of lower prices, but these did not come, except in a very limited degree, and there was little change in the volume of business. As with pig iron, buyers took only what they required from time to time, and would not contract far ahead, except upon terms which sellers were unable to grant under the close restriction and supervision from which the markets suffered. Some revival set in in November, and there was some placing of contracts for the first quarter of 1909, but little beyond that time. There was also at this time an increase in demand for nails and other supplies used in small building, indicating a more hopeful condition as to that class of construction.

The structural market in the East showed some exceptions to the general rules. Owing to the withdrawal of money from the speculative markets, there was a large supply ready to invest in building projects, if conditions as to price should prove to be favorable. It was well understood that the Steel Corporation dominated the price situation, and its action was closely watched. The small and grudging reductions in structural prices made in June were a sharp disappointment. They were not enough to stimulate trade, but sufficient to induce hopes of further reductions. The immediate effect was to reduce the business that was ready to be offered.

A little later, however, structural trade began to revive, and more orders were placed. As the year went on the conditions improved, though nominally no further reductions in prices were made. The structural business in the larger cities of the East is done mainly with the large fabricating companies, which contract for the whole iron work of a building. Orders to the mills are sent by those companies. They began gradually to take orders and contracts for finished buildings at prices which, when closely analyzed, indicated a lower price for the steel than the nominal quotations. Though no acknowledgment of such reductions was made, there is no doubt that structural steel was supplied by many mills at cut prices. The exact extent of these cuts has been a secret guarded with unusual care, and it is impossible to give facts or figures. At any rate business improved, and during the later months of the year building and bridge contracts of large aggregate quantity were placed.

Range of Prices for Fifty Years.

The accompanying table, compiled from the records of the American Iron and Steel Association, gives the average yearly prices of a number of leading articles of iron and steel for a period of 50 years. The price of foundry iron given is at Philadelphia; of bessemer pig at Pittsburg. Bar iron prices up to and including 1881 are at Philadelphia, the early manufacture of bars having been centered in eastern Pennsylvania and New Jersey. Other prices are at mill, or at central basing points. Iron rails ceased to be of commercial importance about 1880, steel rails supplanting them from that date. Wire nails first appeared on the market in commercial quantities about 1887; since that date they have been gradually supplanting cut nails, the latter now constituting not over 15 per cent. of the total production. In making comparisons, it must be remembered that from 1861 onward the prices were in currency which was depreciated in value from the gold standard. This depreciation reached its greatest point in 1864-65,

and from that time on gradually decreased until it finally disappeared in 1879, with the full resumption of specie payments.

IRON AND STEEL PRICES FOR FIFTY YEARS.
In Tons of 2240 lb., except nails which are in kegs of 100 lb.

Year.	Pig Iron.		Bar Iron, best.	Rails.		Nails.		Year.	Pig Iron.		Bar Iron, best.	Rails.		Nails.	
	No. 1 F'dry.	Bessemer.		Iron.	Steel.	Cut.	Wire.		No. 1 F'dry	Bessemer.		Iron.	Steel.	Cut.	Wire.
	$	$	$	$	$	$	$		$	$	$	$	$	$	$
1859	23.33	.....	60.00	49.38	.....	3.86	.....	1884	19.81	.....	38.45	.....	30.75	2.39	.....
1860	22.70	.....	58.75	48.00	.....	3.13	.....	1885	17.99	.....	36.59	.....	28.52	2.33	.....
1861	20.26	.....	60.83	42.38	.....	2.75	.....	1886	18.71	18.96	38.08	.....	34.52	2.27	.....
1862	23.92	.....	70.42	41.75	.....	3.47	.....	1887	20.93	21.37	43.59	.....	37.08	2.30	3.15
1863	35.24	.....	91.04	76.88	.....	5.13	.....	1888	18.88	17.38	39.67	.....	29.83	2.03	2.55
1864	59.22	.....	146.46	126.00	.....	7.85	.....	1889	17.76	18.00	38.30	.....	29.25	2.00	2.49
1865	46.08	.....	106.46	98.63	.....	7.08	.....	1890	18.41	18.87	41.25	.....	31.78	2.00	2.51
1866	46.84	.....	98.13	86.75	.....	6.97	.....	1891	17.52	15.95	38.38	.....	29.92	1.86	2.04
1867	44.08	.....	87.08	83.13	166.00	5.92	.....	1892	15.75	14.37	36.79	.....	30.00	1.83	1.70
1868	39.25	.....	85.63	78.88	158.46	5.17	.....	1893	14.52	12.87	33.53	.....	28.12	1.44	1.49
1869	40.61	.....	81.67	77.25	132.19	4.85	.....	1894	12.66	11.38	26.88	.....	24.00	1.08	1.11
1870	33.23	.....	78.96	72.25	106.79	4.40	.....	1895	13.10	12.72	28.09	.....	24.33	1.56	1.69
1871	35.06	.....	78.54	70.38	102.52	4.52	.....	1896	12.95	12.14	27.22	.....	28.00	2.36	2.54
1872	48.94	.....	97.63	85.13	111.94	5.46	.....	1897	12.10	10.13	24.73	.....	18.75	1.47	1.46
1873	42.79	.....	86.43	76.87	120.58	4.90	.....	1898	11.66	10.33	23.93	.....	17.62	1.31	1.45
1874	30.19	.....	67.95	58.75	94.28	3.99	.....	1899	19.36	19.03	43.75	.....	28.12	2.21	2.80
1875	25.53	.....	60.85	47.75	68.75	3.42	.....	1900	19.98	19.49	48.12	.....	32.29	2.46	2.76
1876	22.19	.....	52.85	41.25	59.25	2.98	.....	1901	15.87	15.93	40.38	.....	27.33	2.29	2.41
1877	18.92	.....	45.55	35.25	45.58	2.57	.....	1902	22.19	20.67	43.53	.....	28.00	2.29	2.15
1878	17.67	.....	41.24	33.75	42.21	2.31	.....	1903	19.92	18.98	39.59	.....	28.00	2.36	2.13
1879	21.72	.....	51.85	41.25	48.21	2.69	.....	1904	15.57	13.76	33.17	.....	28.00	2.01	1.96
1880	28.48	.....	62.04	49.25	67.52	3.68	.....	1905	17.88	16.36	41.89	.....	28.00	2.00	1.93
1881	25.17	.....	58.05	47.13	61.08	3.09	.....	1906	20.98	19.54	43.23	.....	28.00	2.13	1.98
1882	25.77	.....	54.51	45.50	48.50	3.47	.....	1907	22.40	19.43	40.87	.....	28.00	1.86	1.82
1883	22.42	.....	44.24	.....	37.75	3.06	.....	1908	17.24	17.23	39.87	.....	28.00	1.83	1.99

IRON AND STEEL PRODUCTION OF THE WORLD.

The production of pig iron and steel in 1908 showed a large decrease as compared with 1907. This was chiefly due to the great falling off in the United States; but the principal other producing countries also diminished their output. The reaction in the iron and steel industries, which began in this country, was felt all over the world. The changes are shown in the accompanying tables, in which the figures for all countries are given in metric tons, for purposes of comparison.

The total decrease in 1908, as compared with 1907, was no less than 12,476,559 tons, or 20.5 per cent., a loss of over one-fifth. The decrease of over one-third in the United States was the extreme, and was not approached in any other country. We are not accustomed, however, to expect such violent fluctuations in Europe, and the losses were sufficient to indicate a falling trade activity. Indeed, so sharp a curtailment of production has never before been seen in this country as that which came at the end of 1907.

The change is seen in the proportion of individual countries to the total output. In 1907 the United States made 43.1 per cent. of the

world's pig iron, and 46.3 per cent. of the steel; but in 1908 its pro portions fell to 33.6 and 32.6 per cent., respectively.

The decrease in steel was 8,246,000 tons, or 16.1 per cent., being proportionally less than that in iron. The increasing use of steel in place of iron is again illustrated. It may be also that the larger use of scrap in the open-hearth furnace has some effect. At any rate, the proportion of steel to pig iron has been gaining steadily for years, and it continued to gain in 1908.

PIG IRON PRODUCTION OF THE WORLD.
(In metric tons.)

Year.	Austria-Hungary.	Belgium.	Canada.	France.	Germany.	Italy.	Russia.
1900	1,311,949	1,161,180	87,612	2,714,298	7,549,665	23,990	2,296,191
1901	1,300,000	765,420	248,896	2,388,823	7,785,887	25,000	2,869,306
1902	1,335,000	1,102,910	325,076	2,427,427	8,402,660	24,500	2,597,435
1903	1,355,000	1,299,211	269,665	2,827,668	10,085,634	28,250	2,486,610
1904	1,369,500	1,307,399	274,777	2,999,787	10,103,941	27,600	2,978,325
1905	1,372,300	1,310,290	475,491	3,077,000	10,987,623	31,300	2,125,000
1906	1,403,500	1,431,160	550,618	3,319,032	12,478,067	30,450	2,350 000
1907	1,405,000	1,427,940	590,444	3,598,949	13,045,760	32,000	2,768,220
1908	1,390,000	1,206,440	572,123	3,391,150	11,813,511	32,500	2,748,000

Year.	Spain.	Sweden.	United Kingdom.	United States.	All Other Countries.	Total.
1900	289,788	526,868	9,003,046	14,009,870	625,000	39,599,457
1901	294,118	528,375	7,977,459	16,132,408	635,000	40,950,692
1902	330,747	524,400	8,653,976	18,003,448	615,000	44,342,579
1903	380,284	506,825	8,952,183	18,297,400	625,000	47,113,730
1904	386,000	528,525	8,699,661	16,760,986	633,000	46,069,501
1905	383,100	531,200	9,746,221	23,340,258	655,000	54,054,783
1906	387,500	552,250	10,311,778	25,706,882	650,000	59,074,861
1907	385,000	603,100	10,082,638	26,193,863	556,900	60,680,014
1908	375,000	563,300	9,438,477	16,190,994	550,000	48,271,555

STEEL PRODUCTION OF THE WORLD.
(In metric tons.)

Year.	Austria-Hungary.	Belgium.	Canada.	France.	Germany.	Italy.	Russia.
1900	1,145,654	655,199	23,954	1,565,164	6,645,869	115,857	2,217,752
1901	1,142,500	526,670	26,501	1,425,351	6,394,222	121,300	2,230,000
1902	1,143,900	776,375	184,950	1,635,300	7,780,682	119,500	2,183,400
1903	1,146,000	981,740	181,514	1,854,620	8,801,515	116,000	2,410,938
1904	1,195,000	1,069,880	151,165	2,080,554	8,930,291	113,800	2,811,948
1905	1,188,000	1,023,500	403,449	2,210,284	10,066,553	117,300	1,650,000
1906	1,195,000	1,185,660	515,200	2,371 377	11,135,085	109,000	1,763.000
1907	1,195,500	1,183,500	516,300	2,677,805	12,033,632	115,000	2,076,000
1908	1,120,000	1,065,500	598,183	2,727,717	10,480,349	110,000	2,341,000

Year.	Spain.	Sweden.	United Kingdom.	United States.	All Other Countries.	Total.
1900	144,355	300,536	5,130,800	10,382,069	400,000	28,727,239
1901	122,954	269,897	5,096,301	13,689,173	405,000	31,449,869
1902	163,564	283,500	5,102,420	15,186,406	412,000	34,972,497
1903	199,642	317,107	5,114,647	14,756,691	418,000	36,298,414
1904	193,759	333,522	5,107,309	13,746,051	415,000	36,148,079
1905	237,864	340,000	5,983,691	20,354,291	426,000	43,900,648
1906	251,600	351,900	6,565,670	23,772,503	420,000	49,635,998
1907	247,100	443,000	6,627,112	23,733,391	405,000	51,273,340
1908	239,500	427,100	5,380,372	14,247,619	300,000	43,027,340

Some notes follow on production and on the condition of the iron industry in different countries.

Australia.—The Commonwealth Parliament passed a bill for "the encouragement of manufactures," and regulations prepared under this act were issued by the government. A bounty of 12s.—$2.88—per ton will be paid on pig iron made from Australian ore, the quantity to be 1000 tons or over; and the same amount on bar iron puddled from Australian pig, at least 250 tons in quantity; and on steel made from Australian iron, not less than 100 tons to be considered. Bounties of 10 per cent. on value—which apparently means current price—are to be paid for iron or steel sheets or plates, minimum 100 tons; steel or iron wire or wire netting, minimum 50 tons; iron or steel pipes and tubes, minimum 50 tons. All are to be made from Australian iron. The year for bounties is to begin May 1, and the total amount to be paid is limited to £180,000 in five years.

Belgium.—Pig iron output in 1908 showed a decrease of about 8 per cent., and there was a parallel decrease in finished iron and steel. The iron output and the foreign trade are shown in the accompanying tables.

PIG IRON PRODUCTION IN BELGIUM.
(In metric tons.)

	1903	1904	1905	1906	1907	1908
Foundry iron	.....	99,350	98,170	101,430	100,020	.....
Forge iron	.....	224,410	206,309	226,900	220,430	.....
Steel pig	.....	963,840	1,006,641	1,103,130	1,101,490	.....
Total	1,216,500	1,287,400	1,311,120	1,431,460	1,427,940	1,206,440

FOREIGN TRADE OF BELGIUM.

	Imports.				Exports.			
	1905	1906	1907	1908	1905	1906	1907	1908
Pig iron	507,970	694,530	.....	481,124	41,295	31,445	.....	119,095
Wrought iron	97,864	83,643	.....	270,273	628,473	530,119	.....	941,374
Steel	230,169	259,077	.....		395,229	245,101	.....	
Total	936,003	1,037,250	957,286	751,397	1,044,997	807,665	1,128,287	1,060,469

Canada.—Production is given in the accompanying tables. The changes shown were not large. The bounties paid to manufacturers under Dominion laws in 1908 were: For pig iron from Canadian ore, $213,458; pig from imported ore, $569,170; steel ingots, $917,877; steel wire-rods, etc., $297,779; total, $1,998,284. Practically all the steel made is basic.

An important incident in 1908 was the formation of the Canada Iron Corporation, Ltd., which acquired and consolidated under one management the properties and businesses of the Canadian Iron and Foundry Company, the Canada Iron Furnace Company, the John Macdougall Company, the Annapolis Iron Company, and the majority of the outstanding stock of the Londonderry Iron and Mining Company. These properties were operated by the Drummond interests—which control the new company—and by other interests. The Corporation owns iron mines, blast furnaces and foundries in Nova Scotia, New Brunswick, Quebec and Ontario. To these it is proposed to add steel works and other improvements.

PIG IRON PRODUCTION IN CANADA.

	1902	1903	1904	1905	1906	1907	1908
Foundry and forge	.....	.....	.....	146,698	130,120	84,979	114,951
Bessemer	.....	.....	.....	149,203	165,609	154,910	112,811
Basic	.....	.....	.....	172,102	246,228	341,257	335,410
Total	319,557	265,418	270,942	468,003	541,957	581,146	563,172

STEEL PRODUCTION IN CANADA (a)

	1907				1908			
	Converter.	Open-Hearth.	Special.	Total.	Converter.	Open-Hearth.	Special.	Total.
Imports	225,989	459,240	1,151	686,380	135,557	443,442	713	579,712
Castings	.....	20,602	.....	20,602	.....	9,051	.....	9,051
Total	225,989	479,842	1,151	706,982	135,557	452,493	713	588,763

(a) Reported by Statistical Section, Mines Department.

China. (By T. T. Read.)—China has but one plant which makes iron and steel by the use of modern methods. This is situated at Hanyang (Hankow) and is operated by the Han-Yeh-Ping Iron and Coal Company. The iron ores are mined at Tayeh, in Hupei province, about 50 miles southeast of Hankow. The ore is a hematite of good quality, except that it is too high in phosphorus for the bessemer process, and is mined in open cuts. It is transported by a short railroad from the mine to the river, and sent in lighters to the works. The coke is produced at the company's mines at Ping-hsing, in the province of Kiangsi.

Two blast furnaces of 120 tons capacity each are now in operation at Hanyang, and a third of 250 tons capacity is under construction. There are three open-hearth furnaces of 30 tons capacity each and a well-equipped rolling mill. The principal work of the plant is the making of steel rails, fishplates and track-bolts, although it also supplies

steel to the neighboring arsenal. The company reports that 327,000 tons of iron ore was mined at Tayeh in 1908, of which about 100,000 tons were exported to Japan, for the use of the Japanese government iron works at Wakamatsu. Pig iron produced was 66,410 tons, most of which was converted into steel.

The chief production in native furnaces is in Shansi; this has been described by Shockley in *Trans.* A. I. M. E. He estimates that 50,000 tons of pig iron are produced annually in this province.

Iron ores are also worked in Kuangtung and Nganhwei, and ores occur in several of the other provinces, but no definite information is available regarding any of these.

France.—There was a moderate decrease in production, as shown in the accompanying tables, but the depression was less marked than in most other countries.

PIG IRON PRODUCTION IN FRANCE.
(In metric tons.)

	1902	1903	1904	1905	1906	1907	1908
Foundry	.....	.....	.....	635,672	591,275	651,700	695,527
Forge	.....	.....	.....	705,691	741,571	673,885	543,067
Bessemer	.....	.....	.....	160,411	149 971	122,046	118,121
Basic	.....	.....	.....	1,530,671	1,784,726	1,988,343	1,949,107
Special irons	.....	.....	.....	44,267	51,489	152,975	85,328
Total	2,427,427	2,827,668	2,999,787	3,076,712	3,319,032	3,588,949	3,391,150

The consumption of ore in 1908 amounted to 7,357,686 tons of domestic and 1,508,114 tons of foreign iron ore, 144,339 tons of foreign manganese and 649,652 tons of scrap, cinder and burned pyrites.

IRON AND STEEL PRODUCTION IN FRANCE.
(In metric tons.)

	1902	1903	1904	1905	1906	1907	1908
Wrought iron	625,826	595,831	554,632	669,841	747,900	687,249	563,745
Steel ingots	1,635,300	1,854,620	2,080,554	2,210,284	2,436,322	2,766,773	2,727,617
Finished steel	1,231,652	1,317,400	1,482,708	1,442,071	1,454,456	2,261,217	1,894,022

In 1908, of the total steel product, 77,581 tons were made in the acid converter, 1,632,296 tons in the basic converter, 1,002,789 tons in the open-hearth, 12,662 tons in the crucible and 2289 tons in the electric furnace. The materials worked up comprised 18,684 tons of ore, 92,239 tons of bessemer pig, 1,817,170 tons of basic pig, 374,960 tons of forge iron, 118,849 tons of special pig and 797,961 tons of scrap and old material.

FOREIGN TRADE OF FRANCE.
(In metric tons.)

Year.	Imports.			Exports.		
	Pig and Scrap.	Wrought Iron.	Steel.	Pig and Scrap.	Wrought Iron.	Steel.
1907	267,320	38,065	11,059	538,965	47,830	257,393
1908	248,319	34,225	9,827	571,719	49,804	311,687

Germany.—The production and foreign trade of Germany are given in the accompanying tables. The year 1908 was one of depression, though far less marked than in the United States. It may be noted

PRODUCTION OF PIG IRON IN GERMANY.
(In metric tons.)

	1903	1904	1905	1906	1907	1908
Foundry iron	1,796,773	1,865,599	1,905,668	2,106,684	2,259,416	2,254,644
Forge iron	859,253	819,239	827,498	854,536	786,113	635,228
Steel pig	703,130	636,350	714,335	943,573	1,034,650	934,940
Bessemer pig	446,701	392,706	425,237	482,740	471,355	361,472
Thomas pig	6,277,777	6,390,047	7,114,885	8,088,534	8,494,226	7,627,227
Total	10,085,634	10,103,941	10,987,623	12,478,067	13,045,760	11,813,511

PRODUCTION OF STEEL IN GERMANY.
(In metric tons.)

	1905		1906		1907		1908	
	Acid.	Basic.	Acid.	Basic.	Acid.	Basic.	Acid.	Basic.
Converter ingots	424,196	6,203,706	407,668	6,945,526	387,120	7,212,454	374,100	6,510,754
Open-hearth ingots	165,930	3,086,590	230,668	3,534,612	212,620	4,039,940	224,211	3,969,595
Special steels	65,369	120,762	77,596	111,717	85,421	126,077	107,719	.....
Total (a)	655,495	9,411,058	715,952	10,591,855	685,161	11,378,471	706,030	10,480,349

(a) Includes direct castings.

GERMAN IMPORTS AND EXPORTS OF IRON ORE.
(In metric tons.)

	1904	1905	1906	1907	1908
Imports	6,061,127	6,085,196	6,730,636	8,476,076	7,732,729
Exports	3,440,346	3,698,563	3,212,977	3,904,400	3,067,870

GERMAN EXPORTS AND IMPORTS OF IRON AND STEEL.
(In metric tons.)

	1904	1905	1906	1907	1908
Exports	2,770,276	3,349,968	3,619,796	3,432,707	3,731,289
Imports	344,967	322,907	690,081	813,104	559,530

that a diminished production was accompanied by increased exports. This was largely due to the work of the Steel Syndicate, which pushed foreign trade in every possible way, to make up for the loss in domestic trade. This was carried so far that serious complaints were made that foreign orders were taken at unremunerative prices. Toward the end of 1908 some disturbance in trade was caused by the breaking up of two of the pig iron syndicates, and a subsequent fall in prices. The Steel Syndicate continued to keep its hold on the trade.

The fourth table shows the increase in iron production and consumption in Germany over a period of 28 years. In this table steel and finished products are reduced to terms of pig iron. Even taking 1908 as a period of depression, it is shown that in that year the production of iron was four times that of 1880, and the consumption rather more than four times.

GROWTH OF CONSUMPTION OF IRON IN GERMANY.
(In metric tons.)

	1880	1890	1900	1907	1908
Pig iron production	2,729,038	4,658,451	8,520,541	13,045,760	11,813,511
Imported as pig	238,572	405,627	827,095	607,729	399,661
Imported as steel, etc. (a)	86,524	190,892	338,980	459,060	344,583
Total supply	3,054,134	5,254,970	9,886,616	14,112,549	12,557,755
Exported as pig	318,879	181,850	190,505	385,766	421,611
Exported as steel, etc. (a)	982,721	1,152,169	2,118,772	4,706,587	4,930,399
Total exports	1,301,600	1,334,019	2,309,277	5,092,353	5,352,010
Consumption	1,752,534	3,920,951	7,377,339	9,020,196	7,205,745
Consumption per head, kg	25.2	81.7	131.1	145.1	114.4

(a) Reduced to terms of pig iron at the rate of 1 ton steel = 1.33 tons pig iron.

India.—Concerning the most important event in the iron industry of British India for many years, U. S. Consul General W. H. Michael reports from Calcutta, in December, 1908: The Tata Iron and Steel Company, of Bombay, reports that the ore in its Ramana manganese property has proved to be of great value. The exploration and development of the coal and limestone properties already acquired are proceeding, and tramways are under consideration. The board has selected a new site for the erection of works near Kalimati station, and about 20 square miles have been taken on a long lease for the purpose. A railway of 45 miles is already under construction. The design and plans for machinery and plant are being energetically executed.

Japan.—According to the Director General of the Imperial Steel Works, Japan, the report circulated last spring that the government foundry and steel works are proving a failure is incorrect. The steel

works are in full operation, employ 7000 men, and are manufacturing structural shapes, rails, plates, sheets, galvanized flat and corrugated sheets, rods, wire, tires, axles, crucible steel, bolts, nuts, spikes, rivets, shrapnel shells, etc. The plant includes three blast furnaces, with a total daily capacity of 450 tons, with pig-casting machine; ten basic 25-ton open-hearth furnaces; three acid Bessemer converters of 10 tons capacity each; cogging mills, rail mill, bar mill, plate mill, sheet mill, rod mills, etc.; crucible steel plant; wire drawing plant, galvanizing plant, boiler and bridge shop.

Russia.—Production, so far as can be estimated from advance statements covering part of 1908, was less than in 1907, but greater than that of 1906. The home market was not in good condition; but a feature of the year was the export of a good quantity of finished steel, especially rails, to Italy and some other countries. The exports of iron ore to Germany increased. The proposed large combination of iron and steel producers was not completed, owing to financial difficulties; but the negotiations for its formation are still going on, and it is hoped that French or Belgian capital may be secured.

PRODUCTION OF IRON AND STEEL IN RUSSIA.
(In metric tons.)

	1902	1903	1904	1905	1906	1907	1908 (a)
Iron ore	3,987,303	4,218,600	5,272,300	4,050,000	4,590,000	4,400,000	4,450,000
Pig iron	2,597,435	2,486,610	2,978,325	2,125,000	2,350,000	2,768,220	2,748,000
Steel ingots	2,183,400	2,410,938	2,811,948	1,650,000	1,763,000	2,076,000	2,341,000
Steel rails	382,152	332,367	401,541	275,000	315,000	.....	.....

(a) Estimated.

Spain.—The iron industry of Spain showed but little change in 1908. There was, however, a large decrease in the exports of iron ore, which form an important part of the Spanish foreign trade.

PRODUCTION OF IRON AND STEEL IN SPAIN.
(In metric tons.)

	1902	1903	1904	1905	1906 (a)	1907 (a)	1908 (a)
Pig iron	330,747	380,284	386,000	383,100	387,500	385,000	375,000
Wrought iron	53,252	53,288	53,177	52,250	57,100	53,200	51,000
Bessemer steel	103,389	105,263	93,100	113,664	116,200	115,500	111,500
Open-hearth steel	60,175	94,379	100,659	124,200	135,400	131,600	128,000
Total steel	163,564	199,642	193,759	237,864	251,600	247,100	239,500

(a) Estimated.

Sweden.—Like nearly all producing countries in Europe, the Swedish production showed a moderate decrease. The changes, however, were

not of great importance. Production and foreign trade are given in the accompanying tables.

SWEDISH PRODUCTION AND EXPORTS.
(Metric Tons.)

	1907			1908		
	Pig Iron.	Wrought Iron.	Steel.	Pig Iron.	Wrought Iron.	Steel.
Production	603,100	177,100	(a)443,000	563,300	148,500	(b)427,100
Exports...	146,000	44,100	209,100	116,000	26,000	170,500

(a)Includes 82,000 tons bessemer and 361,000 tons open-hearth steel.
(b) Includes 79,500 tons bessemer and 347,600 open-hearth.

United Kingdom.—The production of pig iron in the United Kingdom in 1908 was: First half, 4,635,851; second half, 4,653,989; total, 9,289,840 long tons; a decrease from 1907 of 634,016 tons, or 6.4 per cent. The loss in output was comparatively small, but enough to show a material decrease in trade. Moreover, the depression was continuous through the year, the second half showing an advance of only 18,138 tons over the first half. The average number of furnaces in blast in 1908 was 309, and the average make per furnace for the year, 29,970 tons.

The steel production in 1908, as given in the accompanying tables, showed a larger decrease than the pig iron output; it was 1,227,106 tons, or 18.8 per cent., less than in 1907, and the smallest reported since 1904. In 1908 the open-hearth make was 72.1 and the converter 27.9 per cent. of the total. Of the total 65.8 per cent. was acid and 34.2 basic steel.

PRODUCTION OF STEEL IN THE UNITED KINGDOM.
(In tons of 2240 lb.)

	1903	1904	1905	1906	1907	1908
Open-hearth	3,124,083	3,245,346	3,879,743	4,554,936	4,663,489	1,478,539
Bessemer	910,015	1,781,533	2,009,712	1,907,338	1,859,259	3,817,103
Total	5,034,101	5,026,879	5,889,460	6,462,274	6,522,748	5,295,642

ACID AND BASIC STEEL IN THE UNITED KINGDOM.
(In tons of 2240 lb.)

	1906			1907			1908		
	Acid.	Basic.	Total.	Acid.	Basic.	Total.	Acid.	Basic.	Total.
Open-hearth	3,378,691	1,176,245	4,554,936	3,384,780	1,278,709	4,663,489	2,578,840	1,238,263	3,817,103
Converter	1,307,149	600,189	1,907,338	1,280,315	578,944	1,859,259	906,466	572,073	1,478,539
Total	4,685,840	1,776,434	6,462,274	4,665,095	1,857,653	6,522,748	3,485,306	1,810,336	5,295,642

The total production of finished steel, so far as reported, was 4,518,536 tons; but this does not include bessemer steel bars, forgings and some other classes of material, which would bring the total up to about 5,100,000 tons.

The production of wrought, or puddled iron, on the other hand, increased for the first time in nearly 10 years. The total make of puddled bars in 1908 was 1,168,115 tons; an increase of 193,032 tons, or 19.8 per cent. over 1907. The production of wrought iron in finished forms of all kinds in 1908 was 1,050,772 tons.

The production of rails—the most important item of finished material—in 1908 was: Bessemer steel, 715,407; open-hearth, 192,525; total, 907,932 tons, being a decrease of only 4176 tons, or 0.5 per cent. from 1907. There was a decrease of 117,169 tons in bessemer, but an increase of 112,993 tons in open-hearth steel rails.

Great Britain is still the chief exporting nation, but its trade decreased materially in 1908. This was in part due to general depression, but in part it must be attributed to the active competition of Germany.

EXPORTS AND IMPORTS OF UNITED KINGDOM.
(Values.)

	1903	1904	1905	1906	1907	1908
Exports	£54,741,296	£53,587,013	£60,524,755	£75,256,655	£88,448,689	£78,914,315
Imports	8,662,481	12,529,212	13,128,270	13,486,724	12,527,157	12,236,417

EXPORTS, UNITED KINGDOM.
(In long tons.)

	1903	1904	1905	1906	1907	1908
Pig iron	1,065,380	810,934	981,891	1,662,820	1,947,925	1,295,767
Wrought iron	203,619	170,505	183,406	200,182	211,771	172,072
Sheets	.....	385,408	407,021	442,414	469,329	390,281
Plates	161,722	152,337	204,503	275,045	300,590	207,278
Rails	604,076	525,371	546,644	460,328	433,638	435,739
Steel shapes, etc	156,821	122,930	151,809	226,230	338,716	275,022
Tin plates	292,800	359,634	354,951	374,802	405,329	403,007
All other kinds	.....	735,723	891,290	1,040,379	1,059,068	921,810

NOTES ON IRON ORE PRODUCTION IN THE UNITED STATES.

The production of the two more important States—Minnesota and Michigan—has already been treated in the paragraphs on the Lake Superior region. Some notes on iron ore in other parts of the United States follow.

Alabama.—As is well known, the ore reserves of Alabama are greater than those of any section of the United States, except the Lake Superior region. In this State the Birmingham district ranks second in importance in the United States as a producer of iron ore, its output during 1907 having been in excess of 4,000,000 long tons, from which more than 1,500,000 tons of pig iron were made. In 1908 there was a decrease of about 20 per cent.

The depression in business and actual production did not prevent preparations for a future increase. Extensive improvements were begun at the mines of the Tennessee Coal, Iron and Railroad Company and the Sloss-Sheffield Steel and Iron Company; while some other companies made arrangements to open mines and test new deposits.

The iron ores of the Birmingham district may be conveniently classed under the following heads: Clinton red hematites; brown ores, known as brown hematites; and Cambrian gray and red hematites of eastern Alabama. These ores are used in the manufacture of pig iron for foundries, pipe works and mills, and for making basic iron for open-hearth steel plants. Usually they are too high in phosphorus to be used in the manufacture of bessemer steel.

The geology and origin of these ores has been discussed by many eminent geologists, and it is not proposed in this article to enter into any particulars on this point.

Georgia. (By S. W. McCallie.)—The iron ore production of Georgia was materially affected in 1908 by the price of ore, some of the more important brown iron ore mines having entirely abandoned operations for a part of the year. Like conditions also prevailed in the fossil iron ore district. The only new company of note, which was organized to mine iron ore in Georgia, is the Pigeon Mountain Iron Company. The property of this company, which is situated in Walker county, about 18 miles south of Chattanooga, comprises an area of about 4000 acres. The ore found on this property is fossil or Clinton ore, and the deposit is supposed to be quite extensive.

Iowa. (By James H. Lees.)—The Missouri Iron Company of St. Louis continued in 1908 the development of the ore deposits at Waukon, in Allamakee county, and also carried on the erection of a concentration plant for the treatment of the ore. This work was not completed during 1908, but the plant was ready to be put into operation early in 1909. The company is building a railroad four miles long from Waukon to the ore beds, and so will have direct connection with the Chicago, Milwaukee & St. Paul, which will carry its ore to the concentrator at Waukon Junction, 23 miles distant. This plant is built for a capacity of 1800 tons ore per day, and is constructed on the gravity system. The treat-

ment consists of crushing, screening and sizing, jigging and roasting, and brings the ore up to about 60 per cent. metallic iron. The company plans to build a plant at St. Louis to use the ore raised at Waukon. The main ore bed covers about 240 acres and varies in depth from 30 to 70 and in some cases 100 ft. In addition, there are other smaller bodies in the vicinity. The deposit was very closely platted and numerout test-holes sunk; a conservative estimate places the amount of ore at 11,500,000 tons. This excludes the smaller bodies, as well as some parts of the main body where the ore is leaner. It is safe to say that the actual amount of ore available may be estimated at 18,000,000 tons. This will average 37 per cent. from the grass roots down, and some portions have a higher iron content, in some cases from 47 to 67 per cent. metallic iron. Preliminary tests have been made and have demonstrated the feasibility of the process which is to be employed in concentrating the ore up to a standard where it will be of economic value. As soon as the final trials have been made and it is evident that the plant will do the work expected of it, operations will be begun and the treatment of the ore undertaken on a commercial scale.

New Jersey. (By Henry B. Kümmel.)—The iron-mining industry in New Jersey during 1908 suffered severely as a result of the general business stagnation. Several mines which had been operated for years without interruption were not producers, while others show greatly reduced outputs. The following mines were active during the year, but most of them for only portions of the time: Ahles, Shoemaker, Washington, Mount Hope group, Richard, Hurd, Hude, Hoff, Wharton, Wood & DeCamp, Orchard and Peter.

The total production amounted to 432,566 long tons, nearly all magnetite. The Shoemaker mine produced some limonite, and the ore of the Ahles mine is a soft manganiferous mixture of limonite and magnetite. The value of the ore at the mines was about $1,257,000, an average per ton of $2.94. At the close of the year there were on the mine dumps a little over 115,000 tons. Most of the ore carried from 54 to 58 per cent. metallic iron. These figures indicate a loss of 125,571 tons, a decrease in value of $516,184, and a falling off in the average value per ton of 24c. as compared with the previous year.

The years 1890 and 1907 mark high points in the iron-ore industry of New Jersey, the production being slightly in excess of 550,000 tons each year. There was a steady decrease from 1890 to 1897, when the production was only 257,000 tons. It then rose year by year until another maximum was reached in 1907. The year 1908 was, therefore, the first for a decade to show a falling off in this industry. The mining companies, however, are in a better position than at the beginning

of the depression in the early '90s. There has been a centralization in management, much-needed economies in mining methods have been introduced, labor-saving machinery has been installed, magnetic cobbers and separators have been built. The result of these changes has been to decrease mining costs and place the mines in a better position to withstand a period of depression than ever before.

New York. (By D. H. Newland.)—The industrial depression of 1908 brought an end to the steady progress that has for some time characterized the iron mining industry. Under more favorable conditions the production, undoubtedly, would have undergone further increase and probably have reached the largest total in the history of the State, since there were several properties which had just been brought to the producing stage when the setback came. With the contraction of the market these suspended operations; while the active mines began to curtail their output, a few closing down entirely later. The production for the year amounted to 697,473 long tons against 1,018,013 in 1907. In the Adirondack region, Mineville was, of course, the principal center of activity; the mines belonging to Witherbee, Sherman & Co. and the Port Henry Iron Ore Company were worked steadily, though on a reduced scale. Both companies have profited by the opportunity to carry out needed developments and improvements, and will be in a position to enlarge their operations in the future. The destruction by fire of the new mill at Lyon Mountain curtailed the output from that locality. The old Cheever mine, near Port Henry, was reopened and the Salisbury mine, north of Little Falls, Herkimer county, was equipped with a mill and made ready for active work during the season of 1909. A novel feature, which may have an important bearing on the future use of the Adirondack magnetites, has been the experiments in nodulizing and briquetting the finer magnetic concentrates. As a result of the success obtained with the nodulizing process, the Benson Mines Company contemplates the erection of a plant for that purpose. The experiments have shown that the concentrates can be agglomerated readily, and with beneficial effect as regards the physical and chemical character of the product.

Tennessee.—Some progress was made in opening iron ore mines in the Chattanooga district, but the production of ore was limited by the general depression.

Texas.—It is reported from reliable sources that active prospecting was carried on in the western section of the State. Options were taken on a number of properties where promising indications of iron ore were found.

Utah.—Some progress has been made in exploring the iron deposits in southern Utah, which are known to be extensive, but actual development was not begun in 1908. The most extensive work done was on a deposit in Iron county, in the Iron Springs district, which was explored by about 1600 pits, but as the deepest pit was carried down only 130 ft., the vertical depth of the ores is not known. Their total tonnage, as far as can be measured, is 40,000,000 tons. The nearest railroad station is 22 miles distant. The ores occur in disconnected masses, within an area 1½ by 20 miles in extent, most of them at or near the contact of limestone and andesite, the latter existing as a laccolith or sill. Iron ores and other minerals were deposited as fissure veins in the andesite and as replacement bodies and fissure veins in the overlying limestone. Finally erosion exposed the laccoliths and the ores. The ores in the andesite may extend down to very considerable depths, but this remains to be proved.

Iron Ore in Foreign Countries.

There was active interest in prospecting for and examining iron ore deposits in various countries. Some notes concerning these are appended.

Canada.—The Mines Branch of the Dominion Bureau of Mines has been engaged in a detailed study of iron ore resources, and has published a preliminary report on the subject. In British Columbia Einar Lindeman investigated the iron-ore deposits on Vancouver and Texada islands, British Columbia, and local conditions generally with a view of ascertaining the prospect for the establishment of the iron industry in that province. His report is to the effect that as regards ore supply, the deposits on Texada island and at Head bay, Klaanch river and Quinsam river on Vancouver island, are of sufficient magnitude to furnish ore to a blast furnace for a number of years. These deposits of magnetite ore are all low in phosphorus, but as a rule high in sulphur, though not sufficiently so to render them unfit for smelting. The collieries on Vancouver island can furnish a supply of good coke. The limestone deposits are of great extent and unusual purity, and would furnish an inexhaustible supply of excellent fluxing material. The deposits of raw material being adjacent to the coast are favorably situated for transportation, and shipments could be made all the year round direct to a furnace anywhere on the coast line.

In New Brunswick Mr. Lindeman made a magnetic survey of the iron-ore deposits on the Nipisiquit river, 15 miles from Bathurst, the results of which demonstrate the utility of magneto-metric surveys of magnetite, as it was found that the commercial value of the property did not

consist in the deposits previously known, but in a large orebody on the other side of Austin brook, discovered by magnetic survey, and of which the value was proved later by boring. The location of the bore-holes was fixed from indications of the magneto-metric survey. The property was sold to a syndicate which will shortly begin operations.

In Nova Scotia iron mining was active. The Nova Scotia Steel and Coal Company began to work new mines. Other deposits are being examined.

In Ontario work has been carried on at the Moose Mountain mines, north of the Sudbury district, with good results. Some work was done also in the Port Arthur district. On the Michipicoten and the Atikokan iron ranges, in the Lake Superior district, there were no new developments of importance.

In Quebec an examination of iron deposits in Dunham township, Missisquoi county, was made, where a strip of ferruginous dolomite occurs for a distance of about 8 miles. This is sometimes strongly mineralized, presenting irregularly distributed masses of hematite, occasionally taking the form of veins up to 2 ft. in thickness. An analysis showed: Metallic iron 69.49; silica 1.32; sulphur 0.10; and phosphorus 0.08. Mr. Obalski, chief of the Department of Mines, is of the opinion that prospecting might lead to the discovery of more important masses suitable for working.

Experiments for the industrial treatment of the magnetic sands of the north shore of the St. Lawrence are in progress, showing that the sands can be concentrated into a product yielding from 67 to 70 per cent. metallic iron, about 1 per cent. titanium, and practically no sulphur or phosphorus. The magnetic sands have been treated by the Gröndal process, and concentration was satisfactory, but owing to the polished and worn condition of the sand preventing a good agglomeration, the concentrates were crushed and briquets obtained, which became well agglomerated, yielding a product comparable to that obtained from other ores. It is, therefore, hoped that the magnetic sands will be worked industrially in the near future.

Cuba.—Exploitation of the Juragua and other mines in the province of Santiago continued steadily. Extensive deposits proved by preliminary tests, have been acquired by the Spanish American Iron Company at Mayan, 14 miles from Nipe Bay. One estimate places the quantity of ore at approximately 600,000,000 tons. The company has about 1000 men engaged in the construction of a railway and harbor with mechanical and mining appliances. The work of deepening the harbor is nearly completed; of the 14-mile railroad line from the mountains to the coast the entire extent is graded, 11 miles of track laid, and

bridges are in course of construction. The steel buildings for power plant, machine shop, etc., and the dwellings and offices are being built. The appliances for loading the ore into the ships and for handling the coal are under construction and the furnaces for drying out the ore have been contracted for, while part of the railroad equipment has been delivered.

Dominican Republic.—E. M. De. Garston, British vice-consul at Santo Domingo, reports that iron is found in immense quantities in several sections of the Dominican Republic. A deposit of magnetic iron ore, about nine miles in extent, is found on the borders of the Maimon river, municipality of Cotuy, province of La Vega. In order to develop this deposit it would be necessary to canalize the Maimon river as far down as its confluence with the river Yuma, so as to render it capable of floating small boats or barges for the transport of the ore. Iron is also found in Monte Pueblo, Arbol Gordo, Sabana de Santa Rosa, and south of Yamasa, in the province of Santo Domingo; these are limonite deposits. In Sierra Prieta, close to the Ozama river, which is navigable for small craft, there are masses of magnetic iron, samples of which are said to have yielded from 60 to 70 per cent. of metal.

The Shape of the Iron Blast Furnace.

By Henry M. Howe.

For most of the ideas here presented I am indebted to the patient consideration of Frank Firmstone, F. L. Grammer, and J. E. Johnson, Jr., and to many interesting talks with Sir Lowthian Bell, James Gayley, and E. S. Cooke. I doubt if any one of them agrees with all I say, and some of them certainly do not.

Though it is no doubt true that the present size and shape of the iron blast furnace have been reached in part empirically, and in another considerable part by reasoning which is none too convincing, it is nevertheless interesting, and it may be profitable, to seek the causes which through both empirical and reasoned trials, have guided the evolution of this wonderful engine to its present size, and to its shape, Fig. 1, a pair of truncated cones set end to end, the upper one upright and very acute, the lower one inverted and relatively obtuse.

Such a search as this for causes can hardly be expected to lead to very definite and positive conclusions. Nearly all that can be expected of it is that the hypotheses which it frames shall be sharp and clear cut, and therein fitted for consideration, and for improvement or rejection.

There are those who would refer the shape of the blast furnace chiefly to variations in the mechanical condition of the charge; others who would

refer it in very large part to the need of having the rising gases pass evenly through the different parts of each horizontal section, neglecting neither the axial parts, nor the circumferential, nor the intermediate ones; and still others who would refer that shape to the variations in the volume of the gases in different parts of the furnace. To the bystander it certainly seems as if each of these elements was important, and as if the blast furnace, like the human body, had to comply with a great many different conditions, some not readily grasped, some limiting it in one way, some in another, and some having to be met by such compromise as can be devised.

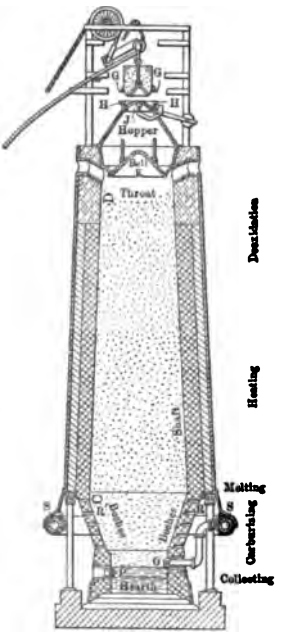


FIG. 1. SECTION OF ONE OF THE DUQUESNE BLAST FURNACES.
GG, flanges on the ore bucket; *HH*, fixed flanges on top of furnace; *J*, counterweighted false bell; *K*, main bell; *O*, tuyere; *P*, cinder ratch; *RR*, water-cooled boxes; *S*, blast pipe.

The advantage which the large unit offers in most industrial operations, of economizing installation, labor, and administration, is reinforced in case of the iron blast furnace, as indeed in case of most metallurgical furnaces, by the farther great advantage of lessening the proportion which the outer heat-radiating and hence heat-wasting surface bears to the whole mass, and hence to the unit of product; in short, of saving heat. Let us consider in what way the furnace builder's natural desire to make his furnace and his output as large as possible is limited by the conditions of the case.

The width at the tuyeres has in general been limited to about 12½ ft. by the fear that, if the hearth were materially wider, the blast would penetrate relatively feebly to its center, that hence too great a difference in conditions between center and circumference would arise, and that from this difference serious unevenness of working would result. It is true that the width of the hearth has at the Lackawanna furnaces been increased to 17 ft., and as a result of these important experiments it may well prove that a hearth-width of much more than 12½ ft. can be used with advantage. In this event it may also follow that the furnace may be widened all the way from top to bottom, and the volume and rate of production be increased in a very important degree. Indeed, the reasoning which we shall follow tends to show that, if there is to be any important increase of size, it is more likely to be an increase in diameter than in hight.

Angle of Bosh.—With the hearth diameter thus provisionally fixed at 12½ ft., the furnace designer naturally tries to gain volume as rapidly as possible by widening his furnace abruptly, and in this way comes the outward flaring bosh, which begins shortly above the tuyeres. In order that the increase of volume may be as great as the attendant conditions permit, the boshes are made as flat as practicable, i.e., the inverted cone of which the lower part of the furnace consists is made as obtuse as practicable. In practice it is found that the angle which the boshes make with the horizontal must not be less than 73 deg., and 76 deg. is a more usual angle.

If, now, we are asked what it is that prevents us from making the boshes still flatter, a very natural reason which suggests itself is that if they were flatter, the descending column of solid materials would not slide freely over them as its lower end is gradually eaten away by the burning of the coke and the melting of the ore and flux, or rather of the reduced iron, the gangue, the ash, and the flux, of which the last three in melting unite to form the slag.

In order to grasp the conditions here we should bear in mind that in the lower part of the furnace, as sketched in Fig. 2, the descending column of solids consists of coke only, because from the tuyeres to a level about 12 ft. above them the temperature is so high that everything but the coke melts, while past it the molten iron and molten slag trickle down.

To make the boshes needlessly steep, on the other hand, as in furnace No. 10, not only sacrifices room, but in fact also leads to irregular working. While we may not dogmatize as to the reasons for this, two independent ones suggest themselves. The first is that too steep a bosh tends to lead the rising gases to travel in undue proportion along the smooth walls and too little through the descending column of solids, with the

consequence that the axial ore may reach the lower part of the furnace insufficiently reduced. In short, this uneven distribution of the rising gases would make the reduction of the axial ore lag behind that of the circumferential ore, and thus set up unevenness of working.

The second is that, whereas with a flattish 76-deg. bosh the charge descends only as the lower end of the column is eaten away, because the constriction which the flatness of the bosh represents causes a degree of incipient jamming which lets the coke descend only when its lower end becomes quite free; yet if the bosh is steeper, and if the coke, therefore, descends more easily, it slides down so freely that it presses down into

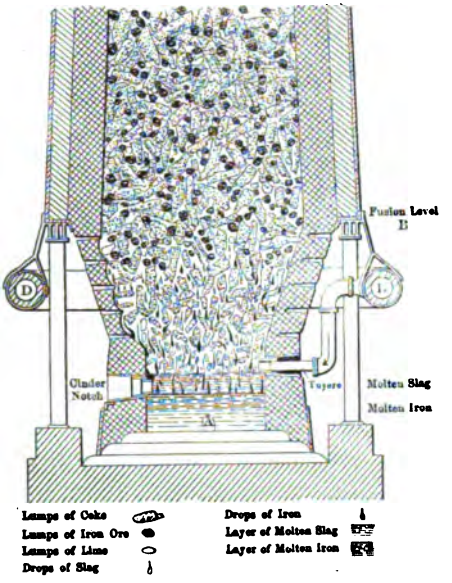


FIG. 2. LOWER PART OF THE BLAST FURNACE.

Note—The ore and lime actually exist here, not in lump form, but in powder. They are shown in lump form simply because of the difficulty of presenting to the eye the powdered state.

the molten layers of slag and iron collected at the bottom of the furnace, and there acts most energetically on them. Further, slight variations in the conditions vary the depth of this immersion of the coke in the ore and slag, and hence vary the degree to which this intense action of the coke on the slag and iron takes place. With a flattish bosh the coke is held up enough so that it never dips materially into either slag or metal; with a steeper bosh the coke slides so freely that it does so dip, and dips to a depth which varies materially. It is this variation that causes serious variations in the degree to which the energetic action of coke on molten slag and iron takes place.

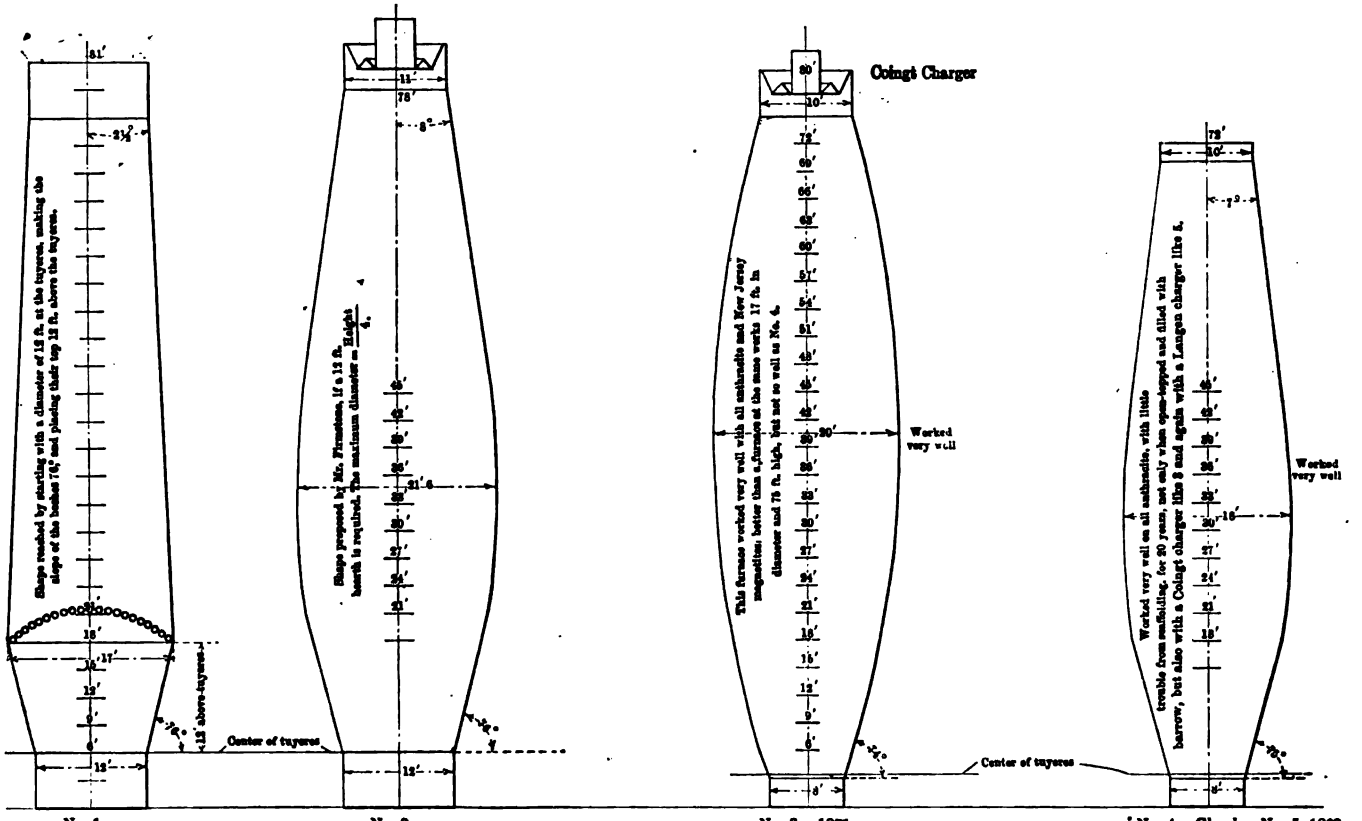

Shape reached by starting with a diameter of 12 ft. at the tuyeres, making the slope of the bosh 76°, and placing their top 12 ft. above the tuyeres.
Shape proposed by Mr. Firmstone, if a 12 ft. hearth is required. The maximum diameter = Height / 4.
Coingt Charger
This furnace worked very well with all anthracite and Mor Jersey magnetites; better than a furnace at the same works 17 ft. in diameter and 75 ft. high, but not so well as No. 4.
Worked very well
Worked very well on all anthracite, with little trouble from scaffolding, for 20 years, not only when open-topped and filled with barrow, but also with a Coingt charger like 3 and again with a Langen charger like 4.
Worked very well
Center of tuyeres
No. 1.
No. 2.
No. 3. 1871.
No. 4. Glendon No. 5, 1869.

FIG. 3. INSIDE DIMENSIONS OF BLAST FURNACES, SHOWING VARYING FORMS.

Be it understood that these statements are made tentatively, not as depicting the actual condition of things, but as indicating a possible contributory explanation of the observed fact that with too steep boshes the furnace works irregularly.

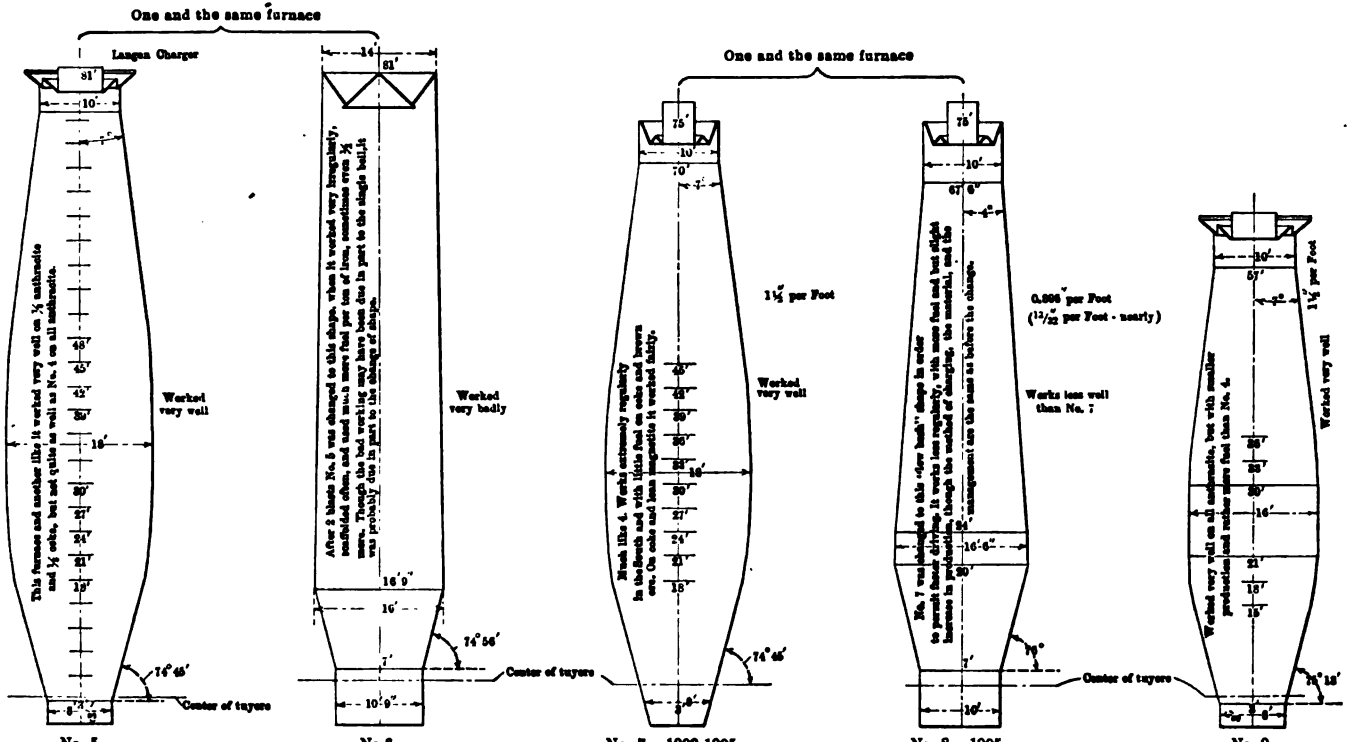
How Far Up May This Outward Flare Be Carried?—In the very swift-running furnaces of the Pittsburg district, Fig. 1, this outward slope of the boshes stops short at about 12 ft. above the tuyeres, and is there replaced by a slight batter in the opposite direction, *CD* in Fig. 1. In other words, at this level the furnaces change from a relatively obtuse inverted cone to a very acute upright one. A reason assigned for this is that, at this level, the descending charge reaches a temperature so high that the gangue and the flux soften and become pasty. Below this level the column of solids descends even through the narrowing region of the boshes, and descends freely enough, because the only solid is the coke, itself perfectly dry and without suggestion of pastiness. But at this upper level, where the gangue and flux becomes pasty, they tend to cement the whole together, coke and all, into something like what a freshly-mixed concrete would be if it had but a small proportion of grout, and if that grout were very sticky, like tar. In order that this rather pasty mass shall descend with the necessary freedom as the coke below works down, it is said to be necessary that the furnace shall, in this region of pastiness, grow wider downward, instead of growing wider upward, as it does at the boshes.

To this two objections are made: First, that such jamming does not occur; second, that it cannot occur. Let us examine these separately.

First, as to the alleged absence of jamming. In many of the older furnaces the outward batter of the boshes was prolonged far above this level, though somewhat steepened; and indeed, it was carried nearly midway toward the top of the furnace, as shown in furnaces Nos. 3, 5, and 7, which worked very well. This indeed shows that at least under those special conditions, it is wholly practicable for the charge to descend, even while in its pasty stage, through a region which grows narrower downward.

To this, in turn, it may be said that, though this may be practicable in case of relatively slow-running furnaces, and in case the slag is relatively fluid, or rather in case the slag passes rather quickly from the solid to a relatively fluid state, yet it is not practicable in case of a sticky, very calcareous slag, and of extremely rapid working, and the freedom of descent which that rapid working requires.

Second, as to the alleged impossibility of such jamming. It is held to be impossible that so wide a column, composed of such relatively small lumps, can possibly thus become so blocked, jammed, or engorged, even



No. 5. Glendon No. 2, 1882.

No. 6. Glendon No. 2, 1890.

No. 7. 1902-1905.

No. 8. 1905.

No. 9. Glendon No. 1, 1877-1886.

FIG. 4. DIFFERENT FURNACE LINES EMPLOYED IN PRACTICE.

with the aid of the pastiness of the barely melting slag, that it would not descend freely between and past converging walls. Imagine, they say, that the materials before the tuyeres and hence to the top of the boshes, in No. 1, have been melted or burnt away, so that this space is empty. How is it possible that a truly supporting arch, or rather dome, of such fine lumps should stretch across a span as wide as 17 ft., and thus hold up the weight of the column above, without the aid of the lifting frictional action of the blast?

The answer is fivefold: First, if we may judge from descriptions of what happens in many scaffoldings after the blast has been taken off, such arching does actually occur. A very trustworthy friend assures me that he has actually seen the coke arched across at or near the top of the boshes after the greater part of the space below had been emptied. The blast, of course, was off at the time.

Second, in actual running, the lifting action of the blast is actually at work.

Third, the assumed jamming need not be of the arch type; it may be of the dome type. The dome of the Pantheon has a wide opening in its center, as indeed most domes have; yet they do not fall. Indeed, each horizontal course of stones in such a dome may be stable and sustained by itself and the courses outside it, without the aid of the courses inside it. Such a domed jamming, even if confined to a relatively narrow ring next the walls, might cause grave irregularity.

Fourth, "wheat will often arch completely across the ordinary elevator bins."[1] If this is possible with so smooth a substance as wheat, with its grains all of about the same size, and without any viscid material to act as a bond, it should be far easier at the top of the boshes, (1) because of the coarseness of the coke, (2) because of its roughness, (3) because of the great variations in the size of the lumps, which leads to "chinking in," and (4) because of the assumed presence of slag in its formative sticky state.

Fifth, if the outward flare of the boshes were extended up past the fusion zone, the charge in its pasty slag-forming stage might well have enough freedom of descent to prevent its actually arching across, or even forming the beginning of domes, and yet not have the much greater freedom of descent needed for the smooth and regular working of a swift-running furnace. Such an extension of the boshes might not create complete or even incipient arches, but yet might impede harmfully a very rapid descent of a very pasty charge.

This is a question to be decided by evidence. The point which this evidence should cover is this: Has it been found practicable to make any furnace of very great output, and running on a very calcareous slag,

[1] E. P. Goodrich, *Trans.* Am. Soc. Civil Engineers, 60, 1908, p. 61.

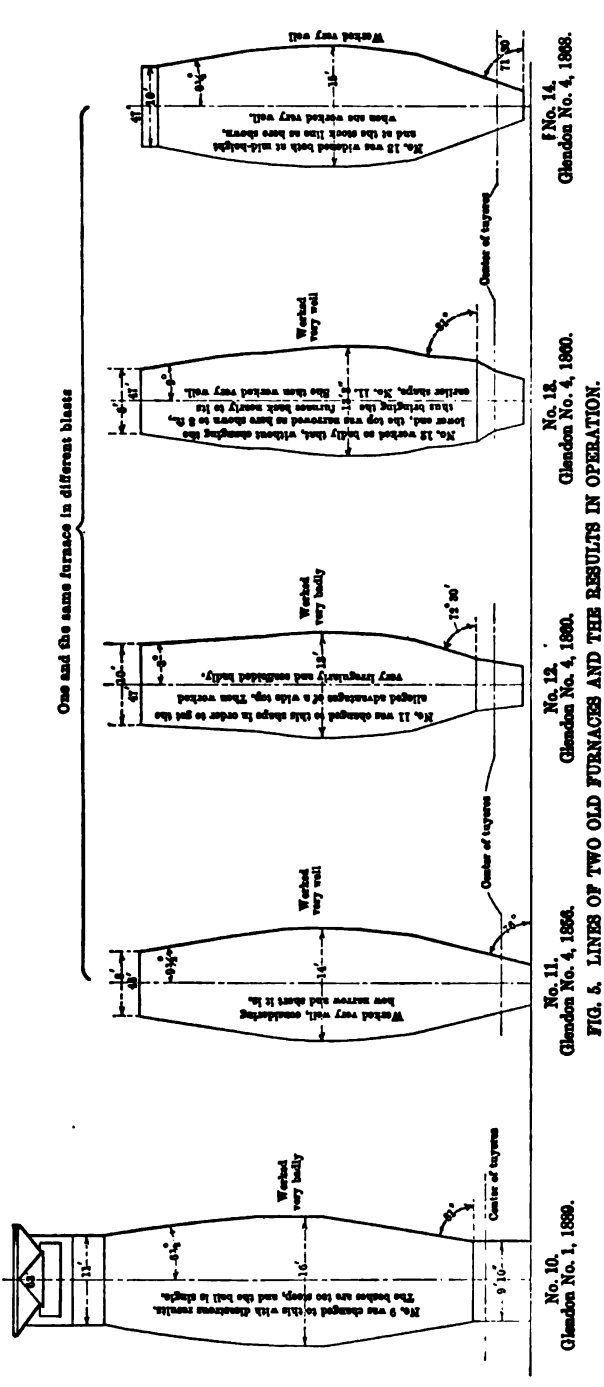

One and the same furnace in different blasts
No. 9 was changed to this shape, and the bottom was too steep, and the bell is single.
Worked very badly
Worked very well
Worked very badly
Worked very well
No. 10.
Glendon No. 1, 1859.
No. 11.
Glendon No. 4, 1866.
No. 12.
Glendon No. 4, 1860.
No. 13.
Glendon No. 4, 1860.
No. 14.
Glendon No. 4, 1868.

FIG. 5. LINES OF TWO OLD FURNACES AND THE RESULTS IN OPERATION.

work regularly and evenly in case the outward flare of the boshes was continued upward past the region of pastiness and incipient fusion?

The Diameter at the Top.—It is of fundamental importance that the charge shall be distributed properly over the top of the furnace. There are serious difficulties in the way of making this distribution proper in case the furnace is very wide at the throat (Fig. 1). In practice the diameter is rarely greater than 16 ft. at this level. On the other hand, a narrow top has the disadvantages of restricting the room available for the charge in the upper part of the furnace; of throttling back the rising gases and so increasing the power needed for driving them through the furnace; and of giving them so great a velocity that they entrain much of the fine ore and carry it out of the furnace.

The Batter from the Top Downward.—Passing downward from the top, the walls batter outward at an angle between 6 and 9 deg., not only to gain space, but also to ease the descent of the charge, which here is obstructed by the rapid deposition of carbon within the ore by the reaction, $2CO=C+CO_2$, or rather by the swelling of the individual pieces of ore to which this carbon deposition gives rise. Witness the harm done by steepening the upper part of the shaft to 3 deg. in No. 12. To widen the furnace still more rapidly would, of course, still further ease the descent of the solid column, and would still farther increase the room available. But it is feared that a more rapid widening would lead to grave irregularities in the distribution of the rising gases, and through this in the reduction of the ore, with the consequence that some of the ore might reach the bottom of the furnace insufficiently deoxidized. However true this may be, it is hard to believe that the particular angle of batter which is appropriate to the very top of the furnace, is also that most appropriate to each and every level thence downward to the top of the boshes. We naturally expect that this present shape, reached as an early approximation to the best, will later on be improved in the way of making the batter in the several different levels conform to the conditions which exist in each.

The Hight of the Furnace.—This is limited usually to 100 ft. and probably better to 80 ft. under most conditions. Mr. F. L. Grammer argued against extreme hight in his article in the *Iron Trade Review,* February, 1902. The reason for this limit probably lies not so much in any crushing and jamming effect of an excessively high column of solid materials on the coke in the lower and narrowing end of the furnace, as in the frictional resistance, which even for constant velocity, increases very rapidly with the hight of the furnace. But, in order to get the benefit of added hight the velocity of the rising gases, instead of being constant, must be increased about proportionally to the hight, as the least reflection shows. The chief purpose in lengthening the furnace is to

increase the rate of production. But the chemical needs of the process demand that, for every ton of iron made, a certain quantity of blast shall be blown through the furnace. Hence, if by hightening the furnace we hope to increase its production by, for instance, 25 per cent., we must expect to blow 25 per cent. more blast through it, and thus to increase the velocity of the rising column of gases by 25 per cent. In short, lengthening the furnace implies increasing the friction in two ways which act cumulatively, (1) lengthening the path through which the rising gases must be forced; and, (2) increasing the already very great speed at which they must be forced through their most tortuous and narrow passages. The average velocity of the gases rising through a 100-ft. furnace of, say 32,000 cu.ft. capacity, with 64,000 cu.ft. of air blown through per minute, is probably much greater than 2000 ft. per minute, or that of a "high wind." In places it may rise to 4000 ft. per minute. Conceive, now, these gases moving at this velocity, which is impressive enough when met in the open, through the narrow interstices between adjoining lumps of coke and ore; conceive further that, where these lumps come nearest to each other, at their corners where the salient points on their rough edges actually touch, there is still farther narrowing of these minute spaces through which this inverted gaseous Niagara must tear, and you may get some idea of the energy needed for overcoming the frictional resistance, and indeed some idea of the terrific dynamic conditions inside the peaceful-looking blast furnace.

With this picture in mind, not only the great entrainment of fine ore from rapidly driven high furnaces, but the hangings and slips, readily explain themselves. The frictional lifting effect of the rising gases may at times readily become so great as to interrupt the regular descent of certain parts of the column, for instance those in which the frictional resistance of the solid column to the upward passage of the blast is increased by the swelling of the ore caused by the deposition of carbon within it, and by the denseness, not to say impenetrability, of the column where thus swollen and compacted. Part of this enormous column may thus be suspended like a ball in the rising jet of a fountain, and may later fall with great violence and even with destructive effect when, by some change in the conditions, the friction is lessened. When such a suspended mass once starts downward, its fall becomes ever freer and freer, less and less impeded by the rising gases, because as it descends, between it and the furnace walls an ever widening space for the free upward rush of those gases is given by the outward batter of these walls.

The considerations which have now been presented give the profile of the furnace closely enough for the purpose of this elementary article, by fixing the hight, the diameter at top and bottom, and the batter of the boshes and of the upper part of the shaft.

The results already reached with hearths wider than 12½ ft. lead us to hope, as already indicated, that widths much greater than this may be found practicable. The proper distribution of the charge at the top of the furnace is not intrinsically so difficult a thing that it may be expected to baffle indefinitely the attempts to bring about proper distribution over much wider tops than are now used. If we could widen the furnace at top and bottom, then for all we now see we ought to be able to widen it proportionally throughout, and thus to increase its volume and production very greatly. But the difficulties in the way of increasing the hight seem of a different and more nearly insuperable kind. Therefore, we may expect that future increase of production will come rather through increase in width than through lengthening the furnace.

Mr. Firmstone gives us sketches of furnaces 1 to 14, inclusive, which are very valuable additions to our records.

Four points here deserve special notice:

(1) The good working of Nos. 3, 4, 5, 7, 9, 11, 13 and 14 in spite of the outward flare of the boshes being carried, though somewhat steepened, very far above the region of pastiness.

(2) The great harm, as regards regularity, scaffolding, and fuel consumption, done by changing from the high bosh of No. 5 to the low bosh of No. 6, and the corresponding, but much less harm done by like changes from No. 7 to No. 8.

(3) The great harm done by the extreme steepening of the boshes, to 81 deg., and substituting a single for a double bell in changing from No. 9 to No. 10.

(4) The great harm done by lessening the batter of the upper part of the shaft to 3 deg. in changing from No. 11 to No. 12, its removal on changing back in No. 13 to a batter of 9 deg., and nearly to the shape of No. 11, and the absence of harm on widening top and mid-diameter without changing the batter of the upper part of the shaft in changing again to No. 14.

Since this article was written Mr. Firmstone has called my attention to an extremely interesting discovery, an essay of Swedenborg's on the Swedish iron blast furnaces, presented to the Bergs Collegium in 1719, or 190 years ago. He says: "If the furnace had but the shape of the flame of a wax candle, which follows a hyperbolic line first expanding in width and then coming to a point above, this would probably be the shape in which the heat and the fire would have its best action." This agrees with the idea of some metallurgists of today, who would have the area throughout proportional to the volume of the furnace gases, as determined jointly by their temperature and pressure. The title of Swedenborg's paper is *Beskrifning öfner Swenska Masugnar och Theras Blasningar,* Bergs Collegio, Nov. 1719.

LEAD.

Reports from all of the smelters and refiners show that the production of refined lead in the United States in 1908 was 413,868 tons as compared to 426,146 tons in 1907. Details of the production are given in the accompanying table, which shows the production of Missouri and Kansas lead according to groups of smelters, the single smelter in Kansas being included in "Southwest Missouri," and the two smelters in Illinois, near St. Louis, in "Southeast Missouri." The lead that is entered as "desilverized" is the production of desilverizing refiners other than those included in the two groups above mentioned. The present conditions of lead smelting in the United States do not permit of any accurate, and at the same time rational, classification.

The above statistics include a considerable quantity of lead refined from scrap and old metal, which is reworked by several of the refiners whose principal business is the refining of base bullion of virgin origin. There is, moreover, a considerable production of lead by smelters who make a business of reworking old metal only. To some extent they market their product as pig lead, and to some extent they refine it as an alloy, which is marketed. It is difficult to draw the line where statistical enumeration should cease in this respect. In 1907 the production of such reclaimed pig lead was 9990 tons, while the production in the form of alloys was 15,508 tons.[1] In 1908 the production of pig lead from old material was 14,323 tons. It is probable that the reports of this industry are incomplete both for 1907 and 1908.

The statistics of lead of foreign origin as reported by the refiners do not agree exactly with the reports of imports by the Bureau of Statistics of the Department of Commerce and Labor, which in 1907 entered the importation of lead in ore and base bullion as 70,537 tons; and in 1908 as 109,286 tons. The discrepancy in 1907 is partly accounted for by the item of antimonial lead, which is recorded in the above table as wholly of domestic origin, whereas a large part of it was doubtless of foreign origin. Some discrepancy may also be explained by metal in transit, process of refining, and accumulated stocks.

The distribution of the domestic production of virgin lead is given as nearly as possible in the table on p. 563. The total for this table does not

[1] As reported by U. S. Geological Survey.

agree with that of the first table because (1) lead derived from scrap is excluded, and (2) the figures in this table are based to a large extent upon the reports of the producers of work lead, which obviously show the origin of the metal much better than the reports of the refiners can do. Differences between the reports of smelters and refiners are, of course, explained by metal in transit, treatment, etc.

The striking features of the statistics for 1908 are the further gain of Missouri as the premier lead-producing State and the decline in the production of nearly all of the other important lead-producing States, especially Colorado. It is probable that a large part of the lead that we have entered as "undistributed" (because we could not be sure of its source) belongs really to Colorado. The item entered under the caption of "Zinc Smelters" represents lead obtained from their residues, the origin of which it is next to impossible to trace.

METALLURGICAL PRODUCTION OF LEAD IN THE UNITED STATES.

(In tons of 2000 lb.)

Year.	Domestic Origin.						Foreign Origin.		Grand Total.
	Desilverised.	Antimonial	S. E. Mo.	S. W. Mo.	Miscel.	Total.	Desilverised	Antimonial	
1905	205,665	8,456	81,299	21,324	3,000	319,744	83,504	2,730	405,978
1906	220,095	7,434	100,492	16,528	980	345,529	67,441	2,686	415,656
1907	213,383	9,614	108,510	17,833	790	350,130	76,016	(a)	426,146
1908	174,650	13,109	113,103	18,014	*Nil*	318,876	94,992	(a)	413,868

(a) The production of antimonial lead, which in 1906 was divided according to domestic and foreign, in 1907 and 1908 has been wholly entered under domestic.

STATISTICS OF LEAD IN THE UNITED STATES.

(In tons of 2000 lb.)

Year.	Produced from Domestic Ores.				Imported in Ores and Bullion. (c)	Total Production and Imports.	Exported in all Forms.
	Desilverised.	Soft. (a)	Antimonial. (b)	Totals.			
1897	144,649	45,710	7,359	197,718	92,117	302,859	60,253
1898	169,364	50,468	8,643	228,475	89,209	348,845	78,168
1899	171,495	40,508	7,377	217,085	76,423	317,196	74,944
1900	221,278	47,923	9,906	279,107	114,397	425,824	100,288
1901	211,368	57,896	10,656	279,922	112,471	458,033	100,026
1902	199,615	70,424	10,485	280,524	107,715	458,456	82,228
1903	188,943	78,298	9,453	276,694	106,407	418,601	81,971
1904	200,858	90,470	10,876	302,204	112,852	415,056	84,142
1905	205,665	105,623	11,186	322,474	98,378	420,852	59,741
1906	220,095	118,000	10,120	348,215	84,134	432,349	47,323
1907	213,383	127,133	9,614	350,130	79,815	429,945	51,502
1908	174,650	131,117	13,109	318,876	109,286	428,162	76,328

(a) Since 1904 a large part of the so-called soft lead was desilverised, but this (being of Missouri origin) has been included in the old classification. (b) The entire production of antimonial lead is entered as of domestic production, although part of it is of foreign origin. (c) Includes "pigs, bars and old."

DELIVERY OF LEAD IN THE UNITED STATES.
(In tons of 2000 lb.)

	1905	1906	1907	1908
Supply:				
Production desilverized	289,169	287,536	289,399	269,642
Production soft lead	105,623	118,000	127,133	131,117
Production antimonial lead	11,186	10,120	9,614	13,109
Imports foreign refined lead	5,720	11,763	9,277	2,759
Stock, domestic lead, Jan. 1	10,000	4,000	4,000	(e)50,000
Foreign in bond, Jan. 1	11,481	8,148	5,691	12,897
Total supply	433,179	439,567	445,114	479,524
Deductions:				
Re-exports of foreign	58,631	47,223	51,424	76,328
Exports of domestic lead	63	74	55	Nil
Stock, domestic, Dec. 31	4,000	4,000	(e)50,000	(e)75,000
Foreign in bond, Dec. 31	8,148	5,691	12,897	18,462
Total deductions	70,842	56,988	114,376	169,790
Delivery	362,331	380,122	330,738	309,734

(e) Partly estimated.

The consumption of lead in the United States according to principal purposes is shown in an accompanying table, the statistics of which are compiled from reports received from the consumers. It is impossible to determine precisely the consumption in this manner, because there are so many small users of the metal from whom reports cannot be obtained. However, the reports of the larger consumers are strongly indicative of the real situation and of course furnish the best possible evidence as to the classification of the consumption. The reports that were received from the consumers footed up to 294,029 tons in 1907, and 257,759 tons in 1908, the same consumers being heard from in each year. Consequently, it appears that the consumption in 1908 was about 87½ per cent. of the consumption in 1907.

CONSUMPTION OF LEAD IN THE UNITED STATES.

Purpose.	1907		1908	
	Tons.	%	Tons.	%
White lead and oxides	115,000	34.8	117,500	40.5
Pipe	41,000	12.4	33,800	11.6
Sheet	21,500	6.5	16,400	5.7
Shot	28,000	8.5	31,600	10.9
Other purposes	125,238	37.9	90,700	31.3
Totals	330,738	100.0	290,000	100.0

A striking feature of these figures is the immense quantity of lead used for "other purposes," which makes it interesting to mention their nature. First in importance appears to be the manufacture of solder, babbitt metal, type metal, and other alloys. However, the lead

consumed for covering electrical cables is but little inferior in amount. A large tonnage is used for jointing cast-iron pipe. Cartridge manufacture also consumes a large amount. A surprisingly large amount is employed for coating iron and steel, *i.e.*, making terne plate. It is also surprising to find how much lead is bought by the manufacturers of brass, who introduce a small amount of lead in certain grades of that alloy. Even some of the galvanizers appear as purchasers of lead. There is a considerable consumption for minor purposes, such as car seals, yacht ballast, and weights of various kinds, the manufacture of plumbers' articles (other than pipe), etc. Finally a good deal of pig lead is bought by municipal water works for jointing cast-iron pipes, which is a form of consumption that it is difficult to report statistically.

PRODUCTION OF LEAD BY STATES.
(In tons of 2000 lb.)

State.	1902 (*a*)	1903 (*b*)	1904 (*b*)	1905 (*b*)	1906 (*c*)	1907 (*c*)	1908 (*c*)
Arizona	599	1,418	1,424	1,986	2,884	2,200	1,867
California	175	52	155	110	432	850	490
Colorado	51,833	43,276	49,290	(*f*)57,856	(*f*)52,992	47,332	26,707
Idaho	84,742	94,611	103,411	(*h*)107,000	(*i*)121,584	111,697	98,394
Kansas	(*n*)	(*n*)	(*n*)	(*n*)	(*n*)	1,800	2,400
Missouri	(*d*)79,445	86,439	92,119	(*g*)102,500	115,103	123,613	125,216
Montana	4,438	3,138	3,454	2,097	2,485	2,005	2,309
Nevada	1,269	2,125	1,779	2,096	1,669	3,400	3,676
New Mexico	741	582	1,295	1,170	640	1,900	611
Oklahoma	*nil*	.....	.....	.....	.....	400	1,000
Utah	53,914	48,573	53,647	42,746	56,280	54,738	43,995
Wisconsin	.....	.....	.....	.....	1,753	3,500	3,496
Other States	(*e*)3,641	(*k*)2,188	(*k*)630	(*k*)2,695	(*m*)943	1,204	600
Undistributed	.....	.....	.....	.....	.....	.....	2,026
Zinc Smelters	.....	.....	.....	.....	.....	1,320	1,290
Total	280,797	282,402	307,204	320,256	356,745	355,959	314,067

(*a*) Statistics of U. S. Geological Survey representing lead content of ore smelted. (*b*) U. S. Geological Survey figures giving production of "merchant lead." (*c*) Smelter s reports. (*d*) Includes production of Wisconsin, Illinois, Iowa, Virginia and Kentucky. (*e*) Includes production of Alaska, So. Dakota, Washington, Georgia, Tennessee and Texas. (*f*) Report of State Commissioner of Mines. (*g*) Includes 1500 tons from Iowa, Illinois and Wisconsin, but not the total production of those States. (*h*) Partly estimated. (*i*) Report of State Inspector of Mines less allowance of 5 per cent. for loss in smelting. (*k*) Includes production of Alaska, Oregon, So. Dakota, Washington, Georgia, Tennessee, Virginia, Kentucky and Texas. (*m*) Includes production of Illinois, Iowa, Kentucky, Tennessee and Washington. (*n*) Included with Missouri.

IMPORTS OF LEAD IN ORE, BASE BULLION, PIGS, BARS AND OLD. (*a*)
(In tons of 2000 lb.)

Source.	1901	1902	1903	1904	1905	1906	1907	1908 (*c*)
United Kingdom	201	396	776	247	795	4,926	217	(*b*)
Germany	336	476	705	366	125	1,003	228	(*b*)
Other Europe	1	671	226	83	59	1,961	3,461	(*b*)
Canada	26,065	9,732	9,600	8,952	8,182	9,257	6,663	644
Mexico	81,727	93,742	93,068	102,903	87,584	66,756	68,767	107,369
South America	4,109	2,690	1,948	290	1,577	158	442	(*b*)
Other Countries	32	6	83	11	56	74	63	1,273
Total	112,471	107,713	106,406	112,852	98,378	84,135	79,814	109,286

(*a*) Refined lead, i.e., in pigs, bars and old is a small part of the total. It was in 1901, 604 tons; in 1902, 2,529 tons; 1903, 3,023 tons; 1904, 8,724 tons; 1905, 5,720 tons; 1906, 11,763 tons; 1907, 9,277 tons; 1908, 2,759 tons. (*b*) Included in other countries. (*c*) 1908 figures do not include import of pigs, bars and old. With these the total import for year is 112,045 tons.

LEAD PRODUCTION OF THE WORLD.

Year.	Australasia.	Austria. (a)	Belgium.(a)	Canada. (a)	Chile. (a)	France. (a)	Germany. (a)	Greece. (a)	Hungary. (a)	Italy. (a)
1897..	22,000	9,860	17,023	17,698	370	9,916	118,881	16,468	2,527	22,407
1898..	67,000	10,340	19,330	14,477	13	10,920	132,742	19,193	2,305	24,543
1899..	87,600	9,736	15,700	9,917	171	15,981	129,225	19,059	2,166	20,543
1900..	87,100	10,650	16,365	28,648	14	15,210	121,513	16,396	2,030	23,673
1901..	90,000	10,161	18,760	23,537	455	21,000	123,098	17,644	2,029	25,796
1902..	90,000	11,264	19,504	10,411	99	18,817	140,331	14,048	2,243	26,494
1903..	141,446	12,162	22,263	8,226	*Nil*	23,258	145,319	12,361	2,057	22,126
1904..	118,979	12,645	23,470	17,241	17	18,800	137,580	15,186	2,104	23,475
1905..	(c)104,639	12,968	22,885	25,391	*Nil*	24,100	152,590	13,729	2,146	19,097
1906..	(d)93,000	14,846	23,765	24,580	*Nil*	25,614	150,741	12,308	1,925	21,268
1907..	(d)97,000	13,598	27,450	21,660	8	24,800	164,079	13,814	1,468	22,978
1908..	(d)119,009	(e)13,100	(d)30,100	20,747	.....	(e)25,000	169,418	(d)16,000	(e)1,500	(e)26,000

Year.	Japan. (a)	Mexico. (a)	Russia. (a)	Spain. (a)	Sweden. (a)	United Kingdom. (a)		United States.	Totals. (f)
						Foreign Ores.	Domestic Ore.		
1897.....	1,737	71,637	450	189,216	1,480	13,312	26,988	179,369	721,339
1898.....	1,705	71,442	241	198,392	1,559	23,239	25,761	207,271	830,473
1899.....	1,989	84,656	322	184,007	1,606	17,571	23,929	196,938	819,116
1900.....	1,877	63,827	221	172,530	1,424	10,738	24,762	253,204	849,182
1901.....	1,806	94,194	156	169,294	988	19,639	20,361	253,944	882,862
1902.....	1,644	106,805	225	177,560	842	9,118	17,987	254,682	902,089
1903.....	1,728	(b)94,181	106	175,109	678	14,900	20,278	256,138	952,336
1904.....	1,803	(c)103,000	90	185,862	589	6,888	20,155	278,634	966,518
1905.....	2,272	(b)101,196	700	185,693	576	7,517	20,977	290,472	986,948
1906.....	4,305	(b)73,699	907	185,470	753	6,984	22,691	323,567	986,423
1907.....	3,067	(b)76,158	(c)100	(d)185,800	813	10,880	24,850	322,854	1,011,377
1908.....	(d)3,000	(b)127,010	(c)100	(d)183,200	(c)800	(c)12,000	18,000	284,858	1,049,833

(a) From official reports of countries unless otherwise denoted. (b) Exports. (c) Commercial statistics of Julius Matton, London. (d) As reported by Metallgesellschaft, Frankfort am Main. (e) Estimated. (f) The totals may be high on account of duplications which can not be eliminated.

LEAD MINING IN THE UNITED STATES.

Arizona. (By William P. Blake.)—The silver-lead mines of Santa Cruz county remained in 1908 in practically the same situation as at the end of 1907. The Hardshell and Flux mines near Patagonia, noted for their large bodies of argentiferous lead ore, are still under bond. It is claimed that the Hardshell has 100,000 tons of ore developed that will assay 10 per cent. lead and 10 oz. silver per ton, while the Flux is claimed to have 50,000 tons assaying 12 per cent. lead and 6 to 8 oz. silver. The Mowry mine, south of Patagonia, was not actively worked in 1908, the smelter remaining closed. The Vekal mines, 35 miles southwest of Casa Grande, which were large producers of silver-lead ore from 1885 to 1904, were reopened. Operations at Tombstone were conducted continuously and more ore was extracted from the lower levels than ever before.

Colorado.—The production of lead in this State suffered a severe decline in 1908. Leadville continued to be the chief district. At that place, the Western Mining Company suspended operations in December,

1907, owing to the fall in metal prices and pulled out its pumps, but work was continued by lessees for several months until the rising water drove them out. The Moyer and Tucson mines of the Iron Silver company were idle throughout 1908. The Yak Tunnel properties, however, maintained a large output, and the heading is being extended to drain the Resurrection property. According to the Leadville *Herald Democrat*, the mines of Leadville in 1908 produced 33,127 tons of carbonate ore, 117,423 of oxidized iron ore, 162,188 of sulphide ore, 70,197 of zinc ore, 92,187 of silicious ore, and 1500 of manganese ore, a total of 476,622 tons. The metallic contents were 68,135 oz. of gold, 3,509,378 oz. of silver, 9005 tons of lead, 7,065,956 lb. of copper, and 37,173,000 lb. of zinc.

At Creede the output was fairly well maintained as to quantity, but the decrease in value was severely felt. In the San Juan, as indeed everywhere throughout Colorado, the mines producing lead and zinc depend mainly for their profit on the gold and silver in the ore, the lead and zinc being chiefly in the nature of by-products, although often very important ones. Unless there be a recovery in the price for silver, it is probable that 1909 will witness a further falling off in the output of this State.

LEAD PRODUCTION OF COLORADO. (a)
(In tons of 2000 lb.)

Country.	1900	1901	1902	1903	1904	1905	1906	1907	1908
Clear Creek..	2,497	1,945	1,641	1,726	1,981	1,631	1,439	1,832	1,205
Hinsdale.....	4,689	3,705	3,107	230	521	446	442	470	82
Lake.....	31,300	28,180	19,725	18,177	23,590	26,424	23,918	17,032	7,169
Mineral.....	7,476	5,260	4,646	4,300	6,673	5,940	7,443	6,490	4,119
Ouray.....	4,739	3,952	2,131	1,675	1,022	2,674	2,861	1,803	1,516
Pitkin.	13,726	16,375	12,487	16,635	9,441	10,987	8,781	6,957	3,713
San Juan....	8,789	7,736	3,850	3,485	4,644	3,223	2,070	6,213	5,133
Others.....	8,921	6,813	5,565	4,529	5,901	6,531	6,038	5,696	5,918
Total...	82,137	74,056	53,152	50,757	53,773	57,856	52,992	46,493	28,855

(a) As reported by the State Commissioner of Mines.

The lead-smelting works now in operation in Colorado are those of the Ohio & Colorado Smelting Company, at Salida, and the Globe, Pueblo, Arkansas Valley and Durango plants of the American Smelting and Refining Company. The Grant works, at Denver, and the Philadelphia plant, at Pueblo, are permanently abandoned. The chances are also that the Eilers plant, at Pueblo, now idle, will never run again.

Idaho. (By Robert N. Bell.)—Two of the principal producers in the Coeur d'Alene, viz., the Hecla and Morning mines, were closed during the first six months of 1908. The Last Chance mine was idle for two months. Operations were resumed at these properties in June. The Hercules, Standard-Mammoth and Bunker Hill & Sullivan were worked continuously. At the end of the year the producing mines of the district

were employing about 2000 men, while the smaller producers and prospecting enterprises had about 1000. The most noteworthy event of 1908 was the completion of the Idaho Northern railway up the North Fork of the Coeur d' Alene river, from Enaville to Murray, thus providing transportation facilities for the north side of the Coeur d' Alene. The line of the Chicago, Milwaukee & St. Paul Railroad down the St. Joe river, extending through the south border of the district, is approaching completion. This will aid in the development of a number of promising mines.

At Burke, the Tiger-Poorman mine, owned by the Federal company, after being developed to a depth of 2200 ft. by a vertical shaft was abandoned, the ore shoot at the bottom having so contracted in length and width that it was no longer profitable to work. At Mullan, the development of the new ore shoot discovered at a depth of 800 ft. in the Star mine was important. This is a new ore shoot on the Morning fissure, which traverses the full length of this property. It is 500 ft. long and from 5 to 15 ft. wide, the ore being of about the normal grade for this part of the district. This property will probably become a large producer. Other promising discoveries were the Alice mine in Ruddy gulch, the Cooney mine at Burke, the Ambergris mine at the head of Nine Mile creek, the Bear Top at Murray, and the Caledonia near Wardner. All of these can ship high-grade ore from bodies of considerable size.

The principal producers in 1908 were the Bunker Hill & Sullivan, Last Chance, Hercules, Hecla, Standard-Mammoth, Morning, and Gold Hunter. Their output was approximately 1,250,000 tons. In spite of the great depth that has been attained, the average yield of the ore is probably higher now than at any period in the history of the district.

An important economy introduced during 1908 at all the big mines was the addition of a picking plant near the portal of the main adit. There the ore is roughly sorted into shipping ore, concentrating ore and

SUMMARY OF REPORTS OF THE COEUR D'ALENE MINES FOR 1907.

	Tons Mined.	Gross Value.	Mining Costs.	Freight and Smelting.	Improvements.	Net Profit.	Net Loss.
Morning	297,500	$1,497,325	$680,132	$512,482	$70,835	$233,876	.....
Standard-Mammoth	287,300	3,193,320	1,085,131	865,828	38,345	1,204,016	.....
Tiger	51,800	252,784	252,625	73,669	32,518	.....	$106,028
Last Chance	163,600	1,242,048	569,824	387,009	28,372	256,843	.....
Bunker Hill & Sullivan	312,800	4,180,210	666,803	1,463,200	183,734	1,866,483	.....
Hercules	20,466	1,615,684	178,038	329,093	343,392	765,161	.....
Hecla	117,430	1,244,685	440,282	332,349	33,864	437,190	.....
Frisco	55,846	493,664	229,564	230,332	61,300	.....	27,532
Gold Hunter	49,453	172,617	141,712	43,000	63,439	.....	75,534
Tamarack & Chesapeake	277	10,349	16,574	2,747	15,858	.....	24 830
Success	7,262	211,884	54,132	81,168	35,781	40,803	....
Pittsburg Lead Co.	2,342	252,851	72,966	73,796	21,440	84,649	.....
Snowstorm	95,435	1,396,258	216,393	569,684	155,892	454,289	.....

clean waste. This has materially increased the extraction in the mills by reducing slime losses. Because of this sorting, the shipments of crude ore from the district in 1908 were much larger than in 1907.

FEDERAL MINING AND SMELTING COMPANY.

	1905	1906	1907	1908
Tons ore mined	664,830	374,332	888,950	599,850
Average lead (a)	6.64%	7.21%	6.72%	7.33%
Average silver, oz. (a)	4.05	4.43	4.15	4.68
Tons concentrate (b)	85,205	130,855	130,373	93,811
Ratio	7.8 : 1	6.7 : 1	6.8 : 1	6.4 : 1
Oz. silver	2,689,867	3,920,884	3,689,298	2,803,628
Average per ton	31.57	29.96	28.30	29.90
Tons lead	44,137	63,029	56,746	43,988
Average	51.8%	48.17%	45.83%	46.88%
Net profit	$1,242,698	$2,685,300	$2,532,250	$1,067,037
Dividends paid	1,098,896	1,647,457	1,917,741	928,917

(a) Average yield, not average assay. (b) Includes mill concentrate and shipping ore.

Iowa. (By James H. Lees.)—The high prices in 1907 greatly stimulated lead mining and much development work was done, but there was a recklessness in management that placed the industry in no condition to withstand the shock of financial stringency. Therefore, while there was some activity during the early weeks of 1908, nearly all the workings were closed before the middle of the year. The few concerns which kept up work did so rather desultorily and marketed but little of the ore raised. The amount of ore won during 1908 did not exceed 100 tons. The chief occurrences of ore in the Dubuque field are either thin sheets filling vertical cracks, or masses lining the abundant caves and crevices of the Galena dolomite or mixed with the loose dirt and rocks which fill these openings to a greater or less extent.

Missouri.—This State, which now holds the premier position in America, made an increased production in 1908, chiefly from the southeastern district, where Bonne Terre and Flat River are the centers. Developments in the Joplin district are treated especially under the caption of "Zinc" later in this volume. Its production of galena ore for a series of years is given in the accompanying table.

PRODUCTION OF LEAD ORE IN THE JOPLIN DISTRICT.

(Tons of 2000 lb.)

Year.	Tons.	Year.	Tons.	Year.	Tons.	Year.	Tons.	Year.	Tons.
1894	32,190	1897	30,105	1900	29,132	1903	28,656	1906	39,189
1895	31,294	1898	26,687	1901	35,177	1904	34,362	1907	39,619
1896	27,721	1899	23,888	1902	31,625	1905	31,679	1908	35,142

Lead ore brought $45 per ton the first week of 1908, $5 higher than the closing week of 1907. It advanced to $47, and then to $50, at

which price it stood for three weeks, then to $50.50, $52 and $52.50, closing February at $52. During the first two weeks of March it was $50 and the latter two weeks $52 per ton. In April it was $54 the first week and $55 the rest of the month. In May the price advanced from $55.50, to $56, to $57, to $59, and finally to $60.50, and in June it continued from $61.50 to $62.50, to $64, to $66. Then the decline in the price of pig lead began to decrease the lead-ore prices to $62 and $61, $60 for two weeks, then up a little again, $61, $63.50 two weeks, back to $61.50, and finally to $60 at the end of August. September opened at $59, declined first to $58, and then to $57, and closed at $55.50. In October the price was $54, $55, $54 and $56. It went up again to $58 for the first week and $60 for the second week of November, dropping back to $55 and $54 at the close. In December the price dropped to $53, then to $51 and during the third and the closing week to $50 per ton.

PRICE OF LEAD ORE AT JOPLIN.
(Per ton of 2000 lb.)

Year.	1899	1900	1901	1902	1903	1904	1905	1906	1907	1908
Highest	$55.00	$56.50	$47.50	$50.00	$60.50	$62.00	$80.00	$87.00	$88.50	$66.00
Average	51.34	48.32	45.99	46.10	54.12	54.80	62.12	77.78	68.90	55.75

WAGES PAID, COAL COST, AND PRICE RECEIVED FOR LEAD BY A COMPANY OF ST. FRANCOIS COUNTY, MO.[1]

Labor (wages per day, January 1).	1890	1891	1892	1893	1894	1895	1896	1897	1898	1899
Mine drillers, underground	$1.50	$1.60	$1.60	$1.60	$1.25	$1.15	$1.25	$1.25	$1.45	$1.45
Mine back hands, underground	1.15	1.25	1.25	1.25	1.00	.90	1.00	1.00	1.20	1.20
Mine rock loaders, underground	1.15	1.25	1.25	1.25	1.20	1.10	1.20	1.20	1.40	1.40
Mine laborers, underground	1.00	1.15	1.15	1.15	1.00	.90	1.00	1.00	1.10	1.10
Mill jigmen	1.50	1.60	1.60	1.60	1.50	1.35	1.50	1.50	1.75	1.75
Mill firemen	1.25	1.35	1.35	1.35	1.00	.90	1.00	1.00	1.20	1.20
Mill laborers	1.00	1.15	1.15	1.15	1.00	.90	1.00	1.00	1.10	1.10
Mill chatmen	1.20	1.30	1.30	1.30	1.25	1.15	1.25	1.25	1.50	1.50
Calciners	1.40	1.50	1.50	1.50	1.50	1.35	1.50	1.50	1.60	1.60
Blast furnace pot pullers	1.50	1.65	1.65	1.65	1.55	1.40	1.50	1.50	1.60	1.60
Coal (cost per ton at Bonne Terre, Mo., January)	2.30	2.30	2.30	2.25	1.90	1.96	2.00	1.80	1.74	1.97
Average price received for lead at East St. Louis during fiscal years ending April 30	3.55	4.00	3.87	3.88	3.15	3.01	2.91	2.77	3.48	3.82

Labor (wages per day January 1).	1900	1901	1902	1903	1904	1905	1906	1907	1908
Mine drillers, underground	$1.60	$1.60	$1.60	$1.60	$1.80	$1.90	$1.90	$2.25	$1.70
Mine back hands, underground	1.30	1.30	1.30	1.30	1.50	1.60	1.60	1.95	1.45
Mine rock loaders, underground	1.55	1.55	1.55	1.55	1.75	1.90	1.90	2.25	1.70
Mine laborers	1.25	1.25	1.30	1.30	1.50	1.50	1.50	1.85	1.35
Mill jigmen	1.90	1.90	1.95	1.95	1.95	1.95	1.95	2.30	1.95
Mill firemen	1.30	1.30	1.35	1.35	1.65	1.65	1.65	2.05	1.65
Mill laborers	1.25	1.25	1.30	1.30	1.30	1.30	1.30	1.55	1.30
Mill chatmen	1.65	1.65	1.70	1.70	1.70	1.70	1.70	2.15	1.80
Calciners	1.60	1.60	1.60	1.70	2.00	2.00	2.00	2.25	2.90
Blast furnace pot pullers	1.60	1.60	1.60	1.70	1.80	1.80	1.80	2.05	1.60
Coal (cost per ton at Bonne Terre, Mo., January)	1.97	2.00	2.15	2.15	2.59	2.06	1.94	2.13	2.18
Average price received for lead at East St. Louis during fiscal years ending April 30	4.48	4.15	4.12	4.02	4.20	4.32	5.00	5.72	4.15

[1] These figures were submitted in a hearing before the Ways and Means Committee in November, 1908, and evidently refer to the St. Joseph Lead Company.

(By H. A. Wheeler.)—The output of the southeastern lead belt of Missouri closely approximated 104,000 tons in 1908. This is high-water mark as regards tonnage, and in view of the discouraging condition of the market at the beginning of the year, this record is very satisfactory. The panic in the latter part of 1907 enabled the rapidly rising wages to be lowered to a more reasonable basis. When the lead market unexpectedly showed such healthy improvement in the spring of 1908, the larger producers resumed on full-time, and so continued to the close of the year.

The increase in the output resulted from improvements largely made in 1907, as in 1908 no new work was attempted until the latter part of the year. No new orebodies were discovered, nor were any new companies floated in 1908. As the market maintained its improved condition, it encouraged the starting of betterments in the latter part of 1908 that should increase production considerably in 1909, if the present bright outlook continues.

The output, as usual, was almost entirely derived from the five large producers in the Bonne Terre and Flat River camps in St. Francois county, especially as the depressed market was felt much more severely by the smaller producers in Madison, Washington, Jefferson and Franklin counties. The decreased tonnage in the latter, however, was not sufficient to affect materially the results of the district. The output of Madison county, which includes the Mine La Motte and Fredericktown camps, amounted only to about 4 per cent. of the total, while the production from the old shallow diggings of Washington, Jefferson and Franklin counties was less than 1 per cent. The unusually small output of the latter is not due to exhaustion, but rather to the sensitiveness of the mines in that area to a low market and to their slowness in recovering from the benumbing effects of the panic.

The searching for new orebodies with the diamond drill, that was so conspicuous in 1906 and 1907, has been taken up again on a large scale by only one of the large companies. Some diamond drilling was done by the other companies in 1908, but on a relatively small scale and mainly to define and extend the orebodies now being worked. The steady absorption by the large companies of mineral-bearing lands in the lead belt that had been in progress until the panic has not yet been renewed; but the few good tracts remaining are likely to be taken over as the times improve. With the steadily growing outputs and increased size of the mills, the present orebodies are being rapidly exhausted and foresight requires the accumulation of virgin properties to supply future ore reserves, especially while the prices are still so moderate. Prices that today look high at $100 to $300 per acre as compared with prices of $50 to $100 an acre that ruled scarcely 10 years ago, will probably look extremely cheap 10 years from now.

An important innovation was introduced in this district in 1908 by the starting of an electric-power plant in which gas engines supplied by producers are the prime-movers. Although the district, being only about 60 miles from the Illinois coalfields, enjoys probably the cheapest coal of any metal-mining district in the country, two 2400-h.p. plants were erected to reduce still further the fuel costs by utilizing the much greater economy of gas engines; they replaced compound condensing Corliss steam engines. Since nearby bituminous coals can be laid down at the metal mines for $1.75 to $2.50 per ton, the margin for economy is not large, especially if higher-grade coals have to be shipped in to obtain efficient work from the producers. Local experience in operating gas producers with western Illinois coals has shown that they are so high in sulphur and ash that it is more profitable to use the higher-grade and more expensive coal from Indiana or southern Illinois.

The St. Joseph Lead Company still holds its prestige as the largest, most important and most profitable of the Missouri producers. It is still producing heavily from its No. 1 mine, which is now more than 40 years old, and its latest shaft, the No. 13, was recently sunk to open an extension of its famous Bonne Terre orebody. While this company reduced its output 50 per cent. and entirely stopped extensive improvements in the latter part of 1907, the mines were reopened on full time early in 1908. The company is operating eight shafts. These supply the old or No. 1 mill at Bonne Terre, having a capacity of 1500 tons, and the new or No. 2 mill at Owl creek, having a capacity of 1200 tons.

The Doe Run Lead Company has completed the installation of a gas-engine power plant at its No. 1 shaft at Flat River that is a duplicate of the St. Joseph company's plant. It has resumed the construction of the concentrator at West Elvins, on which work was stopped by the panic; this mill will have a capacity of about 1500 tons and is expected to be in operation by July 1, 1909. Production was cut down 50 per cent. in the latter part of 1907, but work at full capacity was resumed in the early part of 1908. Four shafts are being operated, from which the ore is shipped 10 miles south to the 1500-ton mill at Doe Run.

The Federal Lead Company is now making the largest production in its history. Its new No. 2 mill at Central is being remodeled by taking out the three-compartment Harz jigs and replacing them with Hancock jigs. This is resulting in a marked saving in labor, power and water, besides materially increasing the capacity. When other improvements are completed it is expected that the No. 2 mill will have a capacity of 3500 to 4000 tons per day. The company is also operating its No. 1 mill, which has a capacity of about 1000 tons per day. A new shaft, the No. 11, is being sunk on the south side of the No. 2 mill to tap the old Central

orebody, and work is shortly to be started on Nos. 9 and 10 shafts, the sinking of which was discontinued in the latter part of 1907. Six shafts are being operated at present, but before the close of 1909, nine will probably be supplying ore.

The National Lead Company made an excellent record in 1908. Equipped as it is with one of the best and most compact plants in the district, it was probably the best money-maker during 1908, as it never curtailed its output and it was under no expense for improvements. Four shafts supply the 1500-ton mill that are only ¼ to 1½ miles distant from the mill.

The Desloge Lead Company maintained its usual output and did not attempt materially to curtail its production during the depression. The new No. 5 shaft has been completed, and while it is not so conveniently situated, being about four miles distant from the mill, it promises to prove a valuable producer. The new orebody opened by this shaft lies almost over the line in Washington county. The Flintshire furnaces were operated during the year on the "Desloge" brand of pig lead, but as the plant is so small that it can smelt only a portion of the output, most of the concentrates were shipped to the Illinois smelters. The company's 1000-ton mill is supplied from three shafts that are ½ to 4 miles distant from it.

The plant of the Mine La Motte company was increased by a new 500-ton mill. The output was not pushed in 1908, owing to the low market, but the reserves were increased and the property developed. The lease of the old copper-mine tract that was sub-let by the Mine La Motte company to the Hudson Valley Mining Company has been transferred to the Eastern Lead Company, which operated most of the year on a royalty basis.

The North American Lead Company at Fredericktown continues to be operated as a copper property and the small amount of lead that it recovers in working the copper sulphides is a by-product that is shipped away to be smelted.

The Madison Lead and Land Company, which operates the old Catherine mine near Fredericktown, restricted its output during 1908 and ran only one of its two mills. It is now making several improvements and is preparing to increase its output.

New Mexico. (By Reinold V. Smith.)—Donna Ana, Sierra, and Luna counties were the most active in the production of lead in 1908. The largest individual producer was the Stephenson Bennett Consolidated Mining Company, operating at Organ, in Donna Ana county. The company is producing ore at the rate of 300 tons per day. Shipments went regularly forward from the mines in the upper Magdalenas, all carbonate

ore derived from development work. The Ocean Wave mill at Hermosa was built during the year.

Nevada.—A large part of the lead production of this State in 1908 came from the Richmond-Eureka mines at Eureka, which were actively operated. At Ward, White Pine county, the Nevada United shipped a good deal of ore containing 40 per cent. lead and 6 oz. silver per ton. The Nevada-Utah Mining Company, of Pioche, Lincoln county, reopened the Raymond & Ely and the Meadow Valley mines and in 1909 should figure as an important producer of lead.

Utah.—Lead mining in this State suffered severely in 1908. Park City was still crippled by the cave in the Ontario drain tunnel that occurred three years ago, but during the summer the tunnel was permanently reopened and the district then took on new life. Most of the mines of this district were in operation, but their production was generally smaller than in 1907. The Tintic and Bingham districts felt keenly the lower prices for silver and lead.

Lead Mining in Foreign Countries.

Australia.—The production of pig lead in 1908 was 119,000 tons, of which 86,000 tons was refined metal, while 33,000 tons were shipped as base bullion. The Australian consumption of lead was 7600 tons. As in previous years, the great bulk of the Australian output was derived from Broken Hill, New South Wales.

PIG LEAD PRODUCTION OF AUSTRALIAN SMELTING WORKS. (a)
(In tons of 2240 lb.)

Works.	1903	1904	1905	1906	1907
Broken Hill Proprietary Co	61,375	68,513	67,062	55,892	54,168
Sulphide Corporation	15,680	23,094	22,246	21,033	22,394
Smg. and Ref. Co. of Australia	7,500	10,599	502	.....	.....
Tasmanian Smelting Co	6,800	7,800	9,000	8,900	10,100
Fremantle Smelter	250	5,053	2,104	2,850	400
Queensland	3,795	2,046	2,422	2,461	4,000
Totals	95,400	117,105	103,336	91,141	91,062

(a) As reported by Julius Matton of London.

LEAD PRODUCTION OF NEW SOUTH WALES. (a)
(In tons of 2240 lb.)

Lead.	1903	1904	1905	1906	1907	1908
Base bullion	92,293	106,038	93,182	79,925	79,870	103,371
In ore exported	29,706	59,507	69,044	58,683	111,830	69,501
Totals	121,999	165,545	162,226	138,608	191,700	172,872

(a) According to the official statistics of New South Wales.

Canada.—The production of lead in 1908 was 22,863 tons, compared to 23,869 tons in 1907, all being derived from British Columbia. The amount paid as bounty on lead production in 1908 was $139,064. The exports of lead in ore in 1908 were 2256 tons and of pig lead 6971 tons, a total of 9227 tons.

As in previous years, the bulk of the production in 1908 was derived from the East Kootenay, the output of which in 1908 was 15,100 tons, nearly all coming from the St. Eugene mine. The production of the latter was 23,300 tons of concentrate and 2400 tons of ore shipped crude. Up to June 30, 1908, this mine had produced 684,311 tons of ore yielding 138,878 tons of concentrate, containing 3,951,141 oz. of silver and 81,450 tons of lead. According to the report of the Consolidated Mining and Smelting Company of Canada, Ltd., the owner of this mine, its ore reserves at the end of the last fiscal year were nearly 50 per cent. greater than a year previous and discoveries of higher-grade ore have been made.

The lead bounty act of 1903 expired on June 30, 1908. The amount of bounty paid by the Dominion Government was $712,000, leaving an unexpended balance of $1,788,000 from the $2,500,000 originally appropriated. The bounty has been continued to June 30, 1913, but the new act differs from the old in that the full rate of bounty (75c. per 100 lb. of lead) is payable until the price of lead at London reaches £14½ per 2240 lb. instead of £12½ as under the old act.

The smeltery of the Hall Mining and Smelting Company at Nelson was idle throughout 1908. That of the Sullivan Group Mining Company at Marysville, East Kootenay, was closed in February and was idle during the rest of the year. The Consolidated Mining and Smelting Company of Canada increased both the smelting and refining capacities of its works at Trail, adding two more Heberlein roasting furnaces (making six in all), and increasing the capacity of its electrolytic refinery to about 75 tons of refined lead per day.

China.—This country is a small exporter of lead ore. In 1907 shipments, originating in Hunan, from Hankow amounted to 1949 tons, nearly all of which went to Antwerp.

(By T. T. Read.)—Lead ores are known to occur in eight of the 18 provinces, but production of the metal seems to be entirely confined to the southwestern part of the Empire. About 350 tons of the metal passed through the customs at I-chang, on the Yangtze, in 1907. Probably this came from Ssu-chuan, but may have been derived from Kuei-chou. In 1907 some 3000 tons of lead ore came down the Yuen and Hsiang rivers, probably from points in Kuei-chou. In 1908 the production mysteriously fell off to almost nothing. It may be that this

is due to the customs classifying in 1908 as zinc ore that which was reported in 1907 as lead ore. The ore is chiefly exported to Europe.

England.—The lead ore of Derbyshire occurs in connection with fluorspar, and inasmuch as the deposits are not rich the success of the lead mining depends largely upon the fluorspar obtained as a by-product. The low price for lead in 1908 caused the mining record of the year to be unsatisfactory. Practically the same conditions exist in Durham, where the Weardale Lead Company is not only the largest producer of lead, but also is an important factor in the production of fluorspar. Some attempts were made to reopen the old mines of the Lake district and the Coldbeck Fell ranges. The Thornthwaite mines increased their output, owing to the better price received for the blende associated with the galena. The Force Cragg mines, formerly worked chiefly for barytes, have been acquired by the Cumberland Mining Company, Ltd., which has put in a new plant to deal with galena and blende.

A correspondent in Wales, who operates an important lead and zinc mine, communicates to us the prices received for its sulphide ores as an indication of the rates on these products in Wales in September, 1908. The prices given are for the ores f.o.b. cars, the buyer paying the freight, and paying cash for the ore in six weeks.

Galena.—Net lead value, fire assay, is computed at the average price of foreign lead for three weeks after the sale, plus the full silver value (average of the London price for fine silver for three weeks after the sale). Smelting charge, £3 5s. per ton (2240 lb.). The freight paid by the smelters is approximately 14s. per ton. The smelters state that after the expiration of this contract (in March, 1909), they must have a smelting charge of £3 10s., instead of £3 5s. The ore averages 82 per cent. lead and 10 oz. silver per ton.

Germany.—Lucius W. Mayer in *Eng. and Min. Journ.* of July 25, 1908, communicated the following data as to lead production at Mechernich: There are five mines in operation, three having both open-cast and underground workings, and the other two underground only. About 1600 men are employed, of whom 555 are occupied underground, mining 2000 tons of ore per day, or 3.58 tons per man. Wages average about four marks for eight hours' work underground and for 10 hours on the surface. The output of the Mechernich mines has been greatly reduced; formerly they turned out 3500 tons per day. In the open-cast mining about three tons of over-burden are removed per ton of ore won. At depth of 450 ft. open-cast operations cease. The ore-bearing seams vary from 4 to 20 m. in thickness. The ore averages between 2 and 3 per cent. lead. The pig lead obtained from it carries about 13 grams of silver per 100 kg. The lead is disseminated as fine

crystals of galena in the mineral-bearing sandstone, some of the galena being so finely disseminated as to be invisible. On this account the loss in milling is large, the average extraction being given as 66 per cent. The cost of mining is stated as 8 to 10 marks per ton of ore delivered at the top of the shaft. The total cost of producing lead is said to be 24 to 25 marks per 100 kg., equivalent to about 3c. per lb. of lead.

GERMAN IMPORTS AND EXPORTS OF LEAD.

(In metric tons.)

Year.	Metal.						Ore.					
	1903	1904	1905	1906	1907	1908	1903	1904	1905	1906	1907	1908
Imports.....	52,440	61,388	78,528	71,040	75,200	77,649	67,573	83,807	92,667	89,979	137,861	133,597
Exports.....	30,243	23,169	32,515	27,039	38,259	40,626	1,270	1,312	1,496	1,916	1,296	1,189

Mexico. (By H. A. Horsfall.)—The production of lead in Mexico increased largely in 1908. The official statistics show that the exportation of base bullion in 1908 was 127,010 tons, compared to 76,158 tons in 1907 and 73,699 tons in 1906. In each year the exportation of lead ore was insignificant. The chief producing districts in 1908 were Mapimi, Santa Eulalia and Sierra Mojada. The mines of the last district did not produce at full capacity, but before the end of 1908 most of them that were idle had resumed shipping.

Russia.—A recent letter from Chaborovsk to the *Novoe Vremya* says that on the Tyutyuché river and bay a second small gage railway is being constructed for the transport of silver lead ore from the mines to the sea shore. This ore is being exported to Germany.

Servia.—A Swedish company has acquired a concession for working silver-lead ore and an accumulation of lead slag at Babe, about 25 miles from Belgrade, and has erected a blast furnace to resmelt the slag, which yields about 6 per cent. lead and 1 oz. silver per ton. About six to seven tons of lead per day were being produced toward the end of 1908. The quantity of slag is estimated at 900,000 tons.

Spain.—The lead from the Almeria mines is treated in the provincial works, Berja, Palomares; Mazarron, etc. The price paid by smelters for ore delivered at their works is that quoted by the *Gaceta de Cartagena,* less four units and one peseta per quintal of 46 kg., the ore having to contain ½ oz. of silver per quintal of lead. The value of the excess or deficit of this percentage of silver is added to or deducted from the price stated. In spite of the high prices and export duties of 10 pesetas for argentiferous lead, some French smelters and merchants buy supplies, chiefly at Garrucha and Adra.

Tunisia.—A lead smelting works is to be built at Megrine, near Tunis, by a new French company, the Société Métallurgique de Mégrine. Tunisia produced 23,000 tons of lead ore in 1905, and many deposits are known to exist besides those mined. At present all the ore is exported.

Turkey.—An important deposit of lead near Tchantepe mountain, about nine hours by foot from the port of Kara-Bogha on the Sea of Marmora, is to be worked. The port has a wharf to which steamers can moor. There is a fairly good wagon road to the mines, over which the cost of transportation is about $2.50 per 2000 lb. The ore is of high grade in lead and the mine is said to be capable of yielding 50 to 60 tons of ore per day.

The Lead Markets in 1908.

New York.—An upward movement, culminating in August, and a subsequent downward movement characterize what was an unusually erratic market. Under the impetus of an improvement in the general situation, heavy buying on the part of speculators took place in the first months of the year, reinforced by purchases on the part of consumers in anticipation of a better business. While the output in Missouri assumed normal proportions as early as January, production in the West continued heavily curtailed until the spring months, and this fact helped greatly in the distribution of the large stocks which had accumulated and the advance in the price which was almost uninterrupted until August.

The business of manufacturers was more or less of a dragging nature throughout the entire year, with the exception of the white-lead branch, where up to November the requirements were of large volume. An equilibrium was reached in the middle of the year. At that time the resumption of Western mining operations on a normal scale made itself felt in the available quantity of marketable lead, but when the white-lead manufacturers ceased their purchases on a heavy scale, upon entering their dull season at the beginning of fall, the rate of offerings again overtook the requirements, and a rapid decline ensued. Prices at the close of the year again reached a level where they might be called cheap from a general point of view.

At the beginning of 1908 the market stood at 3.55@3.60c., New York, and a gradual and slow advance established prices at 3.70@3.75c. at the close of the month. This level was maintained well into March. The surplus which was left over from 1907 had been largely distributed by that time, and the market was depending upon the actual lead output which was restricted by the reduced activity in Western mining centers.

A realization of this brought consumers and speculators into the market, and in consequence a sharp advance took place to 3.90@4c., New York. There was not much change from these figures until late in April. The spring season is always the busiest for the lead manufacturerers, and the resulting influx of increased business brought them into the market for correspondingly larger quantities. Under the influence of this, prices reached 4.05@4.10c. toward the end of April.

The better outlook encouraged resumption of mining operations in the West on a normal scale, but the heavier production did not reach the market until several months later, and meanwhile the shortage of supplies facilitated the forward movement, a steady appreciation taking place throughout May and June, when sales on a large scale were effected at 4.47½@4.52½c. New York. The most urgent demand being satisfied, the market reacted in July to 4.40@4.45c. New York. There was a rally from this level during August, due to speculative influences, which caused an advance to 4.57½@4.62½c., New York, the highest point reached during 1908. This movement was short-lived. The weight of offerings became heavier during September, owing to the increased supplies turned out by the Western mines, which by that time had passed through the refining works.

The last three months of the year recorded, with one short interruption in November, a retrograde movement, the close being 4.15@4.20, New York.

AVERAGE MONTHLY PRICE OF LEAD PER POUND IN NEW YORK.

Year.	Jan.	Feb.	Mar.	Apr.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
	Cts.	*Cts.*	*Cts.*	*Cts.*	*Cts.*	*Cts.*	*Cts.*	*Cts.*	*Cts.*	*Cts.*	*Cts.*	*Cts.*	*Cts.*
1897....	3.04	3.28	3.41	3.32	3.26	3.33	3.72	3.84	4.30	4.00	2.96	3.70	3.58
1898....	3.65	3.71	3.72	3.63	3.64	3.82	3.95	4.00	3.99	3.78	3.76	3.76	3.78
1899....	4.18	4.49	4.37	4.31	4.44	4.43	4.52	4.57	4.58	4.58	3.70	4.64	4.47
1900....	4.68	4.68	4.68	4.68	4.18	3.90	4.03	4.25	4.35	4.35	4.58	4.35	4.37
1901....	4.35	4.35	4.35	4.35	4.35	4.35	4.35	4.35	4.35	4.35	4.35	4.15	4.33
1902....	4.00	4.075	4.075	4.075	4.075	4.075	4.075	4.075	4.075	4.075	4.075	4.075	4.069
1903....	4.075	4.075	4.442	4.567	4.325	4.210	4.075	4.075	4.243	4.375	4.218	4.162	4.237
1904....	4.347	4.375	4.475	4.475	4.423	4.196	4.192	4.111	4.200	4.200	4.200	4.600	4.309
1905....	4.552	4.450	4.470	4.500	4.500	4.500	4.524	4.665	4.850	4.850	5.200	5.422	4.707
1906....	5.600	5.464	5.350	5.404	5.685	5.750	5.750	5.750	5.750	5.750	5.750	5.900	5.657
1907....	6.000	6.000	6.000	6.000	6.000	5.760	5.288	5 250	4.813	4.750	4.376	3.658	5.325
1908....	3.691	3.725	3.838	3.993	4.253	4.466	4.477	4.580	4.515	4.351	4.330	4.213	4.2

London.—The market in January opened at £14¾, declined to £14, rose to £15¼ and closed at £14⅞. In February the market declined to £13 11/16, closing at £13⅞. In March the market hardened, rising to £14⅝ on the 25th, the closing price for the month being £14¼. In April there was a gradual and almost uninterrupted decline, £13 being accepted on the 28th. The price at the close was £13¼. A little strength developed toward the middle of May, when 13⅜ was paid for futures, but

pressure to sell caused a decline to set in on May 21, and in one case as low as £12¾ was accepted. At the close of the month the quotation was £12⅞@13¼. In June there was a recession to £12⅝, at which figure a fair volume of business took place, the closing being £12 11/16. A slow but steady improvement occurred in July, the close for that month being £13⅜. The improvement continued into August, as high as £13¾ being paid, but the close for the month was £13¼@13 11/16. September was uneventful, prices hovering between £13¼ and £13 1/16 until the very end of the month when £13⅜ was paid. October was also uneventful. The price rose as high as £13¾, the closing being £13½. November opened with a strong market, stimulated by the labor troubles at Broken Hill. As high as £14¼ was paid on Nov. 11 for forward delivery, but consumptive demand proving disappointing the market sagged off and the close was £13⅜@13½. Fluctuations in December were comprised within a narrow range, but on the whole the tone of the market was weak, the closing price being £13.

AVERAGE MONTHLY PRICE PER 2240 lb. OF LEAD AT LONDON. (a)
(In pounds sterling.)

Year.	Jan.	Feb.	Mar.	Apr.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
1897.....	11.717	11.708	11.562	11.787	11.837	11.912	12.250	12.679	13.650	13.575	13.100	12.600	12.367
1898.....	12.508	12.362	12.650	13.062	13.700	13.437	12.950	12.800	12.800	13.050	13.412	13.100	12.983
1899.....	13.375	14.350	14.150	14.375	14.146	14.283	14.385	14.733	15.267	16.179	17.096	16.883	14.933
1900.....	16.296	16.542	16.612	16.733	16.900	17.225	17.533	17.633	17.667	17.596	17.229	16.233	16.987
1901.....	15.925	14.667	13.379	12.421	12.275	12.342	12.150	11.692	11.954	11.600	11.267	10.533	12.521
1902.....	10.567	11.617	11.508	11.596	11.600	11.271	11.233	11.121	10.892	10.746	10.717	10.754	11.262
1903.....	11.304	11.708	13.225	12.404	11.800	11.437	11.383	11.146	11.167	11.108	11.108	11.179	11.579
1904.....	11.558	11.592	12.037	12.254	11.754	11.521	11.667	11.737	11.787	12.187	12.892	12.775	11.983
1905.....	12.875	12.462	12.296	12.658	12.762	13.000	13.608	13.958	13.950	14.679	15.337	17.050	13.719
1906.....	16.850	16.031	15.922	15.959	16.725	16.813	16.525	17.109	18.266	19.350	19.281	19.609	17.370
1907.....	19.828	19.531	19.703	19.975	19.698	20.188	20.350	19.063	19.775	18.531	17.281	14.500	19.034
1908.....	14.469	14.250	13.975	13.469	12.938	12.600	13.000	13.375	13.125	13.375	13.538	13.156	13.439

(a) The statistics for 1897-1905 are from the report of the Metallgesellschaft, Frankfurt am Main. Those for subsequent years are from the *Engineering and Mining Journal*.

White Lead and Oxides in 1908.

There was an active demand for white lead during the early months of 1908, due in part to the necessities of paint grinders and other large consumers and distributors, who had allowed their stocks to run down under the depression of the latter part of 1907 to a point that gave them no surplus for the spring business. The consumption of paints during the first half of 1908 was, however, not all that had been hoped for. There was no general revival of the building industry during the year, and the last quarter was distinctly disappointing because business was not greater than in the corresponding months of 1907, when the depression was at its worst. In spite of these facts, however, sales of white lead showed an increase over 1907, and this seems to have been an excep-

tional feature of the paint industry. The reason for this is chiefly the proportionately large demand for pure white lead or high-grade paints due to the educational work of the paint laws enacted in several States and the wide discussion that has attended their adoption.

Lead Carbonate.—In spite of advancing prices for pig lead, consumers of the carbonate were given the benefit of the low-cost metal bought by corroders during the latter part of 1907, and there was no change in quotations from the list established in December, 1907, until the latter part of June, when an advance of ¼c. was made. This advance was maintained on lead in oil until Dec. 10, 1908, when there was a reduction of ¼c. and the year closed at 6½@6¾c. This is the base price for lead in kegs of 100 lb. and upward, with an advance of ¼c. on 25- and 50-lb. kegs, and ½c. on 12½-lb. kegs. This differential is necessitated by the custom, adopted by all corroders during 1908, of making all packages full net weight instead of including the weight of the wood in the invoiced weight as was the previous custom on sizes under 100 lb. Where the cost of the package bears so large a proportionate relation to the value of the contents it has to be charged for in some form, and as the newly enacted paint laws generally require that packages shall be branded at actual net weights, the added price is necessary. It is generally expected that in the near future a further innovation in the trade in paint in oil, will be the adoption of steel packages in place of wood, for all sizes from 100 lb. down. The high cost of cooperage makes this a change in the direction of economy for the corroder, and from the standpoint of the consumer the metal package has many advantages.

The last reduction in the price of white lead applied to lead in oil only, and was due chiefly to the wide margin that had existed during all the year, and especially during the fall months, between the price of dry lead and that of lead in oil. This margin was sufficient to give paint grinders an exceptionally good profit, even though their dry lead averaged them a little above the previous year's cost.

As a matter of fact, quotations on dry lead were largely nominal throughout 1908, and the bulk of the sales were even below the nominally quoted price for carload lots. During the first half of 1908 the price generally quoted was 5⅝c., and this was fairly well adhered to as a price to grinders, although sales at 5½c. were by no means exceptional. The same nominal quotation prevailed during the last six months, but there were more sales at 5⅜c. than at the higher fraction, and the largest corroding interests entered into heavy contracts with grinders, and other large consumers during October at 5¼c. Subsequently, however, the business was mainly at 5⅛c.

Lead Oxides.—Lead oxides did not have as satisfactory a year as did the carbonate. While there was some improvement in demand as the year progressed, many of the large consumers of red lead were practically out of the market, due to a lack of structural work, and in some other industries litharge was substituted for red lead because of an unusually wide difference in cost. Prices for both oxides, and especially litharge, were unsatisfactory to manufacturers, since some of the makers, who have recently become active in the market, freely offered their product at narrower margins above the cost of metallic lead than had almost ever before prevailed. As a result published quotations were wholly nominal, and the bulk of the business in red lead was at 6¼@6½c.; these figures, of course, represent the prices to large consumers, and in casks. Through the ordinary channels of distribution prices ranged from 6½@7¼c., as to quantity and size of package, for both oxides.

The year closed with an evident expectation of a good spring business in all forms of high-class paints, but with marked conservatism on the part of large buyers as to present purchases, based upon a lack of confidence in prices. The buyers of all lead products now watch closely the

PRODUCTION OF LEAD PIGMENTS IN THE UNITED STATES.

Year.	Red Lead.		White Lead. (a)		Litharge.		Orange Mineral.	
	Short Tons.	Value.	Short Tons	Value.	Short Tons.	Value.	Short Tons.	Value.
1897.....	7,798	$744,709	105,904	$9,522,360	8,591	$773,190	477	$76,320
1898.....	9,160	916,000	93,172	9,391,738	7,460	710,192	541	108,200
1899.....	10,199	1,070,895	103,466	10,812,197	10,020	1,032,060	928	139,200
1900.....	10,098	1,050,192	96,408	9,910,742	10,462	1,067,124	825	100,650
1901.....	13,103	1,448,550	100,787	11,252,053	9,460	979,586	1,087	224,667
1902.....	11,669	1,262,712	114,658	11,978,172	12,755	1,299,443	867	138,349
1903.....	12,300	1,385,900	112,700	12,228,024	12,400	1,326,800	1,000	168,000
1904.....	13,938	1,672,569	126,336	13,896,913	12,487	1,248,691	1,125	168,681
1905.....	16,269	1,919,767	122,398	12,068,443	12,643	1,422,616	1,000	120,000
1906.....	13,693	1,874,448	123,640	15,234,990	13,816	1,890,050	2,927	421,488
1907.....	13,270	1,778,717	111,409	12,254,297	14,769	1,624,553	815	123,917
1908.....	11,358	1,156,282	116,628	10,515,315	12,254	1,231,206	393	43,157

(a) The output of "sublimed white lead," a mixed sulphate and oxide of lead, is not included in 1904-08.

IMPORTS OF LEAD PIGMENTS INTO THE UNITED STATES.

Year.	Red Lead.		White Lead.		Litharge.		Orange Mineral.	
	Pounds.	Value	Pounds.	Value.	Pounds.	Value.	Pounds.	Value.
1896.....	1,543,262	$47,450	1,183,538	$52,400	51,050	$1,615	1,359,651	$51,077
1897.....	1,386,070	46,992	1,101,929	48,988	60,984	1,931	1,480,042	67,549
1898.....	682 449	25,780	506,739	24,334	56,417	2,021	795,116	37,745
1899.....	776,197	30,479	583,409	30,212	55,127	3,614	1,141,387	58,142
1900.....	549,551	25,532	456,872	28,336	77,314	2,852	1,068,793	61,485
1901.....	485,466	19,369	384,671	21,226	49,306	1,873	977,644	52 409
1902.....	1,075,839	37,833	506,423	25,320	88,115	2,908	997,494	49,060
1903.....	1,152,715	40,846	453,284	24,595	42,756	1,464	756,742	36,407
1904.....	836,077	30,115	587,383	33,788	44,541	1,500	766,469	37,178
1905.....	704,402	26,553	597,510	34,722	117,759	4,139	628,003	31,106
1906.....	1,093,619	50,741	647,636	41,233	87,230	3,737	770,342	42,519
1907.....	679,171	35,959	584,309	37,482	90,475	4,386	615,015	37,799
1908.....	645,073	28,155	540,311	30,451	96,184	3,327	485,407	26,645

published quotations for pig lead and base their ideas of the value of the carbonate and oxides thereon; and while manufacturers do not always meet these ideas, they have a tendency, on a declining metal market, to keep the demand from this class of buyers down to their current running requirements.

In its recent application to transfer its $24,463,600 of 7 per cent. cumulative preferred and $20,750,000 common stock from the unlisted to the regular list of the New York Stock Exchange, the National Lead Company gave a list of plants owned, which, it stated, are in active operation, as follows: at Brooklyn, the Atlantic Works, the Bradley Works and the Cooks Smelting Works; Staten Island, the Jewett Works; Buffalo, Cornell Works; Cleveland, J. H. Morley Works; Cincinnati, Eckstein Works; Chicago, Shipman Works and Southern Works; St. Louis, Collier Works, Southern Works and St. Louis Works. The plants in active operation of corporations, all of whose stock is owned by the National Lead Company, are: Philadelphia, John T. Lewis & Bros.; Pittsburg, Armstrong & McKelvey Works, Beymer & Bauman Works, the Sterling Works and the Bender & Alfred Works; St. Francis, Mo., and Collinsville, Ill., St. Louis Smelting and Refining Company. The Maryland Works at Baltimore and the Davis Works at Butler, Penn., are not being operated at present.

Recent Improvements in Lead Smelting.

By H. O. Hofman.

Introductory—Physical Properties—Alloys.

New Publications.—Three books and an extended series of articles dealing with lead appeared during 1908. W. R. Ingalls, "Lead and Zinc in the United States," Hill Publishing Company, New York, 1908, pp. x+368, ill., $4. This is a treatise comprising the economic history of lead and zinc in the United States, and the conditions which have affected the development of the industries. H. W. Hixon, "Notes on Lead and Copper Smelting and Copper Converting," Hill Publishing Company, New York, 1908, pp. ix+162, ill., $3. This is a reprint of the book of 1897 with some additions. A. G. Betts, "Lead Refining by Electrolysis," Wiley & Sons, 1908, pp. ix+394, ill., a treatise dealing with the electrolysis of lead bullion in leadfluosilicate solutions. A. H. Sexton, "Notes on Lead," a series of articles dealing with the metallurgy of lead which appeared in The *Mechanical Engineer*, 1908, XXI, pp. 645, 718; XXII, pp. 60, 68, 110, 186, 286, 306, 348, 446, 476. They will form a good elementary treatise when issued in book form. A. S. Dwight in a bio-

graphical notice[1] of the late T. S. Austin gives incidentally many interesting facts regarding the history of argentiferous lead-smelting in the western part of the United States.

Reduction of Lead Oxide.—F. J. Brislee, in reducing the oxides of lead, cadmium and bismuth by means of carbon monoxide at 300 deg. C., found that they all passed through a suboxide state before they were reduced to metal.[2]

Melting Points of PbS, Cu_2S, Ag_2S and FeS.—K. Friedrich carried on a series of investigations regarding the melting points of PbS, Cu_2S, Ag_2S and FeS which throw some light upon the discrepancies of various experimenters whose several results vary as much as 100 deg. C. One of the causes of these differences is to be found in the fact that the sulphides mentioned, whether in the form of minerals or of prepared compounds, often show upon chemical analysis compositions differing slightly from the theoretical. After determining the freezing-points of analyzed specimens which contained more or less sulphur than called for by the formulae, and averaging the results, he gives the following melting-points of the true compounds, accurate to 10 deg. C.: PbS, 1120 deg.; Cu_2S, 1135 deg., Ag_2S, 812 deg.; FeS, 1171 deg. He further concludes that silver and iron monosulphides containing more sulphur than called for by Ag_2S and FeS have higher melting points than those just given, that the same is probably true with lead sulphide, and that the reverse holds good for cuprous sulphide.[3]

Volatility of Lead Sulphide.—W. Biltz examined the volatility of some metallic sulphides when exposed to elevated temperatures. Prepared lead sulphide sublimed at 950 deg. C.; galena from Freiberg at 1112 deg.; precipitated zinc sulphide at 1185 deg.; blende from Santander at 1178 deg.; prepared cadmium sulphide at 980 deg. without fusing; mercuric sulphide at 446 deg.; ferrous sulphide at 1197 deg.; nickelous sulphide at 797 deg.; cobaltous sulphide above 1100 deg.; stannous sulphide fused at 870 deg., became sticky at 1000-1100 deg., changed to fluid at 1120 deg., and was very volatile at 1100 deg.[4]

Lead Sulphide and Calcium Hydride.—Perkin and Platt have shown that PbS as well as Sb_2S_3 are vigorously attacked when heated with CaH_2; little metal is produced by the reaction, the product is a dark-gray mass which appears to be a compound of metal with Ca and S.[5]

Lead Sulphate and Lead Oxide.—R. Schenck and W. Rassbach have continued their studies upon the equilibria existing between lead sul-

[1] *Trans.*, A.I.M.E., 1907, XXXVIII, 406.
[2] *Journ. Chem. Soc.* (London), Trans., 1908, XCIII, 154.
[3] *Metallurgie*, 1908, V, 23, 50.
[4] *Zeit. anorg. Chem.*, 1908, LIX, 273.
[5] *Proc. Faraday Society*, through *Electrochem. and Met. Ind.*, 1908, VI, 13.

phide and its products of oxidation.[1] They found the reactions taking place at elevated temperatures between $PbSO_4$ and PbO to be complicated. In examining the freezing-point curve of the series $PbSO_4$-PbO, they found that the melting point of $PbSO_4$ lay above 1100 deg. C., and that at 950 deg. a retardation occurred which was due to a partial decomposition of $PbSO_4$. The curve shows the existence of two stable basic sulphates, $PbSO_4.PbO$ with 42.4 per cent. PbO, and $PbSO_4.2PbO$ with 59.5 per cent. PbO; a third basic sulphate, $PbSO_4.3PbO$, with 68.8 per cent. PbO, is stable only below 880 deg. C., as it splits above this temperature into $PbSO_4.2PbO$ and PbO. There are also three eutectic mixtures, $PbO\text{-}PbSO_4.3PbO$, with 87 per cent. PbO, freezing at 820 deg. C.; $PbSO_4.2PbO\text{-}PbSO_4.PbO$, with 53 per cent. PbO, freezing at 940 deg. C.; and $PbSO_4\text{-}PbO.PbSO_4$, with 30 per cent., freezing at 950 deg. C. In the solid state there are two transformations, one of the pure $PbSO_4$ at 850 deg. C., the other of $PbSO_4.2PbO$ at 450 deg. C.

Lead Oxide and Silica.—S. Hilpert in an investigation[2] of the formation of lead silicates, found that they begin to form above 710 deg. C., and that the velocity of the reaction, which is slow at 750 deg., increases rapidly with the temperature; he ascertained further that the relative proportions of PbO and SiO_2 at a temperature of 1000 deg., but not at 930 deg., influence the reaction in such a way that the higher the percentage of SiO_2 the lower the velocity. Mixtures with a high percentage of PbO, forming readily fusible compounds, thus assist the union of PbO and SiO_2, or, in the presence of $PbSO_4$, the expulsion of SO_3.

The same author examined[3] the volatilization of PbO from its silicate solutions. He prepared the following mixtures: $1PbO:2SiO_2$; $2PbO:3SiO_2$; $PbO:SiO_2$; and $2PbO:SiO_2$, and heated them severally to 930 and 1000 deg. C. At 930 deg. only the singulo-silicate, $2PbO.SiO_2$, showed loss by volatilization; at 1000 deg., again $2PbO,SiO_2$ showed the greatest loss, then followed PbO,SiO_2, while the other two mixtures retained approximately their original weights.

A. Portevin passes in review[4] the work done by Tammann and his students at the university of Göttingen on lead alloys. Some of these researches have already been given in these reviews, the others are herewith recorded.

Lead-Gold.—According to V. Vogel,[5] Pb and Au form two chemical compounds, Au_2Pb freezing at 418 deg. C., and $AuPb_2$ at 274 deg., and a single eutectic mixture, consisting of $AuPb_2$ and Pb which solidifies at 211 deg. The two compounds undergo transitional changes while solidifying.

[1] *Berichte d. deutsch. chem. Gesellschaft*, 1907, XL, 2185, 2947 (*The Mineral Industry*, 1907, XVI, 673); *ibid.*, 1908, XLI, 2917
[2] *Metallurgie*, 1908, V, 535.
[3] *Ibid.*, 1908, V, 539.
[4] *Rev. Métallurgie*, 1908, V, 909.
[5] *Zeit. anorg. Chem.*, 1905, XLV, 11.

Lead-Magnesium.—G. Grube[1] as well as N. S. Kurnakow with N. J. Stepanow[2] have studied the alloys of lead and magnesium. Grube found that these metals are insoluble in one another; that they form two eutectic mixtures, one of which contains 67 per cent. Pb and freezes at 459.2 deg. C, while the other contains 97 per cent. Pb and freezes at 246.9 deg.; and that between the two eutectics there is the chemical compound Mg_2Pb with 80.9 per cent. Pb solidifying at 551.3 deg. Kurnakow and Stepanow, on the other hand, locate the second eutectic with 2.10 per cent. Mg at 253 deg., the chemical compound Mg_2Pb at 550 deg., and the first eutectic with 31.68 per cent. Mg at 475 deg.

Lead-Palladium.—According to R. Ruer[3] these metals form four chemical compounds, $PdPb_2$, PdPb, Pd_2Pb, Pd_3Pb, and two eutectic mixtures. Only two compounds can be fused without being decomposed, $PdPb_2$ (melting point 454 deg. C.) and Pd_3Pb (melting point 1219 deg.); one of the eutectics with 5 per cent. Pd and consisting of Pd and $PdPb_2$ solidifies at 265 deg., the other with 65 per cent. Pd and consisting of Pd_3Pb and Pd, freezes at 1200 deg.; the melting point of Pd lying at 1541 deg.

Lead-Indium.—N. S. Kurnakow and N. A. Puschin found[4] that these metals form solid solutions; the melting point of indium (154 deg. C.) is gradually raised by additions of Pb to the melting point of lead (327 deg.).

Lead-Thallium.—As determined by N. S. Kurnakow and N. A. Puschin,[5] the freezing-point curve of Pb (melting point 327 deg. C.) and Tl (melting point 301 deg., transformation-point 227 deg.) shows a maximum at 380 deg., with 33 per cent. Pb, which corresponds to a compound lying between Tl_2Pb and Tl_3Pb_2. Starting from the maximum toward Pb, there is a series of solid solutions; going in the opposite direction toward Tl, there is a jog in the curve at 310.4 deg., represented by the point 5.5 per cent. Pb. Here the solid solution with 24.7 per cent. Pb reacts with the liquid solution containing 5.5 per cent. Pb and forms a solid solution with 6.5 per cent. Pb. There are then (in the cold), two solutions, one with 0 to 6.5 per cent. Pb, the other with 24.7 to 100 per cent. Pb.

K. Lewtonja repeated the above experiments.[6] He determined the melting point of Tl at 299.4 deg. and found the maximum to lie at 374 deg. with 34 per cent. Pb, which corresponds to the compound $PbTl_2$. From $PbTl_2$ to Pb there is the same series of solid solutions; the jog

[1] *Zeit. anorg. Chem.*, 1905, XLIV, 117.
[2] *Ibid.*, 1906, XLVI, 177.
[3] *Ibid.*, 1907, LII, 345.
[4] *Ibid.*, 1907, LII, 442.
[5] *Ibid.*, 1907, LII, 460.
[6] *Ibid.*, 1907, LII, 452.

between $PbTl_2$ and Tl was found to lie at 309.5 deg. with 5 per cent. Pb; the solid solution Tl in $PbTl_2$ (with 24 per cent. Pb) reacts with the liquid solution (with 5 per cent. Pb) and forms a new solid solution (with 6.5 per cent. Pb).

Lead-Platinum.—F. Doerinkel found[1] that there exist two chemical compounds, the first has the formula PtPb and freezes at 785 deg. C.; the composition of the second which solidifies at 356 deg. C. could not be determined. He further identifies a eutectic mixture with 5 per cent. Pb freezing at 289 deg., which is made up of the undetermined chemical compound and metallic lead.

Lead-Manganese.—According to R. S. Williams[2] these metals are only slightly soluble in one another and form neither chemical compounds nor eutectic mixtures.

Lead-Antimony.—The constitution of this series has been already examined repeatedly. However, W. Gonterman covered the field again.[3] He found the eutectic with 87 per cent. Pb to freeze at 250 deg. C. and ascertained again that there existed no chemical compound, the contrary opinion having been held by investigators who studied the alloys from the point of view of electrical conductivity.

Lead-Potassium.—According to D. P. Smith,[4] these metals form four chemical compounds: KPb_4(?), KPb_2, an unknown X and K_2Pb which undergoes a transformation in the solid state; there exist further two eutectic mixtures, one of Pb and KPb_4 with 80(?) per cent. Pb, freezing at 275 deg. C., and the other of KPb_4 and K with about 98 per cent. K, freezing at about 5 deg. below the melting point of K. A number of transformations make the diagram complicated.

Lead-Nickel.—G. Voss, who studied this series, found[5] that the two metals have little affinity; Ni can dissolve 4 per cent. Pb, but Pb does not dissolve any Ni.

Lead-Calcium.—L. Donski examined the alloys of lead and calcium;[6] he proved the existence of the chemical compound $CaPb_3$ which melts at 649 deg. C., showed that the alloys with up to 6 per cent. Ca consist of crystals of $CaPb_3$ dissolved in Pb, and identified an eutectic mixture with 9.7 per cent. Ca freezing at 635 deg. C., one branch of which was made up of $CaPb_3$.

Lead-Aluminum.—These metals have no affinity whatever to one another. A. G. C. Gwyer has fixed this generally accepted opinion as a definite fact.[7]

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[1] *Zeit. anorg. Chem.*, 1907, LIV, 358.
[2] *Ibid.*, 1907, LV, 31.
[3] *Ibid.*, 1907, LV, 419.
[4] *Ibid.*, 1907, LVI, 133.
[5] *Ibid.*, 1908, LVII, 45.
[6] *Ibid.*, 1908, LVII, 208.
[7] *Ibid.*, 1908, LVII, 147.

Lead-Cobalt.—According to K. Lewkonja[1] these metals are only slightly soluble in one another. Thus the melting point of Co of 1440 deg. C. is lowered only to 1438 deg. by the presence of Pb, and that of Pb only 1 deg. by Co. An analysis of lead-bearing cobalt gave Co 97.21 per cent.; Pb 0.87 per cent.; one of cobalt-bearing lead, Pb 98.73 per cent., Co 1.08 per cent.

Lead Chromium.—While Pb does not dissolve any Cr, according to J. Hindricks,[2] Cr dissolves as much as 27 per cent. Pb with a reduction of its melting temperature; at 1470 deg. C. this solution changes to one containing 25 per cent. Cr, and this with the lowering of the temperature drops the Cr until all of it has fallen out of solution when the freezing-point of lead has been reached.

Binary and Ternary Alloys of Tin, Lead, Bismuth and Cadmium.—A. Stoffel made a study of the constitutions of these fusible alloys.[3]

Binary Series.—Pb and Sn form a single eutectic with 15.37-18.26 per cent. Sn which solidifies at 183-184 deg. C.; Sn and Cd also form a single eutectic with 71.79 per cent. Sn, freezing at 177 deg. C.; Sn and Bi a single eutectic with 56.94 per cent. Bi, freezing-point of 139 deg. C.; Cd and Bi a single eutectic with 59.82 per cent. Bi, freezing-point 166 deg. C.; Bi and Pb a single eutectic with 56.68 per cent. Bi, freezing-point 125 deg. C.; Pb and Cd a single eutectic with 52.42 per cent. Cd, freezing-point 149 deg. C.

Ternary Series.—The eutectic of Sn-Cd-Pb contains Sn 49.8 per cent., Pb 31.96 per cent., Cd 18.11 per cent., freezing-point 145 deg. C.; the eutectic of Sn-Cd-Bi contains Sn 25.93 per cent., Cd 20.28 per cent., Bi 53.79 per cent., freezing-point 103 deg. C.

Industrial Alloys.—E. Touceda patented[4] a number of lead-cadmium mixtures which are to serve as bearing-metals. The cadmium is to harden the lead and to give it a fine-grained structure in distinction to antimony, which makes the structure coarse. The accompanying table

LEAD CADMIUM ALLOYS.

	Cheap work Light load.	General work Ordinary load.	Bearings subject to shock.	Pinion ends of armatures.	Extremely severe loads.
Pb.....	97.25	92	89	86	82
Cd.....	2.75	8	11	14	18

gives some of the mixtures and the uses to which they are put. In this connection the freezing-point curve of Stoffel given above is of interest.

G. F. Allen patented an alloy of lead and zinc for making lead-

[1] *Zeit. anorg. Chem.*, 1908, LIX, 312.
[2] *Ibid.*, 1908, LIX, 428.
[3] *Ibid.*, 1907, LIII, 137.
[4] U. S. patent, No. 897,431, Sept. 1, 1908.

pipe.[1] The mixture contains Pb 99.25-95 and Zn 0.75-5 per cent. and is said to be stronger than pure lead. While this may be true, it is well known that zinc-bearing lead is readily corroded.

A. Sapozhnikow has investigated the hardness of lead-tin alloys by means of the Brinell method.[2] His results are given in the accompanying table. The addition of tin to lead increases the hardness up to 40 per

SAPOZHNIKOW'S RESULTS.

Pb, per cent	100	90	80	70	60	50	40
Freezing Temp., deg. C	316	295	272	253	235	213	183
Hardness, lb. per sq. in	5714	14,428	17,371	20,657	22,800	21,285	20,820
Pb, per cent	34	33	32	30	20	10	0
Freezing Temp., deg. C	180	180	178	180	198	215	222
Hardness, lb. per sq. in	23,800	22,000	20,820	22,628	21,714	18,928	5,914

cent. Sn, the addition of lead to tin increases it up to 30 per cent. Pb. The greatest hardness is reached with 34 per cent. Pb. The eutectic mixture contains 33 per cent. Pb.

Lead-Burning.—C. Lightfoot proposes[3] to use oxygen instead of air in lead-burning, as he is thus enabled to employ coal gas or natural gas, instead of being confined to hydrogen, and gains a reduction in cost of 50 per cent. The oxy-coal gas blowpipe is being introduced by the Linde Air Products Company, Buffalo, N. Y.

Smelting Practice.

Articles of cost have been prepared by Finlay, Adkinson, Brownlee, Grabill and Guiterman.[4]

Smelting at Selby Works.—T. A. Rickard gives, in a popular article entitled "From Mill to Battleship," some facts about the works of the Selby Smelting and Lead Company, near San Francisco, Cal. The works, which were founded in 1868 and came under the control of the American Smelters Securities Company in 1905, refine annually about 25 million dollars worth of gold and from 12 to 15 million ounces of silver. The ores come mainly from Idaho and Nevada; they are smelted in a blast-furnace, 36x144 in. at the tuyeres, which puts through in 24 hours, with about 10 per cent. Australian coke and a blast pressure of 33-34 oz., 130-140 tons of charge containing about 15 per cent. Pb. The slag aimed for is SiO_2 31 per cent., FeO 35 per cent., CaO 18.5 per cent., Al_2O_3 7 per cent. It contains in addition, MgO 1.5 per cent., Pb 1 per cent., S 1-1.5 per cent., Zn 2.5 per cent. The matte produced is roasted in three 90-ft. and one 150-ft. Ropp mechanical furnaces. Roasted matte con-

[1] U. S. patent, No. 897,953, Sept. 8, 1908.
[2] *Journ. Russian Physic. Chem. Soc.*, 1908, XL, 92, through *Metal Ind.*, 1908, VI, 60.
[3] *Electrochem. and Met. Ind.*, 1908, VI, 82.
[4] *Min. and Sci. Press*, 1908, XCVI, 22; *Eng. and Min. Journ.*, 1908, LXXXV, 165; op. cit., 1908, LXXXV, 992; op. cit., LXXXV, 1110; op. cit., LXXXVI, 73; op. cit., LXXXVI, 960.

tains Pb 12 per cent., Cu 12 per cent., Fe 45 per cent., Zn 3 per cent., S 7 per cent. When the copper content reaches 16 per cent. the matte is concentrated in a copper blast furnace to 45-55 per cent. Cu. This matte is roasted and then leached with sulphuric acid charged with some blue vitriol, and the solution worked up for blue vitriol.

The lead bullion, assaying 255 oz. Ag and 15 oz. Au per ton, is desilverized by the Parkes process. It is melted down in four kettles arranged in pairs, each pair holding 60 tons, and drossed; the dross is freed from adhering lead by a Howard press under a pressure of 90 lb. per sq.in., the entire operation taking 10-12 hours. The drossed lead is siphoned into a softening reverberatory furnace and freed from antimony. The resulting antimony skimmings are smelted with some fine coke, galena and coppery dross in a small reverberatory furnace which yields antimony skimmings with 35-40 per cent. Sb, free from precious metals, to be reduced in a blast furnace to hard-lead. The lead-bullion obtained forms 58-62 per cent. of the weight of the lead in the charge. The softened lead-bullion is desilverized in four oil-fired kettles, 10 ft. diam. and 3 ft. deep, each holding 60-65 tons lead. The resulting zinc-crust, freed from unalloyed lead by means of the Howard press, assays 3000-3500 oz. AgAu per ton, 22-24 per cent. Zn, 3 per cent. Cu; the rest is Pb. The desilverized lead is freed from zinc in a reverberatory furnace of 70 tons capacity. The refined lead is molded from a kettle holding 60 tons into a set of 50 molds placed in three-quarters circle; the shipping bars weigh 100 lb. The zinc-crusts, when distilled, give rich lead with 4000-6000 oz. AgAu per ton to be cupelled. The doré silver is parted with sulphuric acid, and the silver sulphate decomposed with metallic copper. The resulting acid solution of cupric sulphate serves as solvent for copper from roasted matte as was stated above. It may be mentioned that the liquors obtained by the condensation of the acid fumes from the parting-kettles is cherry-red to ox-blood in color, the coloration being caused by the presence of selenium. The paper further discusses the manufacture of sheet lead, lead pipe, shot and of white lead as carried out at the plant.

Smelting in Peru.—L. W. Strauss publishes some detailed notes on the only lead smelter of Peru, which is situated at Vesubio, 14,000 ft. above the level of the sea.[1] The ores treated assay Pb, 5-36 per cent.; Cu, trace to 0.5 per cent.; Zn, 6-20 per cent.; Sb, 0.5-3 per cent.; Ag, 23-100 oz. per ton. They are crushed, roasted in a hand reverberatory furnace, and smelted in a water-jacket blast furnace, 3 ft. 3 in. diam. at tuyeres, with siphon-tap. The concentration is 12:1. The lead bullion assays Pb, 89 per cent.; Cu, 3 per cent.; Sb, 3.5 per cent.; Ag, 698, and Au, 0.95 oz. per ton. The details given have little general interest.

[1] *Min. and Sci. Press*, 1908, XCVII, 361.

Smelting at Mapimi, Mexico.—C. T. Rice describes[1] briefly the smelter of the Peñoles company, Mapimi, which has been remodeled to a considerable extent. Its capacity is 1000 tons of ore per day; at present 700 tons of ore, mainly oxide and rich in arsenic, is treated, and the lead bullion shipped either to the United States or to Belgium. The plant has 12 Huntington-Heberlein pots; the old hand reverberatory furnaces are used for subliming arsenious oxide from flue dust. There are six blast furnaces, 46x162 in. at tuyeres, which are supplied with Robinson tuyeres (see below). A novel feature is found in the slag-pots. They are of five tons capacity, collect both matte and slag, are taken up by an electric crane and emptied into a reverberatory-furnace settler of 80 tons capacity. A pot has a pair of trunnions extending beyond a bail which is hooked to the tackle of the crane. Beneath the slag-spout is an inclined track on which the pot, when brought in front of the furnace, is lowered and made to glide into position on its trunnions. The advantages of the arrangement are that the front of the furnace is not encumbered with tracks and slag cars, that there is less fume around the furnace, that time is allowed for a good separation of matte and slag, and that the slag-trains are made up away from the furnaces.

Smelting at Granby, Mo.—E. W. Buskett gives a description[2] of the Granby works at Granby, Mo., which is accompanied by two plan-drawings; one represents the five ore-hearths with blast-main and dust chambers, the other, the lower part of the baghouse with cooling-pipes coming from the dust chambers. The cast-iron lead basin of an ore-hearth is 4 ft. long, 15 in. wide and 10 in. deep; it is surrounded on three sides by water-jackets about 12 in. high. At the back there are four tuyere openings pointing downward; at the front is the apron-plate, about 18 in. wide, with the usual diagonal groove. The furnaces are worked only during the day. Two men tend a furnace, changing about every 20 minutes. At 6 a.m. a charcoal fire is made on the solid lead in the basin; usually at 7 o'clock the lead has been liquefied and heated; now, ore with some lime is spread over the incandescent coal; after a few minutes' roasting, the mixture is stirred, more coal is given and fresh charge added. A furnace works 14,000 lb. charge and fume during the day, requires 2 bu. lime, extracts 75 per cent. of the lead present in the form of bars, and makes 2000 lb. gray slag with 40 per cent. Pb. The lead-content of the slag is equal to 10 per cent. of the metal in the charge. The remaining 15 per cent. passes off as dust and fume (blue fume) into a dust chamber which is 30 ft. long, 12 ft. high, 12 ft. wide at top and 6 ft. at bottom. The five hearths make, per day, about 2400 lb. of

[1] *Eng. and Min. Journ.*, 1908, LXXXVI, 373.
[2] *Min. World*, 1908, XXIX, 917.

blue fume assaying 55-60 per cent. Pb. From the dust-chamber the gases pass through a balloon-shaped cooling pipe, 400 ft. long, 4 ft. high and 3 ft. wide at the bottom, provided with cleaning-hoppers; thence they travel consecutively through a goose-neck, a horizontal U-shaped pipe 250 ft. long, four goose-necks, a straight pipe 100 ft. long, and reach the fan, 3x5 ft. made by the American Blower Company, which is driven by a 20-h.p. Fairbanks-Morse gasoline engine. The fan delivers into a pipe which runs in the open along one side of the baghouse. From it, eight branches with gates lead into the eight brick rooms on the ground floor of the baghouse. They are about 10 ft. high, and are covered by a sheet-iron flooring which has 640 thimbles 18 in. diam., for the 30-ft. cotton filter-bags. At one time five rooms are filtering, one is burning, one is being discharged, and one is in reserve. A room becomes filled with blue powder in about five weeks. The plant makes per day about 10,000 lb. of fume assaying 70 per cent. Pb. This is burnt in the usual way and returned to the ore-hearth. The gray slag is sold to works having blast furnaces.

Smelting in Silesia.—J. S. G. Primrose gives[1] a general description of the treatment of zinc and lead ores in Silesia which is accompanied by photographs of various furnaces and apparatus. Zinc is the more important product; only a few plants do any lead smelting. The leading lead smeltery is that of Tarnowitz, at which the Huntington-Heberlein process, followed by blast furnace work, has replaced the celebrated method of smelting in the reverberatory furnace. To the twenty 5-ton converters resting with trunnions on four-wheel carriages, a 15-ton pot has been added which is supported by a steel frame 15 ft. above the floor level. This is probably the largest Huntington-Heberlein in operation. It is interesting to record the fact that at Tarnowitz, even with the best of care, some metallic lead is liable to be produced in a blow. At present a single blast furnace with spray-jackets puts through 500 tons of charge in 24 hours. The base bullion produced is desilverized by the Parkes process; the silver crust therefrom is distilled in a small muffle furnace; to the condensers of the muffles there are attached vertical prolongs to collect the fumes.

Smelting at Cañon City, Colo.—A general description of the works of the United States Smelting Company, at Cañon City, Colo., accompanied by photographic reproduction of some of the parts of the plant, is given by an editorial correspondent.[2] The work consists insmelting low-grade zinc-lead ores for matte and pigment.

Metallurgical Calculations.—In the welcome series of articles under this

[1] *Eng. and Min. Journ.*, 1908, LXXXVI, 265.
[2] *Min. World*, 1908, XXIX, 167.

heading, J. W. Richards has reached the metallurgy of lead.[1] His work is so well known to the metallurgical world that the announcement of the issue of Volume III, dealing with non-ferrous metals, will be welcome to all. Attention may be called in the article on lead to a table giving the volatility of lead at different temperatures which has been calculated from the known vapor-tension curve of quicksilver. It shows e.g. that lead is volatile to a small extent at a low red heat. In a similar way it is shown that a current of air passing over silver at a temperature above 700 deg. C. and over gold above 950 deg. C., causes some metal to be volatilized. This explains easily the volatilization of silver in cupelling.

Cost of Smelting.—W. R. Ingalls analyzed the cost of silver-lead smelting in an article that is reproduced in this volume of THE MINERAL INDUSTRY.

New Process of Smelting.—A. Francis patented[2] a process for the treatment of galena. This is to be fused with a flux and poured upon a bath of lead, when air, blown through the lead, will furnish the oxide necessary for reducing the molten galena to metallic lead.

Electrolysis of Fused Galena.—E. F. Kern patented[3] a process for electrolyzing galena, using as electrolyte a bath containing lead chloride, which fuses at 500 deg. C., saturated with lead sulphide, galena melted and cast to suitable form serving as anode. Other chlorides may be mixed with lead chloride as long as the melting point of the electrolyte lies below that of the anode. With an anode current density of 500 amp. per sq.ft., and an electromotive force of about 2.5 volt., the ampere-hour efficiency ranges from 84 to 94 per cent. The reduced lead is malleable and practically free from sulphur; the sulphur (boiling-point 445 deg. C.) liberated is free from chlorine and sulphur-chlorides.

Blast-Roasting.—H. O. Hofman and W. Mostowitsch have investigated[4] the behavior of calcium sulphate at elevated temperature without and with fluxes. The conclusions they arrived at are, that $CaSO_4+2H_2O$ loses its combined water at about 900 deg. C.; that $CaSO_4$ remains unchanged up to 1200 deg. C., when heated in dry air, and then begins to be dissociated into CaO, SO_2 and O; and that it fuses at 1360 deg. C. They found the decomposing effect of silica upon calcium sulphate to begin at 1000 deg. and to be completed at 1200 deg. with the formation of calcium silicate. In examining the effect of ferric oxide, they ascertained that Fe_2O_3 was not decomposed when heated in air up to 1375 deg.; at 1100 deg. it begins to decompose $CaSO_4$; the decomposition was completed at 1250 deg. with the formation of ruby-colored $CaO.Fe_2O_3$.

[1] *Electrochem. and Met. Ind.*, 1908, VI, 8,67.
[2] U. S. patent, No. 882,193, March 17, 1908.
[3] U. S. patent, No. 885,761, April 28, 1908.
[4] *Trans.* A. I. M. E., October 1908; July Bulletin, 1909.

In studying the behavior of PbO, they settled that PbO has no decomposing effect upon $CaSO_4$, and found in their experiments that PbO in a current of air begins to be volatilized at 700 deg. The paper shows what applications can be made of the results of the investigation to various metallurgical processes.

P. Weiller believes[1] that the following four equations represent the leading reactions that take place in the Huntington-Heberlein process between PbS, CaO and O: (1) $PbS+4O=PbSO_4$. (2) $PbS+3O=PbO+SO_2$. (3) $PbS+CaO=CaS+PbO$. (4) $CaS+PbSO_4=CaSO_4+PbS$. Equations 1 and 2 are generally accepted; the author gives his evidence for the probability of equation 3, which appears satisfactory. Doeltz has shown[2] that there is a tendency in the direction of the reaction given in equation 4. R. Tandler objects[3] to this stringing together of reactions which severally have their justification, as representing what takes place in the Huntington-Heberlein process. He prefers to combine equations in such a manner as to have all the acting agents in the process grouped on the left side of a single equation and the products of the reaction on the right. This he does in (5) $PbS+CaO+O=PbO+CaSO_4$. He proposes, however, the following two equations as covering the ground: (1) $PbS+4O=PbSO_4$, and (2) $PbSO_4+CaO \rightleftharpoons PbO+CaSO_4$. But Hofman and Mostowitsch have proved experimentally that the second reaction read from left to right does not take place; further, there is no evidence to prove that it will take place in the reverse sense. The paper of Weiller is discussed by G. Kroupa.[4]

It has always been held that the gases from the Huntington-Heberlein pots could not be utilized for the manufacture of sulphuric acid, as they contain too little sulphur dioxide. According to H. Wislecenus[5] they are converted into sulphuric anhydride by the contact process. In this connection it may be noted that the volume of SO_2 set free during a blow is small at first, increases to a maximum and then again decreases. By running several pots in consecutive order, it has been made possible to obtain a gas of uniform percentage of SO_2.

A. Selwyn-Brown in a general article[6] on the metallurgy of lead and zinc, gives the accompanying drawing of the pot used by the American Smelting and Refining Company, at Murray, Utah, Fig. 1. Details of the mode of operating have already been given in these reviews.[7]

L. S. Austin gives[8] the following data regarding blast-roasting of 1000

[1] *Chem. Zeit.*, 1908, XXXII, 348.
[2] *Metallurgie*, 1905, II, 406.
[3] *Chem. Zeit.*, 1908, XXXII, 437.
[4] *Oest. Zeit. f. B. u. H.*, 1908, LVI, 273.
[5] "Ueber die Grundlagen technischer und gesetzlicher Massnahmen gegen Rauchschäden," Parey, Berlin, 1908, 45.
[6] *Eng. Mag.*, 1908, XXV, 857
[7] *The Mineral Industry*, 1907, XVI, 665.
[8] *Min. and Sci. Press*, 1908, CXVI, 641.

tons of ore in a 50-ton plant by the Huntington-Heberlein process: Loss of weight, 25 per cent.; gain in Au, 7 per cent.; loss in Ag, 1.4 per cent.; in Pb, 8.5 per cent.; and in Cu, 1.4 per cent. Analysis of blast-roasted ore, SiO_2 6 per cent., Fe 35 per cent., Cu 1 per cent., Zn 8.5 per cent., Pb 15 per cent., S 5.5 per cent., Ag 21 oz. per ton. Cost of blast-roasting 43 tons of mixture: 13 men @ $2.50 per day, $27.34; foreman (part) $1.40; slack coal for rough-roasting, six tons @ $1.55, $9.30; power (estimated), $3; lighting (estimated), $0.40; repairs and supplies (estimated), $2.50; total cost, $43.94; cost per ton, $1.02. The cost of roasting in a hand reverberatory roasting furnace at the same plant is

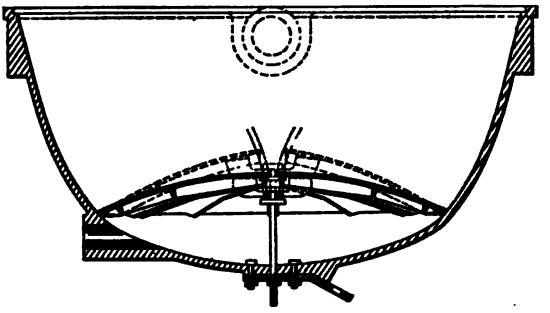

FIG. 1. KETTLE FOR BLAST-ROASTING PROCESS AS USED AT THE MURRAY SMELTER, UTAH.

$2 per ton, of briquetting the roasted material $0.60; the loss in lead shows a range of from 11 to 17 per cent. Comparing the two processes gives: costs, $1.02 vs. $2.60; loss in Pb, 8.5 vs. 11 to 17 per cent.; quality of product for blast-furnace treatment, good vs. passable.

According to a note[1] the Carmichael-Bradford process is used more largely than before by the Broken Hill Proprietary Company, the benefits resulting from it being cheapness of work and reduction in loss of metal. From another source, comes the information that the Dwight-Lloyd sintering-machine is going to replace the original up-draft pots.

A. S. Dwight discussed blast-roasting in general in THE MINERAL INDUSTRY, XVI, 380. To the original drum-type Dwight & Lloyd machine two other forms, the straight-line and round-table machines have been added. Two United States patents[2] have been granted the inventors.

Location of Blast-Furnace Smelting Works.—R. R. Moore in replying[3] to the article by H. Lang, who strongly advocated[4] the level site for a plant of large capacity, states that the iron blast furnace, 80-100 ft. in

[1] *Eng. and Min. Journ.*, 1908, LXXXVI, 954.
[2] U. S. Patents Nos. 882,517 and 882,518, March 17, 1908.
[3] *Eng. and Min. Journ.*, 1908, LXXXV, 546.
[4] *The Mineral Industry*, 1907, XVI, 677.

hight and producing 50 per cent. of its charge as metal, can not well be compared with a copper or lead blast furnace which produces from 2 to 15 per cent. of the weight of the charge in the form of matte or metal. In general, the simplest site will require five levels for unloading, storing, charging, tapping and dumping. Two or more of these may be on the same plane, the unloading-level, however, should always be situated above the storing-level. Excluding return-material, there will be in a three-terrace plant, three horizontal movements and no lifts; in a two-terrace plant, three horizontal movements and one lift; in a one-terrace plant, three horizontal movements and two lifts; in a level plant, three horizontal movements and three lifts. As all four cases require the same number of horizontal movements, the number of terraces best suited will have to be decided upon for each individual case.

The Blast Furnace Corrosion of Crucible.—O. Stulzer records[1] some observations made upon the bottom of a blast furnace at Freiberg, Saxony, which had been blown down. The bottom had been built of fire-brick laid upon a foundation of gneiss. Both the brick and rock were permeated by metallic sulphides, and especially by galena. In order to decide whether the presence of sulphide was due to infiltration or to sublimation, Stulzer carried out experiments in the laboratory by melting down sulphides in a plumbago crucible and adding heated brick and rock; he found that the sulphides easily permeated both brick and rock.

Removal of Hearth Accretions.—J. N. Goddard describes[2] the Bardill device for removing accretions in the crucible of a lead blast furnace. It consists of a cartridge filled with 50- or 60-per cent. dynamite, which is pushed by means of a rod into the crucible through the lead-well, when it will explode in from 15 to 20 seconds. While slag may be splashed into the tuyeres, and some lead forced out of the well, the brickwork is not injured by the explosion. The cartridge consists of a nipple 1 in. diam. and 4 in. long, two sleeves and two plugs. The nipple is lined with asbestos paper $\frac{1}{8}$ in. thick, 1/7 of a stick of dynamite is inserted and protected at the ends by shredded asbestos. One of the plugs is tapped to receive a $\frac{1}{2}$-in. iron rod from 4 to 7 ft. long.

Corrosion of Water-Jackets.—G. B. Lee records[3] some facts regarding the corrosion of steel water-jackets of the copper blast furnaces at Douglas, Ariz. The inner plates were badly corroded in 10-12 months, while the outer plates and stiffeners were little attacked; boilers fed with the same water showed little corrosion; cast-iron impellers of rotary pumps were badly pitted. The foreign substances in the feed-water do not appear to be directly responsible for the corrosion. Discussions by

[1] *Zeit. f. prakt. Geol.*, 1908, XVI, 119.
[2] *Eng. and Min. Journ.*, 1908, LXXXVI, 763.
[3] *Trans.* A.I.M.E., 1907, XXXVIII, 877.

Kent, Douglas, Hixon and Van Arsdale point to two causes, viz., to carbon dioxide, which is liberated when the water becomes heated and travels upward along the inclined inner plane and to electrolytic action. The remedy against carbon dioxide is to use fire-sheets of copper, or to expel the gas by heating and then cool the water before using in the jackets. The correction for electrolytic corrosion is to suspend sheet-zinc inside the jackets. C. D. Demond believes that corrosion may be caused in part by dissolved oxygen. In experimenting to find remedies, he found that the addition of 0.1 per cent. lime to the water was successful, also the use of 1 lb. potassium bichromate in 1500 gal. water, as recommended by Cushman. Another cause is to be found in the uneven character of the steel which may form positive and negative poles, and set up electrolytic corrosion. This unevenness is likely to occur more frequently with the inner plates, which are $\frac{1}{2}$ in. thick and have received less work than the outer ones, which are $\frac{3}{8}$ in. thick. A. S. Dwight, finally, records the experiences of R. L. Lloyd at Cananea, Mexico, where the evil was attributed to sulphurous gases and was corrected by the addition of arsenious oxide to the cooling water.

Heat-Loss Through Water-Jackets.—L. S. Austin determined[1] the heat-loss of a lead blast furnace through the jacket-water at 10.7 per cent. of the total heat. The furnace was 36x100 in. at the tuyeres, had a 6-in. bosh and a 15-ft. smelting-column; it put through in 24 hours 80 tons of charge making a slag with SiO_2 35 per cent.; FeO 28 per cent.; CaO 27 per cent. The jackets received per hour 2000 gal. of water of 22 deg. C. and delivered it at 67 deg. C.; the amount of coke per hour was 1000 lb.

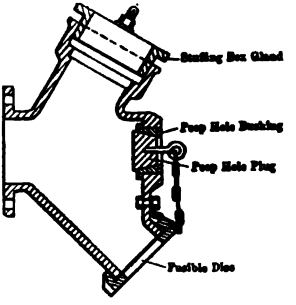

FIG. 2. ROBINSON'S NON-SLAGGING TUYERE.

Tuyere.—C. Robinson describes[2] his non-slagging tuyere, which is shown in section in Fig. 2. Its two characteristic features are an outlet-pipe for slag and a peep-hole with plug. The pipe, 4 in. inner diameter,

[1] *Min. and Sci. Press*, 1908, XCVII, 525.
[2] *Eng. and Min. Journ.*, 1908, LXXXV, 251; *ibid.*, LXXXVI, 756.

is placed on the lower side of the elbow and closed by fusible disc 5 in. diam., and 3/32 in. thick. The disc is composed of 60 parts tin and 40 parts lead, and is held in place by a cast-iron ring-cap. At Mapimi it is made of lead-bullion, the metal being returned to the blast furnace when it has served its purpose. The peep-hole plug is of steel and is attached to the elbow by means of screw-eye and chain. It is forced against a brass bushing by the pressure of the blast, falls into the pipe when the pressure is released, and is pulled again in place by the chain when the pressure returns. At Mapimi the consumption of air, which is delivered at a pressure of 48 oz., has fallen from 72,000 to 54,000 cu.ft. per ton of charge since this tuyere has been put in operation.

Chemistry of the Blast Furnace.—L. S. Austin has given out[1] another instalment of the letters of the late T. S. Austin. It deals with an investigation of the gases from a silver-lead blast furnace at Monterey, Mexico. The aim was to find a relation between the ratio $CO : CO_2$ of the gases and the three important factors in furnacing, namely speed, reduction of lead as shown by the slag-assay, and fuel consumption. If the ratio is low, it is fair to assume that the utilization of the fuel charged is high. If the low ratio is accompanied (*a*) by a high speed and a high percentage of lead in the slag, the inference is that the amount of fuel used is insufficient; (*b*) by a low speed and a high percentage of lead in the slag, the inference is the same; (*c*) by a low speed and a low percentage of lead in the slag, the inference is that the amount of fuel used is excessive. There are given two sets of data. The first is not sufficiently complete to allow drawing any definite conclusions; from the second the following samples of gas, taken on a single day when the furnace was kept filled to the top, are of interest. They show that the CO continued to have a

GAS ANALYSES AT MONTEREY, MEXICO.

CO	CO_2	Sample taken.	CO	CO_2	Sample taken.
% vol.	% vol.	Below surface	% vol.	% vol.	
20.3	9.5	2 ft. (*a*)	9.0	19.7	At surface(*b*)
19.2	18.2	2½ "	8.8	19.8	
10.4	16.8	3 "	9.8	17.6	
8.3	17.0	6 "	2.7	21.8	

(*a*) about 600 deg. C. (*b*) about 120 deg. C.

reducing effect from 2½ ft. below the surface until it left the top of the charge, the gas traveling with a speed of about 5 ft. per second.

Handling of Lead-Bullion.—J. C. Bennett describes in a well-illustrated article[2] the handling of lead-bullion at the Selby smelting works. The lead flows from the well alternately into two tilting kettles. These

[1] *Min. and Sci. Press*, 1908, XCVII, 364.
[2] *Eng. and Min. Journ.*, 1908, LXXXVI, 83.

are supported by trunnions on trucks which run on two sets of tracks, intersecting at an angle of about 130 deg., under the spout of the well. A kettle filled with lead is run to the end of the track, and the lead discharged into a set of 10 flanged cast-iron molds set on a water-cooled box; the molds have flanges which overlap, and the flanges have grooves that become quickly filled with lead which solidifies and forms an efficient packing. From the water-cooled molds the bars of lead are loosened in the usual way with a pick and then transferred by hand to a flat-car which is run to the refinery.

Constitution of Matte.—C. H. Fulton and I. E. Goodner carried on an interesting investigation into the constitution of copper-iron and copper-lead-iron matte, the results of which are summarized in the review by Prof. L. S. Austin in earlier pages of this volume.

Handling of Matte.—J. C. Bennett describes[1] with drawings and photographs the manner of handling matte at the Selby smelting works. In front of the settling-forehearth are two parallel tracks for full and empty trucks. The journal-boxes of a truck are closed and lined with brass; they carry pressed-steel pans 3 ft. 9 in.x2 ft. 3 in. at top, 2 ft. 7 in.x1 ft. 1 in. at bottom, and 7 in. deep. Before running matte into a mold, a shovelful or two of crushed matte is thrown on the place where the stream would strike the steel. Some pans have lasted as long as two years, while cast-iron has proved a failure.

In this connection may be mentioned the Kilker matte-car made by the Pacific Foundry, San Francisco, Cal. It consists of an I-beam frame mounted upon square axles which are carried by 12-in. car wheels. The frame carries a cast-iron center-piece, that supports a steel pin over which is slipped a cast-iron spider. This forms a revolving table which carries flat tilting matte-molds, each holding a cake which is 4 in. thick and weighs about 200 lb. The weight of the car with 16 molds is about 6000 lb. The cast-iron molds here appear to have been quite satisfactory.

Heap-Roasting Matte.—R. C. Canby[2] in roasting, in heaps of 1800 tons capacity, lead blast furnace matte with 5-7 per cent. Cu and about 12 per cent. Pb, found that, during the three months a roast lasted, minerals such as pyrite, pyrrhotite, chalcopyrite and galena had been formed from the matte.

Formation Temperatures of Slags.—H. Steffe investigated[3] the formation and melting temperatures of various ferrous and ferro-calcic silicates. He prepared FeO by heating to 500-600 deg. C. in a current of N, a mixture of Fe_2O_3 and Fe, as indicated by Berthier many years ago; the other ingredients were pure SiO_2, prepared from a quartz crystal,

[1] *Eng. and Min. Journ.*, 1908, LXXXV, 252.
[2] *Ibid.*, 1908, LXXXV, 719.
[3] Doctorate Thesis, Berlin, 1908.

and calcined $CaCO_3$. The mixtures were heated in an electric furnace and in a current of N which gives a perfect control of temperature and excludes any chemical change of the FeO before it has been combined with SiO_2. The results obtained correct those arrived at by Hofman[1] who prepared FeO by heating FeC_2O_4, carried on the fusion a reducing

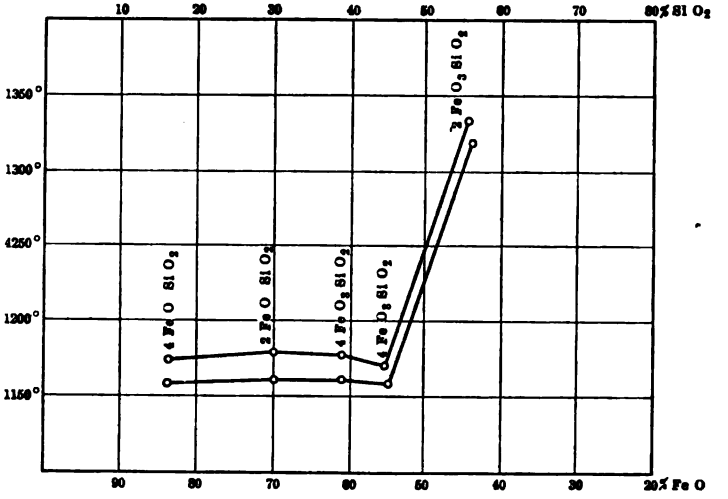

FIG. 3. FORMATION AND MELTING TEMPERATURES OF SOME FERROUS SILICATES.

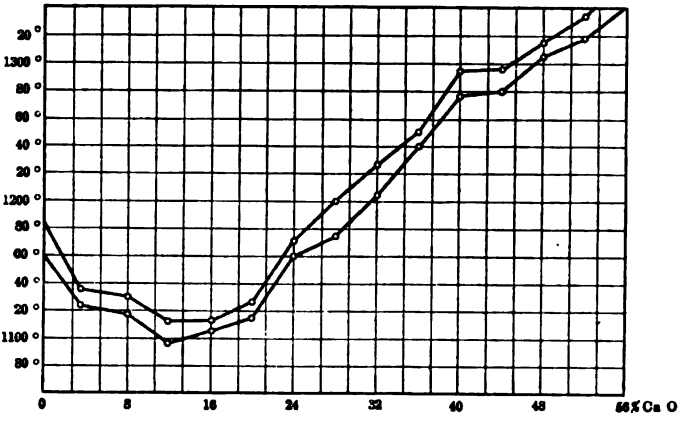

FIG. 4. FORMATION AND MELTING TEMPERATURES OF SOME FERRO-CALCIC SINGULO-SILICATES.

atmosphere of a gas-heated furnace, and used Seger cones as indicators of temperatures. The data of Hofman, while not absolutely correct, still represent the values of conditions as found in blast-furnace work. The results of Steffe with ferrous silicates are shown in Fig. 3 and with ferro-calcic singulo-silicates in Fig. 4.

[1] *Trans.*, A. I. M. E., 1899, XXIX, 682.

The vertical distances between two points represent the temperatures between which mixtures began to sinter and came to full fusion, the sintering temperatures always being lower than the fusion temperatures. The experiments prove that, strictly speaking, there are no definite formation temperatures of ferrous and ferro-calcic silicates, and the temperatures at which the silicates come to complete fusion lie higher than those at which they begin to form. This is in accordance with the researches made by Day and Sheperd, and by Doelter for some non-ferrous silicates. It, therefore, upsets the rule of Plattner, at least for finely divided, intimately mixed constituents, that the melting temperature of a formed ferrous or ferro-calcic silicate is lower than the temperature at which either begins to form. With an ore-charge, however, the old rule still holds good and will do so for ever, as far as we can see.

Constitution of Ca- and Mn-Bi-Silicates.—A. S. Ginberg in a study[1] of calcium-bi-silicate and manganese-bi-silicate finds that $CaSiO_3$ melts at 1512 deg. C., and $MnSiO_3$ at 1218 deg., and that the two form a eutectic mixture containing 12.8 per cent. $CaSiO_3$, which melts at 1180 deg. It may be remembered that Mn replacing Fe in a ferro-calcic-singulo-silicate raises the formation temperature.

Heat of Fusion of Slags.—L. S. Austin determined[2] the total heat of fusion of a blast-furnace slag of which the leading components were SiO_2, 33 per cent.; FeO, 28.5 per cent.; CaO, 25.7 per cent. His average figure is 300 pound-calories. The method followed was to pour a ladleful of slag, as received from the furnace, into a felt-covered bucket filled with 15 lb. water, to stir the water while pouring, and to note the rise in temperature. The temperature of the water before an experiment was regulated so as to be below that of the atmosphere to the extent of about half the rise expected by the experiment.

Smelting Tin-Slag.—L. Parry describes[3] and illustrates a blast furnace that is to be used in smelting tin-slags with lead-bearing material by a process patented by himself. The blast furnace is very primitive, and what is patentable in the process is not quite clear, as the smelting of tin-slags with lead-bearing material to form a tin-lead alloy is a well-established operation.

Smelter Fumes.

New Publications.—Several publications appeared during 1908 which deal with the condensation of gases and fumes in general, and with the effect of fumes upon man. Thus, H. Wislecenus has started a serial

[1] *Zeit. anorg. Chem.*, 1908, LIX, 346.
[2] *Min. and Sci. Press*, 1908, XCVI, 566.
[3] London *Min. Journ.*, 1908, LXXXIII, 279.

"Sammlung von Abhandlungen über Abgase und Rauchschäden," Parey, Berlin, 1908, which is to appear at irregular intervals. Two papers have so far been printed. One by H. Wislecenus "On the Principles Governing the Condensation of Waste Gases and the Law Regarding their Injuries;" the other by E. Schröter, "The Sources of Smoke in Saxony and its Effect upon Forestry."

K. Wächter has written a monograph "Die Gewerbliche Bleivergiftung und ihre Bekämpfung im Deutschen Reiche," Braun, Karlsruhe, 1908. The book deals mainly with dangers attending the manufacture of lead products and the painter's trade; it discusses precautions and remedies, and proposes to forbid the use of lead where its harmful effects can not be avoided. The book is considered to be valuable on the sanitary and sociological side. When the author begins to discuss technical matters relating to mining and metallurgy, he appears to be somewhat at sea.

R. Müller, one of the managers of the lead-silver smelting and refining works of Ems, Prussia, has published a book, "Die Bekämpfung der Bleigefahr in Bleihütten," Fischer, Jena, 1908, pp. 200, in which, after a general introduction, there are discussed in detail the precautions to be taken with the several operations in lead-smelting and refining. The book is the outcome of a prize-essay; it contains many valuable features, but several objectionable ones. J. Rambousek, finally, has issued a pamphlet "Ueber die Verhütung der Bleigefahr," Hartleben, Vienna-Leipsic, 1908, pp. 76, which will interest the medical student more than the metallurgist.

Constructive Details of Condensation Plants.—E. H. Messiter discusses[1] the important question of the control of the smoke in smelting works. Smoke may issue from furnaces or flues, from temporary openings, and from molten or heated materials. The lack of draft causing smoke to issue from flues may be due to an insufficient area, to a temporary obstruction such as fluedust, to a faulty construction, the use of bends of insufficient radii, to the use of T instead of Y connections, or to sudden changes in cross-section. Considering the last, it may be said that a gas-current in passing from a small to a large duct, which e.g. is to serve as a dust chamber, will pass through the latter with but slightly diminished velocity; it will make eddies in the remaining space and deposit only a small proportion of its dust. Tapering the approach to a dust chamber improves matters somewhat. Very little is known at present about the quantities of air drawn in or gas blown out through breakages in flue-walls, although both decidedly weaken the draft. As regards the use of a siphon, e.g. in carrying off the gases from a blast furnace, it is well to remember that the smoke cools steadily from the

[1] *Min. and Sci. Press*, 1908, XCVII, 26.

moment it leaves the furnace and, therefore, continually gains in density; hence the gas in the first leg of a siphon will be less dense than in the second. With an elevated siphon, i. e., one in which the gases ascend in the first leg, there will be as a rule, no difficulty as regards draft, because the smoke, once in motion, will reach the top. With a descending siphon the reverse is likely to be the case, as the denser, hence the heavier, gas has to be pushed up an incline by the lighter. A correction for smoke that issues from temporary openings is found, as regards principle involved, in that applying to leakages in flue-walls in general. In handling smoke issuing from hot material that has been drawn from a furnace, the plan of attack must be to remove the smoke before it has been diluted with air; the more concentrated the smoke the greater the promise for successful removal; hence small hoods, close to the working-openings and receptacles, for withdrawing and receiving hot material, will be the most effective. The ventilation of a smelter-building as a whole is usually interfered with by horizontal air-currents which blow the smoke out of its normal path, dilute it, and render difficult its removal. This interference can be prevented by having at least one vertical wall or partition in each direction. Monitor-roofs with ventilating openings in the sides are frequently a failure; they are always a success if they have balanced swing-doors so connected that when one side is closed, the other will be open.

Settling of Dust.—Gertner has published[1] an illustrated paper of 89 quarto pages in which he reviews the different apparatus and devices that have been and are being used to recover the dust in lignite-briquetting plants. While foreign to the subject of these reviews, the illustrations shown and the ideas of condensation discussed, may be of value in the handling of smelter fume and dust. A similar paper, though very much shorter, is that by Baldus.[2]

Removal of Fume from Melted Materials.—J. C. Bennett describes[3] with drawings, the arrangements for removing fumes from the slag-pots and matte-pans at the Selby lead works which completely clear the atmosphere on the floor of three blast furnaces. They consist simply in balanced hoods terminating in a main connected with a suction fan, the hoods to be raised and lowered telescope-fashion over the slag-pots and matte-pans. The fan, a "36-in. planing mill exhauster," is driven by a 10-h.p. induction motor at 702 r.p.m., and delivers 6778 cu.ft. air per min. at a pressure of 2.89 oz.; the actual power consumed is 6.6 h.p. It was contemplated to equip each branch pipe with a damper or valve that would be automatically opened or closed when the hood was lowered or

[1] *Preuss. Zeit. f. B. H. u. S.*, 1908, LVI, 257.
[2] *Glückauf*, 1908, XLIV, 1729 and 1760.
[3] *Eng. and Min. Journ.*, 1908, LXXXVI, 604.

raised; this would have effected a saving in power-consumption of 40-50 per cent., as it is only at rare intervals that more than three hoods are in use at the same time, but this improvement was not introduced.

Dust Flue.—The Southwest Smelting and Refining Company, of Oro Grande, N. M., has a blast furnace, 40x168 in. at tuyere level, the gases of which pass through a brick dust flue, 200 ft. long, into a stack of plate-iron.[1] The dust flue, illustrated in the original, is 14 ft. high and 16 ft. wide. It has several interesting features. Thus, it is built on an inverse catenary curve; the 8-in. wall is of two courses of brick laid in cement-mortar (2 sand: 1 cement); the bottom is V-shaped, having a slope of 4 in. in 8 ft., and is lined with a half-course of brick; the edge of the V is cut off to make room for a scraper-conveyer, 18 in. in width, which is protected from the draft in the flue by a hood. The dust settling out falls upon the inclined floor, glides down this and beneath the hood to the scraper-conveyer which, driven by a 5-h.p. electric motor, delivers the dust to a bucket-elevator discharging into a cylindrical steel tank, whence it is drawn off to a briquetting machine.

Fume Arrester.—H. Howard has patented[2] a fume arrester which consists of two vertical chambers containing a number of horizontal staggered shelves. The gases enter at the top of a chamber and leave at the bottom, the idea being that gases, hotter than the surrounding air, will be more evenly distributed over the depositing shelves if they are forced to travel downward than if they rush upward. With gases cooler than the air, the most favorable path would be in the opposite direction.

Bag-House.—H. E. Benedict describes[3] with drawings, the bag-house of the United States smelter at Bingham Junction, Utah, the general features of which have already been given in these reviews.[4] The main characteristic is the mechanical shaking of a number of bags at the top, which gives them a wavy motion traveling from top to bottom, and causes the dust to be freed more satisfactorily than when the bags are shaken near the base by a man protected by a rubber suit and a helmet provided with a supply of compressed air. The paper gives illustrations of other shaking devices patented by the author. This Benedict system of shaking the bags is briefly discussed in *Eng. and Min. Journ.*, 1908, LXXXVI, 1009. J. C. Bennett describes[5] the same plant, giving detailed drawings as well as photographs taken during erection. There are 1920 bags, 18 in. diam., and 28 ft. 8 in. effective length in the house, giving a total filtering surface of 259,392 sq.ft. They are intended to take the gases from three blast furnaces, 36x144 in. at tuyeres, rated at 540 tons of

[1] *Mines and Minerals*, 1908, XXVIII, 436; *Oest. Zeit. f. B. u. H.*, 1908, XLVI, 540.
[2] U. S. patent, No. 896,111, Aug. 18, 1908.
[3] *Min. World*, 1908, XXIX, 633.
[4] *The Mineral Industry*, 1907, XVI, 667.
[5] *Eng. and Min. Journ.*, 1908, LXXXVI, 451.

charge, which gives 410 sq.ft. filtering surface per ton of charge, and leaves 17 per cent. to take care of the gases from three cupelling, one liquating, two 60-ton softening and two 60-ton refining furnaces. The gases enter the bag-house with a temperature of 45 deg. C. The fan for handling the gases, a so-called 220-in. machine, is 11 ft. diam., and 5 ft. 6 in. wide; it is rated at 72,500 cu.ft. of air at 17 deg. C., and requires 35 h.p. at 130 r.p.m., causing a vacuum of 0.75 oz. The fan is actually driven at a speed of 116 r.p.m., displaces 76,000 cu.ft. gas per minute and requires 24.75 h.p. There is, therefore, 3.413 sq.ft. of filtering area for every cubic foot of gas entering the chamber. It is believed that 1.14 sq.ft. is an ample provision.

Wet Condensation.—S. I. Clawson patented[1] a device for the wet condensation of smelter-fume, which consists essentially of an upright hollow driven wheel with discharge-openings at the periphery, resembling an upright submerged turbine-wheel; it receives the smoke at the center and discharges it at the periphery into a tank filled with absorbing liquid.

Electric Condensation, Cottrell Process.—This direct-current electrostatic method for condensing smelter fume is in operation in the roasting and parting departments of Selby lead works.[2] Its mode of operating may be shown in connection with Fig. 5. In the flue *A'*, pairs of electrodes *b*, triangular frames of $\frac{1}{8}$-in. wire wrapped by asbestos, are

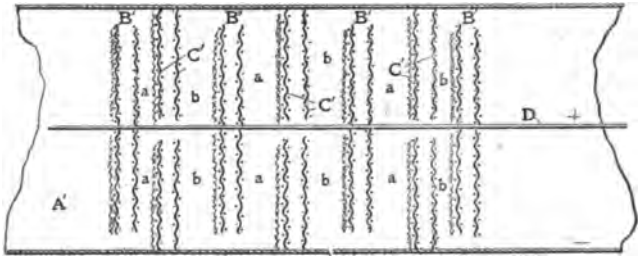

FIG. 5. CONDENSER OF COTTRELL PROCESS.

attached to the positive pole *D* of a high-potential direct-current machine, and pairs of electrodes *a*, sheet-iron $\frac{1}{16}$ in. thick, to the negative pole *B*. A brush-discharge takes place from the wrapped wire-poles to the sheet-poles. Particles suspended in gas-current, passing through the flue, which touch the wire-poles, are electrified, repelled and attracted by the plate-poles. Condensed acid-vapors and particles of dust collect on the negative sheet-iron poles, trickle down and are collected on the bottom of the flue, whence they are removed. The plates appear to withstand cor-

[1] U. S. patent, No. 880,506, March 3, 1908.
[2] *Eng. and Min. Journ.*, 1908, LXXXVI, 375

rosion satisfactorily. The velocity of the gas current may not exceed 4 ft. per second.

Elimination of Sulphuric Acid from Sulphurous Gases.—C. B. Sprague experimented[1] with various substances with the aim to remove sulphuric acid from the sulphurous gases of roasting furnaces in order to enable him to filter them. It is well known that wool or cotton are quickly corroded by sulphuric acid while they readily resist sulphurous gases. Zinc oxide proved successful in absorbing the sulphuric acid from roasting-gases.

Reduction of Sulphur Dioxide by Means of Carbon.—C. Y. De Lay and G. C. Carson have been investigating the reduction of sulphur dioxide in furnace gases by means of carbon.[2] They find that the reduction is easily effected, and that the resulting sulphur-vapor is readily collected as soon as the gases are cooled down sufficiently. Lamp-black, produced by the imperfect combustion of crude oil, is a satisfactory reducing agent.

Condensation of Furnace Fumes and Recovery of Sulphur.—F. R. Carpenter proposes[3] four methods for condensing fumes from smelting works and recovering the sulphur they contain. They are: (1) Separation of SO_2 by freezing and subsequent recovery of S by reduction. (2) Passing the gases through a reducing atmosphere and recovering the reduced S by settling or filtering. (3) Converting part of the SO_2 into H_2S and producing S by the interaction of the two gases. (4) Converting the whole of the SO_2 into H_2S ($2SO_2+2H_2O+3C=2H_2S+3CO_2$; $SO_2+H_2O+2C=H_2S+CO_2+CO$; $SO_2+H_2O+3C=H_2S+3CO$) and producing S from this by means of the Claus-Chance kiln : H_2S+O (from Fe_2O_3)$=S+H_2O$. The paper contains illustrations of the patent obtained by the author (U. S. patent, No. 871,912, Nov. 26, 1907).

Harmful Effect of Smelter Smoke.—R. E. Swain and W. D. Harkins have continued their studies on smelter smoke.[4] Their paper is entitled "Arsenic in Vegetation Exposed to Smelter Smoke." A second paper by W. D. Harkins and R. E. Swain has the title, "The Chronic Arsenic Poisoning of Herbivorous Animals."[5] J. K. Haywood has published[6] a second instalment of his investigations upon the effects of smelter fumes in and around Anaconda, Mont.

Lead Poisoning.—Von Schmidt finds[7] that certain changes in the blood, which are easily recognized, are sure proofs of early stages of lead-poisoning, and form, therefore, a valuable means for diagnosis.

[1] *Min. Science*, 1908, LVII, 53.
[2] *Min. and Sci. Press*, 1908, XCVII, 401.
[3] *Proc.*, Colo. Sci. Soc., 1908, IX, 65.
[4] *Journ.*, Am. Chem. Soc., 1908, XXX, 915.
[5] *Ibid.*, 1908, XXX, 928.
[6] *Bull.* 113, Bureau of Chemistry, U. S. Department of Agriculture, Washington, 1908.
[7] *Oest. Zeit. f. B. u. H.*, 1908, LVI, 175.

Agglomerating Flue Dust.—F. Heberlein has taken out an Austrian patent (No. 31,247) for a method of agglomerating finely divided iron ore, manganese ore, blue billy, flue dust, etc., which consists essentially in adding carbonaceous fuel to a charge and blowing it in a Huntington-Heberlein pot with the object of sintering and driving off volatile matter.[1]

Desilverization.

Pattinson Process.—According to a private communication from A. Wetzstein, Butte, Mont., Sept. 23, 1908, Tredinnick has succeeded in inducing the American Smelting and Refining Company to try his improvements on the steam-Pattinson, or Luce-Rozan, process at the Company's refining plant at Omaha.[2]

Parkes Process.—F. T. Havard in a general article[3] on the suitability of basic refractory materials in metallurgical furnaces, advocates the use of magnesite and chrome-brick in softening and cupelling furnaces, and furnaces for reducing antimony skimmings; calcined magnesite or a mixture of it with litharge is said to form an excellent material for the rammed breast of a cupelling furnace.

W. Morrison patented[4] a modification of the usual Parkes mode of procedure for separating gold, silver and platinum from lead on a laboratory scale of working. It consists briefly in adding with the zinc some potassium cyanide as a flux, bringing the whole to a red heat, allowing to cool, and then separating in the cold the crust from the lead.

B. B. Bayley describes,[5] with illustrations, the desilverization of base bullion by the Parkes process and the working-up of the by-products as practiced at Port Pirie, N. S. W. This is a student's prize paper covering the same ground as Delprat[6] last year. It contains many illustrations and chemical data not found in the older paper.

K. Friedrich examined the losses that take place in the distillation of zinc-silver alloys.[7] He found losses ranging from 0.4 to 5.8 per cent. of the silver present, the losses increasing with the quantity of zinc present and the temperature of distillation. The data agree in a general way with that of Rose.[8]

Electrolysis.—A. B. Betts patented[9] two processes for the treatment of slime obtained in the electrolytic refining of lead by the Betts process. One consists in oxidizing part of the slime, mixing the product with raw

[1] *Oest. Zeit. f. B. u. H.*, 1908, XLVI, 555.
[2] *The Mineral Industry*, XVI, 689.
[3] *Eng. and Min. Journ.*, 1908, LXXXVI, 802.
[4] U. S. patent, No. 890,160, June 9, 1908.
[5] *Trans.*, Aust. Inst. Min. Eng., 1907, XII, 79.
[6] *The Mineral Industry*, XVI, 670.
[7] *Metallurgie*, 1908, V, 593.
[8] *Trans.* I. M. M., 1904-05, XIV, 394.
[9] U. S. patents, Nos. 891,395 and 891,396, June 23, 1908.

slime, and fusing the mixture, when all the antimony and lead are to enter the slag formed which separates readily from remaining doré silver; the latter is to be purified by treatment with hydrofluoric acid. In the other process, the slime is oxidized, mixed with sulphur-bearing material (Sb_2S_3), and fused. The products are said to be copper-bearing matte-doré silver, and lead-bearing antimony-slag.

G. Mardus carried on a research[1] upon refining lead electrolytically with a hydrofluoboric acid (HBF_4) which is to replace the hydrofluosilicic acid (H_2SiF_6) of the well-known Betts process. In the latter there are, it is stated, two inherent losses, as about 8 per cent. of the H_2SiF_6 used is lost when the electrolyte $PbSiF_6$ is prepared by neutralizing H_2SiF_6 with $PbCO_3$, and as leadfluosilicate is decomposed after a while by hydrolysis ($PbSiF_6+H_2O=PbF_4+4HF+SiO_2$), when PbF_2 and gelatinous SiO_2 settle out and go into the anode mud. Hydrofluoboric acid, which may be considered as a complex compound of HF and BF_3, may be prepared by dissolving boric acid, $B(OH)_3$, in hydrofluoric acid, HF; the common impurities, H_2SO_4 and HCl, are removed by the addition of PbO, which also neutralizes almost wholly the HBF_4 forming readily soluble leadfluoboride, $Pb(BF_4)_2$. The electrolyte for the experiments was made up of a 7.2-normal solution of HBF_4 containing 0.56 per cent. Pb, and of a solution of PbF_2 which was 0.7-normal with reference to the free acid and contained 42.33 per cent. Pb. Sheet-lead served as anode and sheet-copper as cathode, the bath was air-stirred. Tests made, varying the current-densities, the composition of electrolyte and the temperature, showed that, while the cathode deposits on the whole were smooth and coherent, short-circuiting occurred along the rim of the anode. The use of 0.011 per cent. gelatine in an electrolyte with 7.8 per cent. Pb and 8.8 per cent. acid gave with a current density of 0.005-0.01 amperes per sq.cm., and a pressure of 0.09-0.28 volt., smooth deposits and no short circuits. Reducing the free acid from 8.8 to 4.4 per cent. still gave good deposits; reducing the lead-content to 3.9 per cent. and the free acid to 1.8 per cent., made 0.01 amp. per sq.cm. the highest permissible density; with 7.8 per cent. Pb and 8.8 per cent. acid the current density could be increased up to 0.02 amp. per sq.cm. In the process the electrolyte became enriched in lead and impoverished in acid. The efficiency of the process without the use of gelatine about 98 per cent.; with gelatine it rose to 100+ per cent. This high figure is explained by the deposition of organic matter with the lead. To study the influences of metals, usually found in base bullion, upon the process, refined lead was alloyed with Ag, Cu, Sb, Bi, Sn and Zn, and electrolyzed. Pure

[1] Doctorate-Thesis, Berlin, 1908.

cathode lead was obtained from all alloys excepting with the tin-alloy, a large part of the tin going over with the lead.

Cupellation.—C. H. Fulton has carried on some investigations regarding cupelling temperatures in a muffle-furnace.[1] While the data have to do mainly with assaying, they embody many valuable points that have a bearing upon large-scale work. The paper, which contains nine charts with graphic representations of the observed data, concludes with a summary in which it is shown that for pure lead the muffle-temperatures should be at least 800 deg. C., and preferably 850 deg., for uncovering a button; that the temperature may be lowered to 770 deg. during cupellation; that it must be raised to 830 deg. for finishing, but not be carried beyond 910 deg.; that the temperature of the cupelling lead is higher than that of the muffle; that surfusion to the extent of 77 deg. may take place; that beads from surfused silver are free from lead; that feather-litharge forms an indication of a proper cupelling-temperature provided the draft is not excessive; and lastly that the draft in a muffle has to be regulated as well as the temperature.

K. Friedrich has carried on some preliminary investigations upon the losses of silver that take place in cupelling.[2] They show that the losses increase with the temperature, the amount of lead, and that the loss is greatest toward the end of a cupellation.

G. Melvill publishes[3] some notes about the practice of cupelling gold-lead bullion in South Africa. The tests used are oval, 3 ft. 5½ in. long by 2 ft. 3 in. wide by 7 in. deep. The body is rammed with a mixture of 150 lb. mabor, 75 lb. White's English portland cement, and 75 lb best crushed fire-brick that has been moistened with 12 per cent. water; the material for the breast is made up of 60 parts mabor, 30 cement and 30 fire-brick. The author lays stress on passing the dry materials through a 40-mesh sieve and the moistened mixture through a 14-mesh sieve. The test, made according to English practice, has four litharge-holes, 1½ in. diam. each, with a channel sloping toward it; three channels are 1½ in. wide and ½ in. deep; the fourth, ½ in. wide and 2 in. deep, serves to finish the charge. A test will work 3½ bars of 1000 oz., per hour; its life varies; a record figure is 520 bars of 1000 oz. each. The litharge produced assays 1.5 oz. Au per ton. The direct loss in gold agrees within 6 oz. of that called for by the lead bullion assay. The fineness of the gold bullion is 0.842.

Refining Silver.—B. Neilly carried on some experiments in refining silver bullion which contained arsenic and antimony.[4] The method of

[1] *West. Chem. and Met.*, 1908, IV, 31.
[2] *Metallurgie*, 1908, V, 595.
[3] *Journ. Chem. Met. and Min. Soc. South Africa*, 1908, IX, 157.
[4] *Journ., Can. Min. Inst.*, 1908, XI, 586.

Rose[1] of blowing air through the molten metal held in a graphic crucible, was not successful, as its tenor could not be raised above 92 per cent. Ag. It was then decided to remove any nickel, cobalt and copper present with as much of the arsenic and antimony as possible in the form of speiss, and to refine by blowing air upon the surface. To this end the silver was melted down, iron added in the form of nails until they ceased to be dissolved, the crucible removed from the furnace and cooled quickly, which gave a clean separation of speiss and silver. Melting down again the purified silver with some sand-borax mixture (2:1) to form a thin cover and blowing upon the metal air at a sufficient pressure to cause a depression of about ¼ in., brought the silver quickly to a fineness of 999.85. The assay data of one set of tests are as follows: Original bullion: Ag 80.9 per cent., Sb 9.6 per cent., As 7.4 per cent. Original bullion blown for 1 h. 20 min.: Ag 88.33 per cent., Sb 9.54 per cent., As 2.13 per cent. Original bullion treated with iron to form speiss: Ag 94.02 per cent., Sb 2.98 per cent., As 3.00 per cent. Bullion refined by surface-blast after removal of speiss: Ag 99.985 per cent. The losses in silver were under 1 per cent. In one case, in which they amounted to 0.79 per cent., the speiss-loss was 0.08 per cent., the slag-loss 0.12 per cent., and by difference the volatilization-loss 0.20 per cent.

The Cost of Silver-Lead Smelting.[2]

By Walter Renton Ingalls.

The cost of smelting and refining in the United States ranges widely among the various plants, depending upon the size and nature of the plant; the cost of labor, fuel, fluxes, and material; the character of the ores smelted, etc. Thus, nine plants during the same period of six months, a few years ago, showed costs of smelting, referring to the blast-furnace operation only, per ton of charge (ore and flux) which ranged from $2.50 to $4.80. This appears in the following list: A, $3.418; B, $2.525; C, $3.260; D, $3.331; E, $3.754; F, $3.429; G, $3.929; H, $4.039; I, $4.781. Average, $3.607.

In the treatment of the argentiferous ores of the West, the present practice is to roast only those that are low in lead, and charge raw into the blast furnace the rich galena. The cost of roasting is $2 to $2.50 per ton of ore roasted. The cost of smelting a ton of charge in a large modern plant, under favorable conditions, is about $2.50, of which about 84c. is for coke and $1.66 for labor, power and supplies. The expense of administration amounts to about 16c. additional. Consequently, the total

[1] *The Mineral Industry*, XIV 420

[2] This article is a condensation of one published originally in *Eng. and Min. Journ.*, of Aug. 15, 1908. While the present article is greatly condensed from the original, certain points have been amplified and figures have been brought up to date.

cost per ton of charge (ore and flux) is about $2.66. If the ore amounts to 80 per cent. of the charge, which corresponds to the ordinarily good practice, the cost per ton of ore is about $3.33.

An average of the work of many large smelting plants shows that for every ton of charge smelted in the blast furnace, about 0.4 ton of material (ore and matte) must be roasted. The cost of roasting ranges from $2 to $2.50 per ton. Taking the lower figure, in view of economies that have been effected by the blast-roasting process, we may figure that on the average 80c. is to be added to the cost of smelting, making the total cost per ton of ore about $4.12. Figuring on the same basis of 80 per cent. of ore in the charge, the average for the nine works previously mentioned would be $3.607÷0.80=$4.50, approximately.

Interest Charges, Amortization, Etc.—Works, capable of smelting 1000 tons of ore per day, or roughly 330,000 tons per annum, cost nearly $1,000,000, or about $3 per ton of annual capacity. Reckoning amortization at 10 per cent. per annum, and interest on the investment at 6 per cent., the smelter must add 48c. per ton of ore smelted on account of these fixed charges. Moreover, the smelter is bound to carry a large stock of ore on hand. Assuming that the works which is treating 330,000 tons of ore per annum has always 30,000 tons on hand, and that the average value of the ore is $30 per ton, the interest charge on each ton of ore smelted is upward of 15c. A month's delay in realizing on the products adds 15c. more.[1] This makes a total of $5.28 up to the production of base bullion. The smelter recovers about 95 per cent. of the lead and 97 per cent. of the silver in the original ore. He pays the miner for only 90 per cent. of the lead and 95 per cent. of the silver, wherefore he has a certain leeway on these metals, as he may have also on gold of which he pays the miner for only 95 per cent., but recovers 100 per cent.[2]

Copper.—Besides the gold, silver and lead of the ore there is a certain amount of copper, modern practice demanding the presence of 0.5 to 1 per cent. in the charge in order to insure a clean slag. This copper goes partly into the slag, and partly into the base bullion (from which it is recovered during the refining process), but chiefly it is obtained in the form of matte, which is concentrated up to about 40 per cent. copper and then is despatched to Omaha, where it is converted to blister copper. The converting of this leady matte is more costly than the treatment of ordinary copper matte, and indeed throughout the lead smelting process cop-

[1] It will appear subsequently that this estimate of the time that ore and crude metal are in process of treatment is under, rather than over, the average. As a matter of fact smelters roughly figure interest on the basis of 90 days.

[2] The actual extraction of lead is less than 95 per cent., but in good practice it is 95 per cent. on the basis of fire assay, on which much of the ore is purchased, so it is proper to figure 95 per cent. However, this is drawing it rather tightly upon the smelter, and considering the further loss of 1 per cent., which the lead suffers in refining the smelter who pays for 90 per cent. of the lead in the ore does not obtain any great margin on this item, nor does he on the purchase of the silver. The smelter does not really recover 100 per cent. of the gold, although he may apparently do so, and even more, because of the cumulative effect of small amounts of gold, too little to figure in the ore settlements, which give the smelter more to start with than his books show.

per is subject to high losses, especially in the slag of the first smelting, which follows from the common metallurgical principle that losses are quantitatively constant (or nearly so) and proportionately variable. Consequently in the treatment of an ore so low in copper as 0.5 to 1 per cent. the precentage of loss is large. It may be generalized as 30 per cent. It is for this reason that the smelter makes so large a deduction from the copper in the ore (1.3 units from the wet assay) and pays for it at 3@7c. less than the price of refined copper at New York. Of course it will be understood that the deduction of 1.3 units pertains to ores that contain sufficient copper to deserve payment, and that those ores when mixed with many others that contain no copper give an average furnace charge with 0.5 to 1 per cent. copper. It may be explained also that all of the copper does not finally appear as refined metal, a fairly large quantity being obtained and marketed as bluestone. In the generalization which I am attempting it is impossible to go far into these details.

Freight and Refining.—The products of the smelteries are base bullion and lead-copper matte. The latter goes to Omaha for converting and the blister copper thence is passed on to Perth Amboy for refining. The base bullion goes to Denver, Omaha, Chicago and Perth Amboy. As in the case of smelting there are differences among these works as to the cost of refining and other conditions, but inasmuch as the prices for lead and copper are based on the market at New York it is best to confine attention to the refining and handling of base bullion at that center.

The freight rate on base bullion from Salt Lake City to New York is $10.80 per ton; from Denver and Pueblo to New York it is $6.40 per ton. The cost of refining is $6@6.50.[1] Other costs are lighterage, $0.625; selling, $0.40; miscellaneous, $0.325. This gives a total of $7.60 exclusive of freight. The cost of a lead refinery is about $6.66 per ton of annual capacity, on which amortization at 10 per cent. and interest at 6 per cent. come to $1.07, making the total cost of refining, lighterage, selling, etc., about $8.67. Consequently the charges on a ton of ore smelted at Salt Lake and yielding 10 per cent. of lead are as follows: Smelting, $5.28; freight on bullion, $1.08; refining, etc., $0.87; total, $7.23.

In addition to this total, the reports of the American Smelting and Refining Company indicate a general expense amounting to 25@40c. per ton of ore smelted, the smaller figure being achieved in the more recent years. Consequently we may put the total cost of smelting and refining at about $7.50 per ton of ore.

In custom-refining it is the practice to pay the smelter, i.e., the seller of the base bullion, for the gold at $20 per oz.; for the silver at the New York price less 1c. per oz. (this is 98 per cent. when silver is worth 50c.

[1] At Chicago the cost is only $4, and under favorable conditions lead refining should be done at that figure.

per oz.); and for the lead at 98 per cent. of the New York price. The actual extraction of lead is 99 per cent. The loss of silver is so small that it may be disregarded for present purposes.

American Smelting and Refining Company.—The reports of this company throw but little light upon the subject of the cost of smelting. Its statement of assets, gross earnings, profits, etc., for a series of seven years, is given in the accompanying tables. Before proceeding to discuss these figures, it is important to make certain explanations. In each year the figures are for the fiscal period ending April 30, wherefore the major part of the period pertains to the preceding calendar year. The item that I have entered as "Repairs" is given in the reports of the company as "Ordinary Repairs and Betterments." What I have called "General Expense" includes all of the general expenses of administration, together with interest and taxes. "Net Earnings," so called by the company, are evidently not properly designated, being merely the operating profit. The true profit, or actual net earnings, appears later in what the company calls "Net Income." Under "Improvements," I have entered what the company calls "Appropriation for Extraordinary Improvements and New Construction." From the uniformity of this account, I judge that it represents chiefly the new construction undertaken to replace worn-out or antiquated plant; in other words, it is in this way that the company makes good the depreciation of its property, which otherwise would have to appear in an amortization account.

I. A. S. AND R. CO. COMPARATIVE STATEMENT OF ASSETS.

Year.	1903	1904	1905	1906	1907	1908
Property account	$86,845,671	$86,845,671	$86,845,671	$86,845,671	$86,845,671	$86,845,671
Investment account	1,028,598	1,680,306	(a)3,982,576	(a)4,179,915	(a)3,810,595	3,950,088
Metal stock (Au. Ag. Pb. Cu.)	18,010,687	17,032,900	16,418,543	19,415,200	18,251,587	17,519,664
Material, fuel, flux	1,107,253	1,224,688	1,118,902	1,114,893	1,317,544	1,380,742
Cash	2,339,154	4,047,423	4,636,649	4,757,929	6,706,984	5,629,034
Total assets	$109,331,362	$110,830,387	$113,002,340	$116,313,607	$116,932,381	(b) $115,825,725

(a) Does not include 177,510 shares of the common stock, American Smelters Securities Company, par value $17,751,000.
(b) Includes $500,526 as "net current assets."

II. A. S. AND R. CO. COMPARATIVE STATEMENT OF INCOME ACCOUNT.

Year.	1902	1903	1904	1905	1906	1907	1908
1. Earnings	$7,038,682	$9,403,711	$9,425,443	$10,506,683	$11,665,886	$13,250,058	$9,403,282
2. Repairs	791,306	770,854	818,141	878,648	828,582	976,535	933,130
3. General expense	1,385,757	1,056,786	701,729	729,224	675,945	763,854	836,866
4. Net earnings	4,861,619	7,576,786	7,905,573	8,898,811	10,161,358	11,509,669	7,633,287
5. Employees' fund			91,254	216,816	449,204	540,420	*Nil*
6. Improvements		655,683	597,582	425,289	938,100	1,054,996	622,096
7. Metal account	1,300,000	1,500,000	500,000	637,795	*Nil.*	*Nil.*	*Nil.*
8. Net income	3,561,619	5,421,103	6,716,737	7,618,912	8,774,055	9,914,253	7,011,191
9. Dividends	3,500,000	3,500,000	4,750,000	6,000,000	6,750,000	7,000,000	
10. Surplus for year	61,619	1,921,103	1,966,737	1,618,912	2,024,055	2,914,253	11,191
11. Total surplus	2,951,968	4,873,071	6,839,808	8,458,720	10,482,775	13,397,028	13,408,219

Except in its recent statement to the New York Stock Exchange that the average amount of ore smelted is 3,500,000 tons per annum, the Smelting Company has never made any statement of its production of metals or amount of ore smelted. The nearest that it has come to communicating this important information was in the report for the fiscal year ended April 30, 1903, wherein it is stated that the volume of business transacted during the year is reflected in the following figures: Metal content of ore purchased: gold, 1,025,132 oz.; silver, 62,389,438 oz.; lead, 492,960,350 lb.; copper, 47,919,666 lb. Fuel consumption: coal, 544,790 tons; coke, 433,431 tons; fuel oil, 3,523,904 gal. Freight traffic: total tonnage moved, 4,434,484.

Tonnage of Ore Smelted.—These data enable us to arrive approximately at the amount of ore purchased, and we may assume that the amount smelted was substantially the same. It is a fair assumption that the ores were purchased in approximately the proportions required to make a suitable smelting mixture, and that the lead content was in the neighborhood of 10 per cent. of the total ore. The purchase of 246,480 tons of lead would therefore imply 2,464,800 tons of ore. Some of the copper purchased was included with the lead charge, but some was smelted separately. As to this particular I can do no more than surmise that 100,000 tons of copper ore may have been smelted separately, and that the total amount of ore smelted by the company in this year was about 2,564,800 tons. It will appear that this estimate is probably not far out of the way. In 1901 the American plants of the company alone were smelting at the rate of about 2,000,000 tons of ore per annum, and from that time onward business increased. In the fiscal year ending April 30, 1903, the total movement of freight is given as 4,434,484 tons. Deducting 991,221 tons for fuel (allowing 13,000 tons for the oil) and 270,439 tons of lead and copper, we have left 3,172,824 tons for ore and limestone, of which the latter would normally be about one-sixth, deducting which there remains 2,644,018 tons for ore. There is some traffic in matte and other products from one works to another, but making allowance for such duplications and overestimates it seems not unreasonable to assume 2,500,000 tons of ore smelted. On this basis, namely 2,500,000 tons, it appears that the total actual profit to the company in the year ending April 30, 1903, was a little less than $2.20 per ton of ore. Inasmuch as this is determined by making the tonnage the divisor of the whole profit of the company and it is not to be doubted that even in 1902-03 the company was making handsome returns from its mercantile and investment accounts, I believe it is reasonable to assume that its profit in smelting, properly considered, at that time may have been about $2 per ton. Mr. Edward Brush, of the company, before the Ways and Means Com-

mittee, Dec. 16, 1908, stated that in the fiscal year ended April 30, 1908, the company smelted 3,372,750 tons of ore. The net profit in that year was $7,011,191. Consequently the total profit per ton of ore was a little less than $2.08. The actual smelting profit was, of course, something less, because the company realizes more or less from its various ventures that are not to be referred directly to its smelting business.

Average Grade of the Ore.—The figures given for the fiscal year ending April 30, 1903, also convey valuable information respecting the average metal contents of the ore smelted in the United States and Mexico. Proceeding still on the assumption that the total tonnage was 2,500,000, the average was 0.41 oz. gold, 24.95 oz. silver, 197.4 lb. lead, and 19.17 lb. copper. The substantial accuracy of this deduction is confirmed by the report of the Census for 1904. (The Census confusingly designates the year as 1905, because its investigation was made at that time, but the investigation related to 1904.) According to the Census, the amount of argentiferous ore treated in 1904 was 2,271,724 tons, which yielded an average of 0.42 oz. gold, 16.53 oz. silver, 198 lb. lead, and 22.72 lb. copper. It is to be remarked that the figures of the Census relate only to ore smelted in the United States, while my previous figures have included the ore smelted both in the United States and in Mexico. Moreover, the latter figures are for contents of the ore purchased, while the Census figures are for yield of the ore. However, the agreement is sufficiently close to confirm the belief that my estimate is a close approximation.

Another interesting deduction may be made from the statistics of the Smelting Company for the year ending April 30, 1903. During that period, the average price for silver was 50¼c. per oz.; of copper, 12.452c. per lb.; of lead, 4.147c. per lb. Computing ore of the average grade shown for the year ending April 30, 1903, on the basis of 100 per cent. of the metal contents at the average New York prices for silver, lead and copper, and $20.56 per oz. for gold (which is what the United States Smelting, Refining and Mining Company realized for its product in 1907, although the coinage value of gold is $20.67 per oz.) it appears that the maximum gross value of this average ore was $31.54 per ton, itemized as follows: 0.41 oz. gold @$20.56, $8.4296; 24.95 oz. silver @50¼c., $12.5374; 197.3 lb. lead @4.147c., $8.1820; 19.17 lb. copper @12.45c., $2.3867. Total, $31.5357. Having already shown that the average profit per ton of ore smelted in that period was probably about $2, the actual net profit to the smelter was a little more than 6¼ per cent. of the ore value.

Subsequent Increase in Profits.—It is impossible to follow analytically the subsequent history of the company in any way that has a very sound

foundation. The reports show a marvelous increase in the profits, which were $5,421,103 in 1902-03 and $9,914,253 in 1906-07. During this period of four years the amount of ore smelted by the company increased greatly, but there is no reason to surmise that it increased in the same ratio as the profits; indeed, there is sufficient evidence to warrant me in saying positively that it did not, and that if the tonnage of ore smelted in each year were made the divisor of the net profits reported the quotients would be steadily increasing, up to the last year or two. However, any such figuring would be misleading because the company has undergone great expansion and derived greatly increased profits from sources that are not properly referable to the direct smelting operations. The company avers that it has not increased treatment charges, and there is much evidence in support of that assertion.

Explanation of Increasing Profits.—In directing attention to the subject of the increasing profit shown by the reports of the Smelting Company, it is important to consider a variety of conditions. It is well known that it is much more economical to smelt on a copper basis than on a lead basis. The difference in favor of the former is fully $1 per ton of ore. Consequently, the more copper ore to be smelted, the more the profit, and the increasing net earnings of the Smelting Company are doubtless due to some extent to the increased amount of ore smelted on the copper basis. It is also well known that the margin on ore purchased in Mexico is much greater than on American ores, and a large part of the profit of the Smelting Company is derived from its Mexican business, which has been rapidly increasing. The lowest margin, probably, is realized by the smelteries in Colorado, which until latedy have treated in the neighborhood of 1,000,000 tons per annum and operated rather uniformly at that rate. A few years ago, the profit in smelting in Colorado was only about $1 per ton, and probably it is no larger at the present time. It is claimed also that the profit in smelting in Utah has only been about $1 per ton since competition has been active at that point. On the other hand the profit at non-competitive points and in Mexico must be large.

The increase in the earnings of the Smelting Company has also been promoted without doubt by its profit-sharing system, which was designed to increase efficiency and has had that effect. The company has benefited from economies in administration, as is clearly shown by the decreasing amount to the account of general expense. Furthermore, it has derived great advantage from the introduction of metallurgical improvements, such as the Huntington-Heberlein process, and the concentration of operations at the most economical plants. Finally, we come to the question of metal stock account, wherein the purchaser of ores may lose or make a great deal through fluctuation in the value of the metals. In the long run

such fluctuations are expected to balance, and temporary gains or losses are commonly charged to an account representing quotational profit or loss. In a long upward trend of prices, a buyer of ores may realize a great profit; and similarly in a sharp decline, he may suffer an immense loss. From 1901 to the end of 1907 the general trend of the metal markets was upward, and undoubtedly the greatest factor in the increase in net income up to April 30, 1907, was the appreciation in the value of metals on its hands, just as since June, 1907, its net income suffered severely from the decline. The company carries in its statement of assets an item of "metal stock" ranging from $16,418,543 to $19,415,200, which represents its valuation of ores and metals on hand. The nature of its business requires that large quantities of ore and crude metal be in stock at all times. It appears from the data deduced in this article that the stock necessarily carried is from 20 to 25 per cent. of the annual turnover; in other words, the ore and its products are in process of treatment and in transportation for 2½ to 3 months.

Division of Ore Value.—Now let us see what division is made of the value of an ore assaying 0.41 oz. gold, 24.95 oz. silver, 197.3 lb. lead and 19.17 lb. copper, which was the composite of all the ore bought by the American Smelting and Refining Company in 1902-03. The smelter and refiner probably realize from this ore approximately as follows: Gold, 0.41 oz. @$20.56, $8.43; silver, 24.95 oz. ×0.97 @50¼c., $12.16; lead, 197.3 lb.×0.94 @4.147c., $7.69; copper, 19.17 lb.×0.7 @12.45c., $1.67. Total, $29.95.

The expenses from the time of receipt of the ore at the smelting works to the sale of the refined metals are approximately as follows:

1. Smelting, 1 ton @ $4.50	$4.50	7. General expense	$0.40
2. Converting 40 lb. copper matte @ 0.7c.	0.28	8. Amortization	0.25
3. Freight on 190 lb. lead bullion @ 0.43c.	0.82	9. Tie-up of metals	0.30
4. Freight on 13½ lb. copper bullion @ 0.5c.	0.07	10. Metal account	0.30
5. Refining 190 lb. lead bullion @ 0.38c.	0.72		
6. Refining 13½ lb. copper bullion @ 0.7c.	0.09	Total	$7.73

1. As previously computed. 3. The rate of 0.43c. is the mean of the rates from Salt Lake and Pueblo; this assumption is necessarily arbitrary. 4. In this case also the assumption of freight rate is necessarily arbitrary. It is intended to cover all freight charges on copper from the time of leaving the first smelter. Copper matte goes to Omaha from East Helena, Salt Lake, Denver, Pueblo, and elsewhere—even from Perth Amboy—and the copper bullion thence goes to Perth Amboy. Probably the assumption of 0.5c. per lb. to cover all of this movement is too low. 7, 8. These figures are deduced from the reports of the American Smelting and Refining Company; the allowance for amortization appears to be too

low. 9. As previously computed. 10. This appears to be the average allowance that has been made by the American Smelting and Refining Company, as insurance against depreciation of metals on its hands.

Inasmuch as the smelter is supposed to realize a profit of $2 per ton of ore, the total deduction for its account must be $7.73+$2=$9.73, and from the value of the ore, $29.95, there is left $29.95—$9.73=$20.22 to pay for the ore and the freight upon it to the smelting works.

Now let us see how that would figure out to the producer. We may assume a settlement on the lines of the following: Gold, 0.41 oz. @$19.50, $8; silver, 24.95 oz.×0.95×50½c., $11.91; lead, 197.3 lb. @2c., $3.95; copper, 19.17 lb. @5.45c., $1.04; total, $24.90; deducting a treatment charge of $4.68 leaves $20.22 as the net value to producer. This corresponds to an ore contract reading "Gold to be paid for at $19.50 per oz.; silver at 95 per cent. of the New York quotation; lead at 40c. per unit; copper at the New York quotation, less 7c. per lb.; treatment charge, $4.68 per ton; neutral basis; delivery at smelter's works." This has a familiar sound, except that so small a percentage of copper is not always paid for, but it must be remembered that I am here figuring on a composite ore, the copper of which is obtained chiefly in special classes of a higher average.

Conclusions.—After a consideration of the data, it is impossible to escape the conclusion that the great increase in the net earnings of the American Smelting and Refining Company from year to year, is to be attributed to: 1. Enlargement in the volume of business. 2. Institution of economies (a) in administration; (b) through centralization of operations; (c) through metallurgical improvements; (d) through increase in operative efficiency. 3. Appreciation in the value of metals, due partly to natural causes and partly to manipulations by the company. The profits on exempt lead, and on contracts with the producers of lead ore, whereby the value in excess of a certain price per pound is divided between the producer and the Smelting Company, must attribute largely to the treasury of the company. Since the middle of 1907, the depreciation in the value of metals has offset some of the gain previously realized. 4. Increase in the amount of ore smelted on the copper basis, which is more profitable than the lead basis. 5. Increase in earnings of subsidiary companies, such as the Steamship Company. 6. Earnings from investments of surplus, e.g., the preferred stock of the American Smelters Securities Company. 7. Profits from investments, e.g., the sale of a portion of its holding of the stock of the United Lead Company, carried into earnings for the year ending April 30, 1907.

The position of the Smelting Company being so strong in many respects, and the surplus which it carries being so large, the company may be forgiven for not writing off anything for amortization of its

plants. As I have previously pointed out, the outlay made on account of extraordinary improvements is of the nature of an amortization account, but the amount expended so far in this way is of doubtful sufficiency. The smelteries and refineries now owned by the company must be worth in the neighborhood of $15,000,000, i.e., it would cost that amount to replace them. The average amount expended for extraordinary improvements during the five years ending with April 30, 1907, was a little less than $750,000 per annum, which is only 5 per cent. of the physical value of the plants. This, it seems to me, is an insufficient allowance for amortization.

According to the statement filed by the company with the New York Stock Exchange, in January, 1909, its smelteries and refineries were the following:

SMELTERIES

Place.	Plant.	Furnaces.	Annual Capacity.
Denver, Colo.	Globe	7	322,000
Pueblo, Colo.	Pueblo	7	328,000
Pueblo, Colo.	Eilers	6	295,000
Durango, Colo.	Durango	4	146,000
Leadville, Colo.	Arkansas Valley	10	509,000
Salt Lake, Utah	Murray	8	523,000
East Helena, Mont.	East Helena	4	235,000
Omaha, Neb.	Omaha	2	82,000
Chicago, Ill.	National	2	60,000
Maurer, N. J.	Perth Amboy	3	140,000
El Paso, Tex.	El Paso	10	492,000
Monterey, Mex.	Monterey	10	460,000
Aguascalientes, Mex.	Aguascalientes	10	720,000
Chihuahua, Mex.	Chihuahua	3	153,000
		86	4,465,000

REFINERIES

Place.	Plant.	Lead Tons.	Copper Tons.	Gold and Silver oz.
Omaha	Omaha	156,000	.....	36,000,000
Chicago	National	84,000	.....	16,400,000
Maurer	Perth Amboy	66,000	66,000	36,000,000

The annual product of the refineries is about as follows: Gold, 1,250,000 oz.; silver, 66,000,000 oz.; lead, 225,000 tons; copper, 66,000 tons.

Lead Mining in Missouri and Idaho.

By J. R. Finlay.

Missouri is the first in the list of States in the production of lead ore, and first in that of zinc ore. The mining is confined to two districts, the Southeast and the Southwest. In both fields the external conditions are

favorable. The cost of living is low, labor abundant, fuel and transportation cheap and markets close at hand. The internal factors also are favorable to low costs. The depths reached are not great, and the orebodies are fairly large. In both fields, also, the ores are favorable for water concentration.

Mining in southeast Missouri is based on orebodies that carry an average of about 5 per cent. in metallic lead, or a little more. The ore is called disseminated from the fact that the galena is often sprinkled through the limestone, although usually most of the lead is confined to rich streaks. The ore concentrates well and can be turned into a 65 to 70-per cent. product, with a saving of 80 per cent. Commercially speaking, therefore, the ore yields about 4 per cent. net lead. Developments have proved that the orebodies are exceedingly persistent and extensive. The formation lies approximately flat, though grades of from 3 to 10 per cent. are not uncommon. The ore now being mined occurs in the lower 100 ft. of the St. Joseph limestone, and often at the very bottom of that formation in contact with an underlying sandstone. Occasionally it happens that in the 100 ft. just mentioned, there are successive enrichments making workable orebodies one above the other. In this case more than one level may be necessary. But it is more common to find only one large irregular sheet of ore immediately above the sandstone, so that it can all be worked from one level; although sometimes the ore may shoot up some distance above the general level. The upper orebodies are relatively unimportant.

In the Flat River district proper, these orebodies are arranged in several parallel zones trending northwest. These zones lie in a space about 3½ miles wide from northeast to southwest, and about nine miles from northwest to southeast. It is expected that these zones will be extended materially both to the northwest and to the southeast. I think there is also good reason to expect that other zones will be developed southwest of those now worked. There are some indications already of three such new zones at the west end of the district. I do not know why the ore follows this northwest course. I have never been able to see any system of persistent fissuring in that course. Most of the fissures have a course of east-west to northeast-southwest. These fissures have a most obvious relation to the orebodies which often follow them out long distances on either side of the real ore channel. Fig. 1 shows this relation in plan. The ore zone may carry some lead scattered through the rock on both sides of the workable channels, which may be only 5 ft. wide. The fissures are apparently the source of the ore from which it has fed out into the surrounding rocks. The richest ore, therefore, is right at the fissure, and it fades out on either side, so that midway between fissures the ore

may be too poor to work. Fig. 2 is a longitudinal section of the ore zone across the fissures, showing this relation. The ore is workable to a thickness varying from 6 ft. to as much as 100 ft.

It will be evident from the above that to follow the ore underground, it is almost necessary to stope the ore as you go. There is enough vertical irregularity to prevent following the ore successfully by horizontal drifts; and there is enough horizontal irregularity to make it impossible to keep in the channel, unless you are prepared to follow up each turn. If the

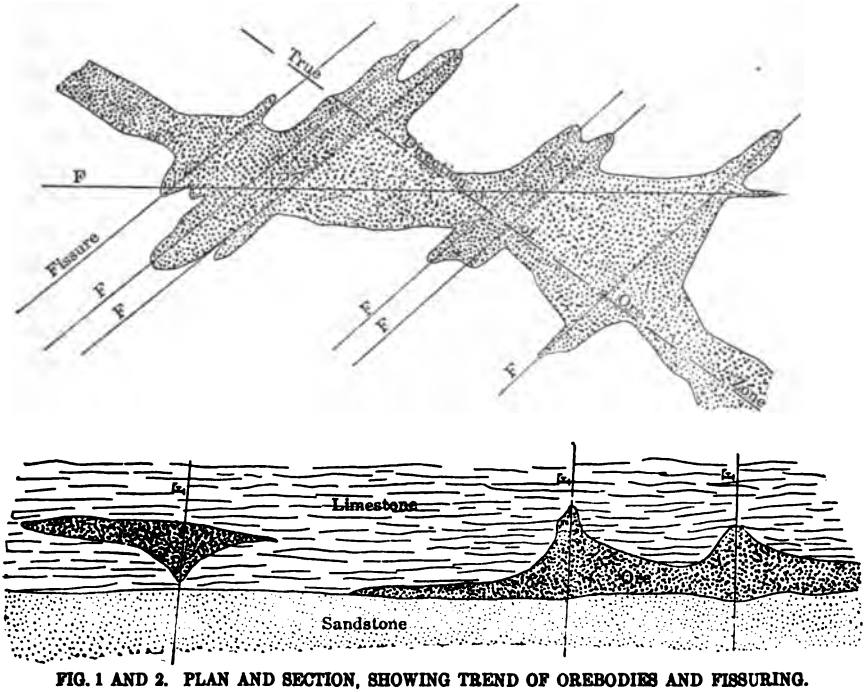

FIG. 1 AND 2. PLAN AND SECTION, SHOWING TREND OF OREBODIES AND FISSURING.

ore rises you must be prepared to go up after it; if it sinks you must go down after it.

The above description refers especially to the mines in the vicinity of Flat River only. At Bonne Terre, the orebodies are a little different, in that the longer axis there seems to extend northeast-southwest, instead of northwest-southeast. These orebodies are northeast from the ore zones of Flat River. It now seems probable that a connection will be established between Bonne Terre and the Flat River orebodies.

The most difficult part of actual mining operations is the preliminary exploration by drilling. This determines the depth to which the shafts

must be sunk, and their location. Usually only one level is necessary, but the fact that the ore does not lie exactly flat, makes some provision for hauling cars up and down hill necessary. This can best be done, I believe, by electric haulage. This has been installed at one of the Federal plants and is very effective. Provision must also be made sometimes for secondary pumping to raise water from depressions that may reach lower than the shaft-pumping station. The stoping is very simple. No timbers are used. Round pillars of ore are left, containing 10 to 15 per cent. of the ore. It is often possible to leave pillars in the poorer parts of the deposit by laying out the main entries so as to follow the rich ore along the fissures. Underground diamond drilling is necessary in some mines to prospect ahead for water channels. These are open fissures that carry so much water that, if broken into carelessly, they make disastrous gushes. Some shafts are pretty wet, making 1300 to 1500 gal. of water a minute. The usual output from each shaft is about 300 tons a day. This output may be greatly exceeded, however, by the use of electric haulage so as to cover a large area from one opening. Ventilation may be secured by drilling large churn-drill holes from the surface. The cost of mining, hoisting and pumping is from $1 to $1.50 per ton. To this may be added 10c. a ton for drill prospecting, and about 10c. a ton for hauling the ore to the mill. The total cost of ore is, therefore, from $1.20 to $1.70 at the mill.

The milling practice is now pretty well established. The ore is ground to 9 mm. Everything smaller than 9 mm. is screened out as soon as the ore passes the crusher. When crushed, the ore is screened to various sizes, from 9 to 2 mm., and this product jigged. The tailings from the coarser jigs are all reground. The material below 2 mm. is classified and treated on Wilfley tables, as are also the reground tailings. Middlings from the tables are also reground in Huntington mills and treated on Frue vanners. The cost of milling in a 1000-ton plant is from 30 to 75c. per ton. The cost of a concentrating mill, together with a power plant for the mines, may be estimated at $500,000 for 1000 tons capacity. The new plant built by the Federal Lead Company is expected to handle about 2400 tons a day. It is built of steel and concrete, has a large air-compressing and electric plant, and elaborate crushing and sampling arrangements. It cost $900,000.

The average cost of producing pig lead for the St. Louis market from this field seems to be from 3 to 3.25c. per lb., but in the boom period of 1906-7 it is doubtful if any of the mines were producing it for less than 4 cents.

The Southwest Missouri zinc and lead district is very extensive, ore having been mined over an area of perhaps 2000 square miles, but within

this extensive field by far the greater part of the production has come from three or four localities. Of these the most important may be called the Webb City zone, which is said to have produced about one-half of the entire output of the field. In the immediate vicinity of the city of Joplin, there are extensive mineralized zones extending in a northwest and southeast direction. A third place that has produced extensively is in the neighborhood of Galena, Kansas. I shall attempt a general description of these orebodies by using as an example the great Webb City zone.

This productive area extends from Oronogo on the northwest to Porto Rico and Duenweg on the southeast, a distance of 10 miles. For this distance the average width of the zone is perhaps three-quarters of a mile, though it widens at one or two places to a mile and a half and narrows at other places to a quarter of a mile. In a rough way, I estimate the productive ground at 4800 acres. It would not be inaccurate to describe this entire tract as a continuous orebody, although it shows great irregularities. The total production of this zone has been approximately 3,000,000 tons of zinc and lead ore, derived from mining and milling 75,000,000 tons of rock. The value actually realized has been about $90,000,000.

The rocks in which the ore occurs constitute a flat-lying formation of chert and limestone about 250 ft. thick. At the bottom of the formation is a persistent bed of flint about 20 ft. thick, called the Grand Falls chert. Above this is limestone containing many layers and nodules of flint. The Grand Falls chert is a brittle stratum of flint containing innumerable crevices so that it serves as a ready channel for the circulation of water. On this account much ore has been deposited in it, forming the so-called "sheet ground," which occurs as a regular flat bed, like a seam of coal. Laterally its extent is variable, as also is its richness, but the mineralization is rather uniform over extensive areas, often as much as 2000 ft. wide. Practically all of the successful sheet-ground mining to date has been confined to the great Webb City ore-channel, between Oronogo and Porto Rico. It is generally believed that the sheet ground yields about 3 per cent. of the rock mined in zinc or lead ore. The zinc ore obtained averages not far from 60 per cent. zinc; the lead ore about 80 per cent. lead. The ore is obtained by concentrating mills, which save about 60 per cent. of the zinc and 90 per cent. of the lead actually contained in the rock.

Owing to the shallowness of the deposits there is no occasion for large expensive shafts. As the extreme depth is only 250 ft., and the average depth in mining perhaps less than 175 ft., it is evident that a single-compartment shaft, except in the unusual contingency of encountering a

very large amount of water can be sunk very cheaply. It is probable that the average shaft of the Joplin district does not cost more than $4000. Hence it is cheaper to open up the ground by numerous shafts rather than by extensive openings underground. It will also be evident that aside from the question of first cost the tramming of ore is cheaper on the surface than underground. The effect of these considerations is that the accepted method of operating in the district is to have one mill supplied with ore from several shafts, the ore being transported to the mill by inclined tramways. The mining underground involves the usual requirements of selecting the ground so as to mine out the best of the ore without leaving too much in the pillars and without making the openings too dangerous.

The external factors which affect mining in the Cœur d'Alene are the most favorable of the whole Rocky mountain region. The altitude is moderate; the climate, mild; timber and water power are abundant and cheap. Transportation to consuming centers is, however, expensive, and wages are high. Labor is efficient and abundant. The mines are generally deep, measured from the surface, but the configuration of the country has permitted their attack by adit levels; so that most of the ore has not needed hoisting from great depths, and pumping operations have generally been inexpensive.

The internal factors are favorable. The veins are typical fissures. The ore is galena, which seems to be a metasomatic replacement of preexisting veins of iron carbonate. Ransome believes that the Burke quartzite, a formation of flaggy, evenly bedded, light-colored rock about 1700 ft. thick, contains nearly all the payable ore, although veins are found traversing an immense mass of slates and quartzites of presumable Algonkian age, some over and some under the Burke quartzite. The whole sedimentary series is estimated to have a thickness of 13,000 feet.

The ore shoots are persistent and profound, with a thickness varying from 8 to 100 ft., and a length varying from 100 to 1000 ft. normal to the plunging axis. Single bodies have produced several million tons of ore. The ore in the main has to be concentrated. The proportion shipped to the smelters varies from a quarter to a tenth of the amount mined. Of the proportion shipped a considerable amount is picked out by hand either underground or at the mill, the lower grades being concentrated. In addition to the sorting of first-class ore, there is still larger sorting of waste in the stopes. In many cases it is necessary for safety to fill the stopes, and in all cases it is economical to reject waste. The various mines differ greatly in the amount of sorting and filling done. Several of the mines have run for years without shipping any first-class ore and without sorting any waste in the stopes, everything

mined being sent to the concentrator. On the other hand, one prominent mine, the Hercules, ran several years without a mill, shipping only first-class ore.

The mines may conveniently be divided into two groups: the Wardner and the Cañon creek. In Wardner there is only one vein and two important mines; the Bunker Hill & Sullivan, an independent concern, and the Last Chance, owned by the Federal Mining and Smelting Company.

The mining is done almost wholly by the filling method. Whether square sets are put in first and then filled, or whether the stopes are filled without timbering depends on the firmness of the ground. This varies in different parts of the mines. In nearly all cases enough waste for the filling can be sorted out of the vein-stuff itself.

The Bunker Hill mine in 20 years up to June 1, 1907, had produced 3,388,106 tons of ore. The ore shipped in 1906-7 was somewhat above the average in grade, but it will serve as an illustration of the general problem of mining on the Wardner vein. Out of 336,630 tons mined, 87,640 tons were shipped to the smelters, or one ton in 3.84. The shipping product averaged 45.83 per cent. lead and 18.78 oz. silver per ton. The ore as mined assayed 13.32 per cent. lead and 5.89 oz. silver, the milling loss being estimated at 10.43 per cent. lead and 17.06 per cent. of the silver, or 11.96 per cent. of the combined product. The Cañon creek mines differ from the Wardner mines only in the shape of the orebodies. The dip is not far from vertical; the ore-shoots are much longer, thinner and more regular. Wages average 46c. per hour, 4c. higher than in Wardner.

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LITHIA.

The lithium minerals of commercial importance are spodumene, lepidolite and amblygonite. Mining is carried on only in California and South Dakota, chiefly the latter. In 1908 there was but little production in California. The Etta mine at Keystone, S. D., was operated. Its product is shipped to the Standard Essence Company, Maywood, N. J.

STATISTICS OF LITHIUM ORE AND SALTS IN THE UNITED STATES. (a)
(In tons of 2000 lb.)

Year.	Production. (b)		Imports. (c)		Year.	Production. (b)		Imports. (c)	
	Tons.	Value.	Pounds.	Value.		Tons.	Value.	Pounds.	Value.
1902.....	1,245	$25,750	21,216	$22,951	1906....	383	$7,411	*Nil.*	.....
1903.....	1,115	23,425	5,596	3,669	1907....	530	11,000	60	$100
1904.....	577	5,155	19	48	1908....	.....	.....	.....	.....
1905.....	79	1,412	*Nil.*	.....					

(a) Statistics of the U. S. Geological Survey. (b) Ore. (c) Lithia salts.

An interesting process for the manufacture of lithia from lepidolite was described by W. J. Schieffelin and Thomas W. Cappon in *Journ. Soc. Chem. Ind.*, June 15, 1908, p. 549. The cost of making lithia by this process was about 90c. per lb. and the process was introduced when the price of lithia was $3.50 per lb. The entry of new concerns into the manufacturing business precipitated a decline in the price which culminated toward the end of 1907 when 40c. per pound was quoted. That price prevailed during 1908.

MAGNESITE.

The only important deposits of magnesite in the United States are in California, mainly in the Coast Range. But few of them are worked. The consumption on the Pacific Coast is small and the high cost of transportation to the Atlantic seaboard prevents competition with the mineral imported there from Austria and Greece. Even to supply the small consumption of the Pacific Coast only those deposits that are close to railways can be profitably worked. The annual production of California is 7000 to 9000 tons worth from $3 to 3.50 at the mines. The bulk of the production is made by the Willamette Paper and Pulp Company, at Porterville, Tulare county. The crude mineral is hauled by traction engines to the kilns where it is calcined, the burned product being shipped to paper-makers. To make one ton of calcined mineral requires 2.4 tons of crude. The value of the calcined varies from $12 to $20 per ton.

STATISTICS OF MAGNESITE IN THE UNITED STATES.
(Tons of 2000 lb.)

Year.	Production. (a)		Imports.		Consumption.	
	Short Tons.	Value.	Short Tons.	Value.	Short Tons.	Value.
1897	1,143	$13,671	(b)	.....	.....	.....
1898	1,263	19,075	16,039	$134,130	17,302	$153,205
1899	1,280	18,480	20,807	(c) 174,779	22,087	193,259
1900	2,252	19,333	28,821	(c) 216,158	31,073	235,491
1901	4,726	43,057	33,461	(c) 250,958	38,187	294,015
1902	2,830	20,655	49,786	373,928	52,616	394,583
1903	1,361	20,515	54,776	461,399	56,137	481,914
1904	2,850	9,298	33,704	286,828	41,554	296,126
1905	3,933	16,221	74,374	638,619	78,307	654,840
1906	4,032	40,320	90,396	863,492	94,428	903,812
1907	6,405	57,720	99,008	875,359	105,413	933,079
1908	8,967	52,342	84,494	736,763	93,461	789,105

(a) Reported by the State Mining Bureau of California. (b) Not reported. (c) Estimated.

The Willamette Paper and Pulp Company in 1908 opened a deposit at South Tule, Tulare county. A deposit near Winchester, Riverside county, was opened by the California Magnesite Company. This deposit furnished some material which was utilized in Los Angeles for the manufacture of flooring, wainscoting and other building material. A small amount of magnesite was mined from the deposit at Cloverdale, Sonoma county, and at Red Mountain, Alameda county. A mine in Fresno

county was developed by the Fresno Magnesite Company. The American Magnesite Company, works at Oakland, Cal., produced magnesite brick and carbon dioxide in 1908. The plant was shaken down in 1906, but has since been rebuilt.

Market.—According to F. L. Hess, of the U. S. Geological Survey, none of the California veins compares well in size with the reported width of the Hungarian veins. In quality, however, the comparison with the foreign material is favorable; in fact, the California article is ordinarily better. Magnesite from Portersville now costs about $6.50 per 2000 lb. laid down at San Francisco; probably that from the Gilliam creek (Sonoma county) deposits can be delivered for somewhat less. Production at the Kings River deposits will cost about the same as at Portersville. Magnesite from Sonoma and Napa counties can probably be calcined and delivered in San Francisco at $15 per 2000 lb. Imported magnesite was quoted in New York City in April, 1908, at $7.25@8 per long ton, equal to $6.38@7.08 per short ton; calcined magnesite was quoted at $16.75@25, and when comparatively free from lime, ground, sells in small lots at the latter price. With this difference in price for calcined magnesite of $5@6 between San Francisco and New York, it seems possible that this product could sometimes be shipped at a profit to the eastern coast of the United States on vessels that would otherwise sail without a full cargo and would for this reason be willing to carry the material at low rates.

During 1908 the price of Grecian magnesite ex ship at New York was $9 per 2240 lb. According to a communication from the Willamette Paper and Pulp Company, of Porterville, Cal., its production from December, 1904, to May 9, 1908, was 14,241 tons of raw magnesite, which yielded 5888 tons of calcined. The cost of production was as follows: Raw material, $4.93 per ton; calcined, $15.99 per ton, f.o.b. cars at Porterville, or $18.99 per ton delivered at San Francisco. The capacity of the calcining plant of this company is 6000 to 7500 tons per annum. The capacity of other producers in California is probably less than 500 tons of calcined product in the aggregate. In November, 1908, foreign magnesite was sold in New York at the following prices: Crude, $10 @12 per 2000 lb.[1]; calcined, unground, $14@16.50, according to quantity; calcined and ground, $26.50.

Uses.—The bulk of the magnesite produced in California is calcined for sale to paper-makers. A comparatively small proportion is sold crude to the manufacturers of carbon dioxide gas. More or less calcined magnesite is used for the manufacture of building material, there being several factories in California engaged in that business. The

[1] This figure is higher than the market reports of *Eng. and Min. Journ.*

manufacturers of carbon dioxide gas dispose of their magnesia residue to the paper mills.

Magnesite should be high in magnesium carbonate. The commercial mineral frequently contains as much as 98 per cent. In burning a charge of 90 per cent. of high-grade magnesite mixed with 10 per cent. coke, 50 per cent. of the weight of the magnesite is obtained in the form of carbon dioxide. The price of liquid dioxide depends upon the use to which it is to be put, the maximum price being about 10c. per lb. However, the price is very variable and must be determined by negotiation. About 48 per cent. of the weight of the magnesite remains as residue in the kiln, plus the weight of the ash of the coal or coke used. Coal or coke may be used as fuel. In the case of coke, the requirement is about 10 to 12 per cent. of the weight of the magnesite. The value of calcined, unground, magnesite is $14@15 per 2000 pounds.

There is a large consumption of calcined magnesite in connection with the manufacture of open-hearth steel. The requirements of magnesite for this purpose are a high percentage of magnesia, absence of any large quantity of lime, low content of alumina, a moderate content of silica, and a proper content of ferric oxide, the amount of which

THE PRINCIPAL SUPPLIES OF MAGNESITE.
(In metric tons.)

Year.	Austria-Hungary. (a)	Germany. (c)	Greece. (d)	India. (d)	United States. (d)
1897.....	(b)	53,086	11,311	(e)	1,038
1898.....	(b)	50,114	14,829	(e)	1,146
1899.....	(b)	60,910	17,184	(e)	1,161
1900.....	(b)	67,988	17,277	(e)	2,043
1901.....	40,236	67,732	20,348	(e)	4,286
1902.....	53,467	58,947	23,020	3,597	2,567
1903.....	69,058	60,834	28,415	838	1,234
1904.....	53,781	65,142	9,133	1,193	2,585
1905.....	92,359	87,585	37,063	2,645	3,568
1906.....	87,765	81,481	40,584	1,861	3,658
1907.....	113,695	73,996	55,816	188	5,809

(a) Exports of calcined magnesite. (b) Previous to 1901 magnesite was included with other minerals not elsewhere specified. (c) Chloride and sulphate of magnesium. (d) Crude magnesite. (e) Not reported.

should be neither too high nor too low. A calcined product containing 89 to 90 per cent. of magnesia, 3 per cent. of silica, 0.5 per cent. of alumina, 1 per cent. of lime, and 5 per cent. of ferric oxide is said to conform to the requirements of the open-hearth steel industry.

The manufacturers of magnesite brick in the United States are the Harbison-Walker Refractories Company, of Pittsburgh, Penn., and the American Refractories Company, of Chicago, the latter being a new concern. The Harbison-Walker company obtains its supply of raw material from the Veitscher Magnesitwerke Aktiengesellschaft, of Aus-

tria. The American Refractories Company has secured control of a large deposit in Styria. This company has already begun the manufacture of magnesite and chrome bricks at its plant at Joliet, Ill.

Magnesite in Foreign Countries.

Austria.—A large deposit of magnesite in the Millstatt Alps, near Radenthein, in Carinthia, about 128 miles from Trieste, has been secured by the Austro-American Magnesite Company, in which the American Refractories Company, of Chicago, Ill., is interested. This company was engaged in 1908 in the installation of a plant capable of producing 40,000 tons of calcined magnesite per annum. The mineral will be brought down to the plant by an aërial railway. The calcining kilns are to be fired by producer gas. The plant is expected to be in operation by the middle of 1909. The deposit is said to contain many millions of tons of magnesite, averaging 89.12 per cent. magnesia, 2.95 per cent. silica, 0.47 per cent. alumina, 1.8 per cent. lime, 5.20 per cent. ferric oxide, and 0.81 per cent. volatile matter.

According to *Montan Zeitung,* Apr. 15, 1909, a large deposit of crystalline magnesite of excellent quality has been discovered at St. Martin, near Oblaru, Upper Styria. It is said to be favorably situated for exploitation.

Greece. (By F. Meijnan.)—There are at present three companies operating magnesite quarries on the island of Euboea. One small company is working a deposit at Hermione on the island of Nauplia. This company exports only crude material and has not erected kilns. The Euboea deposits are worked (1) by the Anglo-Greek Company, Ltd., with officers in London, England, the quarries being situated near Limny, on the west coast of the island; (2) by the Société de Travaux Publics et Communaux, Athens, with quarries near Manthoudi on the east coast; (3) by the Société Hellénique de Mines de Magnésite, with offices at Athens, Greece, and Amsterdam, Holland, the quarries being near Pily, on the east coast.

The Anglo-Greek company mines by open-cut and transports its output, consisting of about 100 tons per day, by a 75-km. aërial tramway, to the port of St. John, near Limny. The product is exported partly as crude mineral and partly as calcined magnesite. The calcining is done in modern bituminous-fired kilns at the company's works in St. John.

The deposits of the Société de Travaux are opened by galleries about 650 m. long. The quarries are connected with the port of Kymassi, near Manthoudi, by a railroad about 2 km. long, over which the daily

output of 75 tons is shipped. This product is partly calcined in a modern lignite-fired kiln which works automatically with continuous fire.

The Société Hellénique works its quarries by galleries, and ships the product to Pily by an overhead tramway about 3 km. long. The production amounts to 60 tons per day. Calcining is done in small, open kilns, but an automatic kiln is being built.

An average analysis of the magnesite from the three quarries gives $MgCO_3$, 95 per cent.; CaO, 1 to 2 per cent.; SiO_2, 0.5 to 2 per cent.; and some combined water. A complete analysis of crude mineral from the mines of the Société Hellénique is as follows: MgO, 46.28 per cent.; CaO, 0.41; FeO, 0.03; Al_2O_3, 0.07; SiO_2, 2.27; CO_2, 49.85; H_2SO_4, 0.02; P_2O_5, trace; combined H_2O, 1.15.

The price for crude magnesite for 1908-9 is 18 to 20 fr. ($3.47@3.86) per ton in bulk, and for calcines, 72@75 fr. ($13.90@14.48) in bags, both f.o.b. steamers at Grecian ports.

New South Wales.—A large deposit of high-grade magnesite is said to have been discovered near Fifield.

Portuguese East Africa.—Large deposits of magnesite, equal to the best Grecian magnesite, have been opened near Malelane and Kaapmuiden, 100 miles from Lourenco Marquez, close to the railway line. They are operated by a company called the Magnesite Mines of South Africa.

MAGNESIUM.

This metal was made in the United States in 1908 by the Virginia Electrolytic Company, at Holcomb's Rock, Va. This company employs a process of its own, which it considers to be very superior, and expects to become an important factor in the market for this metal in competition with the importers. The price for the metal in 1908 was $1 per lb. The domestic production was 500 lb. The process employed at Holcomb's Rock is described in U. S. patent, 900,961, Oct. 13, 1908 (G. O. Seward and F. von Kügelgen). A mixture of magnesium fluoride with the chloride of calcium or some other alkaline-earth metal more electro-positive than magnesium is heated in order to render it anhydrous, a small percentage of an alkali fluoride being added as a flux. After the magnesium has been separated electrolytically, hydrated magnesium chloride is added and the whole subjected to a further dehydration, the resulting mixture of magnesium fluoride and calcium chloride being utilized in a second electrolysis.

Use as Deoxidizer.—Magnesium is now being employed to some extent as a deoxidizer in brass manufacture, having the advantage over phosphorus that an excess may be used without harm, and indeed may improve the quality of the brass. (Magnesium is a metal which belongs to the same family as zinc.) Ordinarily the addition of 0.05 per cent. of magnesium to the brass is sufficient for deoxidizing purposes.

Improved Method of Melting Magnesium.—Owing to the fact that the melting point of magnesium is low and it oxidizes rapidly when melted, the loss on melting, when carried out in the usual manner, is large. Paul Rakowicz, of the Chemische Fabrik Griesheim Electrom of Frankfurt am Main, Germany, has patented a method of melting magnesium which it is said, minimizes the loss. The magnesium is placed in a graphite or clay crucible and allowed to arrive at a temperature beyond that of a pasty mass. If it goes beyond, more magnesium is added. When the whole of the magnesium in the crucible has become pasty the crucible is removed and the magnesium allowed to melt by the heat transmitted from the walls of the crucible itself. In this manner overheating is avoided. After it has been so melted the other metals with which it is to be alloyed are added.

Magnalium.—This is an alloy of aluminum and magnesium manufactured by a syndicate in Germany. It contains from 90 to 98 per cent. aluminum, and possesses properties which render it suitable for many industrial purposes. It is imported in the form of pigs or ingots for castings or forgings and its manipulation is said to be very simple. The chief advantages claimed for magnalium are that it can be forged, soldered by men without special training; it can be rolled and annealed and that in all forms its strength is greater than that of pure aluminum, although its weight is less. The specific gravity is about 2.5, varying slightly with the composition, while that of pure aluminum is 2.64. The tensile strength of magnalium has been found to be as follows in pounds per square inch: Castings, 19,000 to 42,000; forgings, 30,000; annealed and hard rolled plates, 42,000 and 52,000 respectively; drawn rods, 60,000; wire, 41,000 to 53,000; and tubing, 74,000. The melting point varies from 1185 to 1250 deg. F., and the specific heat is 0.2185.

The properties of the alloy vary somewhat with the composition. A small proportion of magnesium makes a good alloy for casting, but for forging, rolling or wire-drawing a larger admixture of magnesium is required. Tubing is made of material containing about 90 per cent. aluminum, while rolled plates contain about 94 per cent.

The electric and thermal conductivity of the alloy is 56 per cent. of that of pure copper. The material takes a high polish and resists oxidation better than aluminum. It is unaffected by dry or damp air, water, gaseous ammonia, carbonic acid, hydrogen sulphide and most organic acids. It is very slightly affected by nitric and sulphuric acid and more rapidly by alkalies or strongly alkaline solutions. It is slightly attacked by salt water, and surfaces exposed to sea water should be protected by a coat of lacquer. A multitude of articles made of magnalium are found in the European markets, varying from parts of delicate balances and optical instruments to agricultural implements, horseshoes and engine castings.

The new alloy is sold at prices only slightly higher than that of aluminum, considering its greater tensile strength and the smaller quantity required for a given purpose. Quoted prices for magnalium f.o.b. New York are as follows: Crude metal, $47 to $61 per 100 lb. according to quantity; wire, $85 to $90 per 100 lb. with extra charges for very small diameters; plates, $80 to $85 per 100 lb. with extra charge for plates less than one millimeter thick; rods, $80.70 to $85.70 per 100 lb.; tubes, $160 to $180 according to size; and solder $1 per lb. A material reduction in price is made when the alloy is ordered in large quantities.

MANGANESE.

The manganese mining industry in the United States is insignificant. In 1907 the production was only 5604 long tons, chiefly from Virginia. In 1908 it was a little larger. Practically the whole output in 1908 came from Virginia. The chief reason for the insignificance of this industry in the United States is the trifling character of the deposits. This refers to manganese ore, properly speaking. There is in the United

PRODUCTION OF MANGANESE ORES IN THE UNITED STATES (a).
(Tons of 2240 lb.)

Year.	Manganese Ores.				Manganiferous Iron Ores.				Man. Zinc Ores.	Total Production.	
	California.	Georgia.	Virginia.	Other States.	Arkansas.	Colorado.	Lake Superior.	Va. & N. C.	New Jersey.	Long Tons.	Value.
1896..	318	2,538	1,588	*Nil.*	3,038	9,072	110,317	.....	35,655	162,526	$339,083
1897..	450	962	2,408	190	4,430	18,600	80,260	.....	50,000	(b)158,600	328,176
1898..	393	2,477	3,307	1,250	2,775	17,792	112,318	.....	47,470	187,782	416,627
1899..	263	1,623	3,626	105	855	29,161	53,702	.....	53,921	143,256	306,476
1900..	131	3,447	7,881	312	*Nil.*	43,393	75,360	*Nil.*	87,110	217,546	1,172,447
1901..	610	4,074	4,275	3,036	*Nil.*	62,385	512,084	20	52,311	638,795	1,644,117
1902..	846	3,500	3,041	90	*Nil.*	13,275	884,939	3,000	65,246	973,937	2,145,783
1903..	16	500	1,801	508	*Nil.*	14,856	566,835	2,802	73,264	660,582	1,670,349
1904..	60	*Nil.*	3,054	32	600	17,074	365,572	*Nil.*	68,189	454,581	789,132
1905..	1	150	3,947	(e)20	3,321	45,837	720,090	*Nil.*	90,289	863,663	1,681,472
1906..	1	.....	6,028	892	8,900	32,400	1,000,008	*Nil.*	93,461	1,141,681	(e)3,403,993
1907 .	100	*Nil.*	(d) 4,604	900	4,133	(d) 99,711	(e)1,120,000	*Nil.*	93,413	1,322,861	(e) 3,860,265
1908..	*Nil.*	*Nil.*	(c)6,144	200	*Nil.*	(d) 35,581	(e) 495,000	*Nil.*	110,225	647,150	(e)1,294,300

(a) Statistics of 1900-1906 are by the U. S. Geological Survey. (b) Includes 1300 tons of manganiferous iron ore from Vermont. (c) As reported by Virginia Geological Survey. (d) As reported by U. S. Geological Survey. (e) Estimated

CONSUMPTION OF MANGANESE ORE IN THE UNITED STATES.
(Tons of 2240 lb.)

Year.	Imports.		Consumption.		Production of Man. Silver Ores. (b)	
	Long Tons.	Value.	Long Tons.	Value.	Long Tons.	Value.
1897	119,961	$1,023,824	278,561	$1,352,000	149,562	$424,151
1898	114,885	831,967	302,667	1,248,594	99,651	295,412
1899	188,349	1,584,528	331,605	1,891,004	79,855	266,343
1900	256,252	2,042,361	473,798	3,214,308	188,509	897,068
1901	165,722	1,486,573	804,568	3,130,990	228,187	965,959
1902	235,576	1,931,282	1,209,513	4,077,065	174,132	908,098
1903	146,056	1,278,108	806,638	2,948,457	179,205	649,727
1904	108,519	901,592	563,100	1,690,724	105,278	348,132
1905	257,033	1,952,407	1,120,696	3,633,379	127,170	445,095
1906	221,260	1,696,043	1,362,941	5,100,036	163,760	573,160
1907	209,021	1,793,143	1,531,882	5,653,408	103,844	259,473
1908	178,203	1,350,223	825,353	2,644,523	51,554	123,407

(b) Mined in Colorado and used as flux in silver-lead smelting; not included in the statistics of consumption. The statistics of manganiferous silver ore for 1907-08 are as reported by the U. S. Geological Survey.

States a large production of manganiferous iron ore and manganiferous zinc ore.

UNITED STATES PRODUCTION AND IMPORTS OF IRON-MANGANESE ALLOYS.
(In tons of 2240 lb.)

	1905		1906		1907		1908	
	Production.	Imports.	Production.	Imports.	Production.	Imports.	Production.	Imports.
Ferromanganese	66,179	52,841	55,520	84,359	55,918	87,400	40,642	44,624
Spiegeleisen.....	227,797	55,457	244,980	103,267	283,430	48,995	111,376	4,579
Totals.....	293,976	108,298	300,500	187,626	339,348	136,395	152,018	49,203

Arkansas.—A small production was made in the Batesville district in 1908.

California.—Large deposits of manganese ore are claimed to exist near Hemet, Riverside county, but the mines cannot be worked owing to their remoteness from market and the high cost of transportation thereto.

Georgia. (By S. W. McCallie).—There was little or no manganese produced in Georgia in 1908. This condition is not by any means due to the exhaustion of the mines themselves, but chiefly to the financial condition of the main producer, viz., the Southern Steel Company, which lately has been in the hands of a receiver and has made no effort to put any manganese ore on the market.

Tennessee.—The manganese ores of this State occur near the eastern border of the Appalachian Valley deposits. The only deposits now known occur in Cocke county, near Newport and Del Rio, and near Shady Valley, in Johnson county. The ore is both soft and hard, the soft ore occurring in pockets and seams associated with clay, and the hard ore in irregular masses scattered through the clay and the soft ore pockets.

Wisconsin. (By W. O. Hotchkiss.)—Wisconsin produces no manganese ore, but two of the mines on the Gogebic range produce an iron ore carrying a low percentage of manganese. One mine in 1908 produced 60,365 tons of ore carrying from 2 to 5 per cent. manganese, with an average of about 3 per cent. The other mine shipped 33,000 tons of ore bearing 2.86 per cent. of manganese. In the Baraboo district several drill holes show beds of manganiferous ore containing from 2 to 25 per cent. manganese, and from the indications given by the drill considerable quantities are available. The manganese bearing ores seem to run in lenses similar to the iron ores and are associated with them.

WORLD'S PRODUCTION OF MANGANESE ORE. (*a*)

(In metric tons.)

Year.	Australia.	Austria-Hungary.	Belgium.	Bosnia (*b*)	Brazil. (*d*)	Canada.	Chile. (*d*)	Colombia.	Cuba.	France.	Germany.	Greece	India.
1897.	.....	10,043	28,372	5,344	16,054	14	23,528	8,382	.. ....	37,212	46,427	11,868	74,862
1898.	.....	14,219	16,440	5,320	26,417	45	20,851	11,176	.....	31,935	43,354	14,097	61,469
1899.	.....	10,484	.....	5,270	65,000	279	40,931	10,160	.. ....	39,897	61,329	17,600	88,520
1900.	.....	14,550	10,820	7,939	108,244	34	25,715	8,748	..,973	28,992	59,204	8,050	129,865
1901.	368	12,077	8,510	6,346	100,414	447	18,480	95	25,586	22,304	56,691	14,166	122,831
1902.	4,692	12 883	14,440	5,760	157,295	175	12,990	*Nil.*	40,048	12,536	49,812	14,960	160,311
1903.	1,415	11,489	6,100	4,537	161,926	135	17,110	(*c*)	21,070	1,583	47,994	9,340	174,563
1904.	843	15,460	485	1,114	206,260	123	2,324	(*c*)	33,152	11,254	52,886	8,549	152,708
1905.	1,540	23,732	*Nil.*	4,129	224,377	22	1,323	(*c*)	(*d*) 8,096	6,751	51,463	8,171	257,969
1906.	1,131	20,577	120	7,651	201,500	84	35	(*c*)	(*d*)13,997	11,189	52,485	(*d*) 9,200	503,686
1907.	1,134	22,463	2,100	3,500	(*c*)	1	(*c*)	(*c*)	(*c*)	18,200	74,683	10,000	907,717
1908.	(*c*)	(*c*)	(*c*)	(*c*)	(*c*)	*Nil.*	(*c*)	(*c*)	(*c*)	(*c*)	67,693	(*c*)	(*c*)

Year.	Italy.	Japan.	New Zealand.	Portugal.	Queensland.	Russia.	Spain.	Sweden.	United Kingdom	United States. (*e*)
1897.....	1,634	15,448	182	1,652	403	263,115	100,566	2,749	609	161,138
1898.....	3,002	11,497	220	907	68	329,276	102,228	2,358	235	190,787
1899.....	4,356	11,336	137	2,049	747	659,302	104,974	2,622	422	145,548
1900.....	6,014	15,831	166	1,971	77	802,236	112,897	2,651	1,384	221,714
1901.....	2,181	16,270	208	904	221	522,395	60,325	2,271	1,873	649,016
1902.....	2,477	10,844	*Nil.*	(*c*)	4,674	536,519	46,069	2,850	1,299	989,519
1903.....	1,930	5,616	71	30	1,341	414,334	26,194	2,244	831	671,151
1904.....	2,836	4,324	199	(*c*)	843	430,090	18,732	2,297	8,880	461,854
1905.....	5,384	14,017	55	(*c*)	1,541	508,635	26,020	1,992	14,582	877,482
1906.....	3,060	54,339	16	22	1,131	1,015,686	62,822	2,680	23,126	1,159,948
1907.....	3,654	10,410	*Nil.*	(*c*)	1,134	927,917	41,504	4,300	16,356	1,322,861
1908.....	(*c*)	(*c*)	*Nil.*	(*c*)	1,395	(*c*)	(*c*)	(*c*)	(*c*)	647,150

(*a*) From official statistics. (*b*) Includes Hersegovina. (*c*) Statistics not available. (*d*) Export returns. (*e*) Includes output of manganiferous iron ore.

Brazil.—This comparatively new industry in Brazil developed considerably during 1908. There are two distinct classes of deposit—those in which the mineral is associated with iron ore and limestone, and those where it occurs in gneiss. To the former category belong the deposits in Miguel Burnier and Ouro Preto (Minas Geraes), and in Corumba (Matto Grosso). The Miguel Burnier ore is worked partly by open cast, partly by means of shafts, and contains about 55 per cent. of manganese, comparatively free from phosphorus. Many attempts have been made to roast the ore on the spot or briquet it, in order to save freight, but failed, owing to the lack of a suitable binding medium. Manganese ore in gneiss is found at Lafayette-Queluz (Minas Geraes) and Nazareth (Bahia), the ore from the latter place containing about 47 per cent. Mn, 7.2 per cent. Si, and 0.38 per cent. P. Extensive deposits have also been found in Matto Grosso, near the Bolivian frontier; and the concessions belonging to a Brazilian mining company have recently been transferred to a Belgian company with works at Urucum. In spite of the increased output of the Brazilian mines and the favorable rates of freight accorded by the railway for conveying the ore to Rio, the industry has suffered considerably

from Russian competition and the financial conditions in the United States, the industry being almost entirely dependent on the export trade.

Some of the most important manganese mines in the Republic are near Queluz, in the State of Minas Geraes. One of the largest of these is Morro da Mina, situated at an elevation of 1100 m. above the level of the sea, and 6 km. from the station of Lafayette, on the Central Railway, with which it is connected by a branch line constructed by the mining company. The mineral-bearing zone of the property covers about five hectares, and for a long time attracted but little attention, the ore having been reported by the first explorers as of inferior quality. Several years ago, however, the deposit was carefully studied and mining operations were commenced on a large scale. The outcrop of ore can be traced over a distance of nearly 1000 m. The country rock is a deeply altered eruptive schistose, and the deposit has a lenticular form. All the workings are open-cut terraces running lengthwise along the east side of the hill, and topographic conditions are such that it is possible to carry on the work simultaneously along a face 750 m. in length. This enables mining operations to be carried on rapidly, and 600 tons of ore have been taken out in 10 hours, and the output could be greatly augmented by increasing the number of laborers. The workings have a total length of more than 3000 meters. Borings have shown that the ore deposit extends more than 120 m. below the outcrop, and in 1904 the company estimated its available ore body at 5,000,000 tons. Analyses of carload lots of the ore give an average of 50 per cent. of manganese, 1 to 2 per cent. of silica, and 0.06 to 0.08 per cent. of phosphorus. After the ore is sorted it is conveyed in the mine cars of the company to the ore bins, and is then automatically loaded, through chutes, to the freight cars of the Central Railway. The total quantity of ore mined up to the end of 1907 was 260,198 metric tons; the output for 1908 greatly exceeded that of previous years. The output depends largely upon the number of cars the Central Railway can furnish for transporting the product.

Canada.—Manganese mining in New Brunswick, which was once of considerable importance, has of late become inactive, chiefly because the largest deposits are remote from shipping facilities and the cost of transportation from them is too high.

China. (By T. T. Read.)—The only producer of manganese ore in China is the Han-Yeh-Ping Iron and Coal Company, which mines sufficient amounts for use at its steel plant. The ore is mined at Ching-K'ou-chou, just south of the Yangtze, about 100 miles toward its mouth

from Hankow. The company also owns deposits along the Han river, but does not work them yet. The production for 1908 is stated by the company as 3600 tons.

India.—The depression in the steel industry throughout the world affected adversely the manganese mining industry of India. In 1907 the price for ore containing 50 per cent. manganese was 1s. 4d. per unit, c.i.f.; for 40 to 47 per cent., 1s.; for 47 to 50 per cent., 1s. 3d.; in 1908 the prices fell to 10d., 9d. and 8d. respectively. At these prices the Indian ore could not compete with foreign ore, and only those producers who had made contracts at higher prices were able to operate successfully. The decline was felt especially in Mysore and Goa, where many companies ceased operation. In the Bellary district the depression was felt by all of the companies except the one working in the Sandur State. The General Sandur Mining Company, however, instead of restricting its operations, extended its work on a large scale. The branch railway to Ramanmalai was extended to Kanevihalli, which connects the extensive deposits of the Ramandrug and the Kanevihalli ranges. Manganese mining in the Vizagapatam district was not affected so severely as in the Mysore district, owing to its proximity to the ports.

The premier manganese mines of India are undoubtedly those of the Central Provinces, although, area for area, the deposits of Sandur are superior in respect to ore reserves. The present position of India as a leading producer of manganese ore is mainly due to the mines of the Central Provinces. Compared with these, the south Indian mines are in their infant stage. The leading exporters are the Central Provinces Prospecting Syndicate and the Central India Mining Company, Limited, both of which own some of the finest deposits in India. Although taxed severely by the heavy railway freight to Bombay, the ore of the Central Provinces can be worked profitably because of its high manganese content. The ore is so rich that it is exported in large quantities even to the United States.

The deposit at Kodegaon in the Nagpur district, together with several good deposits in the Chindwara district, was bought a few years ago by the Indian Manganese Company. More recently the deposits near Kumsi were purchased by the Workington Iron and Steel Company. The General Sandur Mining Company, operating in the Sandur State, is the most highly capitalized manganese mining company in the world. In the Balaghat district of the Central Provinces the rich deposit at Ukua-Samnapur was purchased by the Carnegie Steel Company. This, together with the deposit at Ramrama belonging to the Tata Iron and Steel Company, are being developed. The Central India

Mining Company owns several good deposits in the Nagpur and Bhandara districts, which are connected by its own railways with the Bengal Nagpur railway.

PRODUCTION OF MANGANESE ORE IN INDIA.
(In metric tons)

District.	1903	1904	1905	1906	1907
Bombay.....	.....	*Nil.*	.....	7,638	35,743
Central India.					
Uhabua State.....	6,909	11,749	30,736	50,877	686,572
Central Provinces.					
Balaghát district.....	8,024	10,489	162,517	325,907	24,140
Bhandára district.....	.....	8,695			
Nagpur district.....	95,159	67,214			
Madras.					
Visagapatam district..	64,470	54,260	64,717	119,264	151,890
Total.....	•174,562	152,707	257,970	503,686	898,345

According to R. O. Ahlers, the export of Indian manganese commenced in 1893, when 15d. per unit was paid for ore of 50 per cent. and over, but the price gradually declined till 1898, when it stood at 10½d. It then gradually rose till 1900, when again 15d. per unit was paid; another gradual fall succeeded till the beginning of 1905, when sales at 9d. were difficult to negotiate, and many of the mines were working at a loss. Then came the sudden spurt, commencing in April, 1905. Before the end of 1906, good prices (16½d.) were quoted, and during the first half of 1907 sales were negotiated at as high a figure as 18d. A sudden drop followed and in the latter part of 1908 prices stood at 9½d., at which level they have been fairly constant for six months.

The increased demand for manganese and the high prices this metal commanded, stimulated prospecting all over the world, and nowhere with better results than in India. Manganese mining commenced in India in 1892, at Vizianagram in the Madras presidency, and 3130 tons were produced in the first year. Till 1900 this was the only manganese-producing district in India. In that year the Central Provinces commenced with a production of 35,356 tons for the first 12 months. In 1903 the deposits of Central India were opened, and these three localities produced in 1905 the total of 247,472 tons of ore, a figure increased to 495,730 tons in 1906.

As a result of the prospecting operations, there was added to the manganiferous areas of India a new district, situated in the southwest and embracing the native State of Mysore, the northwestern part of the Madras presidency, the southwestern part of the Bombay presidency, and the Portuguese territory of Goa. There was soon a considerable demand for manganese properties in this area, and by 1907 more

than a thousand square miles had been taken up for manganese mining in Mysore territory alone.

This new southern area assumed prominence only in 1906, when 9101 tons of ore were shipped from Mormugao harbor, and so rapid was the rise in production, that in 1907 the shipments from Mormugao were more than 10 times that of the preceding year. These southern manganese occurrences may be broadly divided into two classes, viz.: (1) Those occurring in conjunction with the South Indian laterites. (2) Those occurring in the hematite-quartzite bands of the Dharwar beds.

The Goa deposits and others toward the Western Ghats come under the first heading. The manganese deposits to be more particularly described are those of the native State of Sandur, Bellary district, in the north of the Madras presidency.

The manganese deposits of the Sandur range are situated at or near the crests of the hills, in the softer rocks of the series. The manganese is mostly a psilomelane, very hard indeed, and of a bright steely appearance where exposed to the atmosphere, but schistose and friable in the interior of the deposits. Braunite is present to a small extent, as also is wad. To the south, ore of a pisolitic nature is found. It occurs only among the plateau deposits and is more intimately related to the iron capping than to the massive deposits.

The manganese ore of the Sandur deposits is remarkably free from harmful impurities. Silica is very rarely over 2 per cent. The phosphorus content is practically negligible and is mostly about 0.01 per cent. On the other hand, the percentage of iron is high, very rarely as low as 5 per cent., and ranging from that upward, through the stages of ferruginous manganese and manganiferous iron to pure hematite. Iron and manganese are, throughout these deposits, intimately associated, sometimes one predominating and sometimes the other. Even the purest manganese deposits have large veins of hematite running through them, and high-grade blocks of manganese are often penetrated by stringers of limonite.

The deposits are at first sight imposing, not only on account of the immense exposures of solid manganese ore, but also in their appearance of continuity. All round the outer ridge of the western range of hills manganese crops out on practically every spur at the same horizon. In penetrating the deposits the ore is found, with certain variations, to be fairly consistent. The outer portion, for a foot or two, is generally the purest and hardest, while the innermost portion, of the same width, is the poorest and softest. The changes in tenor are slight and are confined to the outermost and innermost zones; the remainder of the deposit, which is, of course, of varying thickness,

is homogeneous. Almost without exception the manganese ore ends abruptly against a conformable layer of hematite, which is thin in comparison with the overlaying manganese; the plane of contact between the manganese and iron ore is irregular, but always well defined. The hematite gradually gives place to a ferruginous shale, and this again to a red earth, which in depth gradually loses color and passes into a whitish lithomarge. The change from hematite to lithomarge takes place in a comparatively short thickness, rarely more than 2 ft. to 6 ft. The underlying lithomarge is, in comparison with the superposed manganese, of great thickness and extent. Below the lithomarge the country rock is seen.

Japan.—Manganese is found in various provinces of Japan, the largest and most productive areas being mined in Hokkaido, Aomori-ken, and Kyoto-fu. The most important manganese mine in the Empire, however, is situated at Owani, Mitsu Province, where manganite is also found. The latter occurs in cavities of other manganiferous ore masses, in slender prismatic crystals of a black color and high semi-metallic luster. In addition to the foregoing, the small islands of Oshima, Satsuma, and Sado produce small quantities of a superior quality, the major part of their products being exported. The ore is found in the mountains, usually near the top, and lying close to the surface. It is transported to the seacoast or railroad in baskets carried by coolies. The greater part of the exported manganese is shipped by Suez; occasionally a cargo is sent by sailing vessels by way of Cape of Good Hope. In years past local exporters enjoyed a splendid trade with the United States, but on account of the varying quality of the ore much dissatisfaction was occasioned, with the result that the business has been greatly reduced.

Russia.—The manganese industry in the Caucasus was in a greatly depressed condition in 1908. The stagnation is attributed to two main

STATISTICS OF MANGANESE IN RUSSIA.

Years.	South of Russia.		Caucasus.	
	Mined Poods.	Exported Poods.	Mined Poods.	Exported Poods.
1898.....	3,640,475	329,001	16,259,204	14,468,282
1899.....	5,914,882	1,277,248	34,052,432	22,433,519
1900.....	5,407,860	2,144,191	40,363,492	26,335,531
1901.....	4,243,514	700,000	22,904,535	18,860,179
1902.....	3,503,920	394,300	25,925,115	27,488,992
1903.....	2,091,547	830,611	22,974,603	26,935,785
1904.....	3,693,122	1,193,959	20,256,309	28,544,427
1905.....	9,235,794	2,258,425	20,876,487	18,700,054
1906.....	9,580,000	735,030	50,170,000	26,729,626
1907.....	15,500,000	2,227,000	40,833,000	29,621,710

causes, namely, the depression in the steel industry everywhere and the high freight rates on the Tchiaturi railway. Many large concerns liquidated their business, and others confined themselves to the completion of existing contracts. On the other hand, it is claimed that the trouble arose from the fact that the manganese producers took advantage of the strenuous demand a few years ago to ship inferior qualities of ore, with the result that they lost their customers. However, the railway rate of seven copecks per pood over a distance of 35 versts from Tchiaturi to Sharopan was reduced at the end of 1908 by 1½ copecks per pood.

The Caucasus ore is mined in the Sharopan district of the Kutais province. In the Ural only a small quantity of manganese is mined, approximately 300,000 poods annually, and only by one firm, which handles this ore for the manufacture of colors. The south of Russia, producing a large quantity of manganese ore, is also a large consumer of this ore in steel manufacture.

PRICES OF CAUCASIAN MANGANESE ORE IN ENGLAND.
In pence per unit of metal per ton.

	1903	1904	1905	1906	1907
January	8	8	8	14	
February	8	8	8	14	
March	8 —8¼	8¼	8	14	16—16½
April	8¼—8½	8½	9½	14	
May	8¼—8½	7¼—8½	9	13	
June	8	7½	9	13	
July	8 —8¼	7¼	9	13	10 15
August	8 —8¼	8	9	15	13
September	8¼	8	9	15 —17	10½—11
October	.....	8	9	15 —16	10½—11
November	.....	8	10½	13½—14	9½
December	.....	8	10½	15	9½

The Tchiaturi railway, covering only 35 versts, collects a very high tariff—viz., 5½ copecks per pood, inclusive of other expenses, for goods which cost at the station of shipment only eight copecks per pood. The capacity of the railway is insufficient, and the cars are loaded in proportion to stores of ore, calculating, approximately, one car per 20,000 poods. The miners who wanted a considerable number of cars bought their turns from other miners, and hence a speculation arose, which led to a difference in the price of manganese of nine copecks per pood. The amount of manganese ore stored on Jan. 1 of successive years proves the incapacity of the railway as follows: 1904, 25,549,000 poods; 1905, 19,578,000 poods; 1906, 37,701,000 poods; 1907, 72,377,000 poods; 1908, 94,590,000 poods. These figures speak for themselves, and there is no wonder that some miners have taken to road transport.

As is well known, the manganese deposits in the Caucasus have the characteristic feature of nests of small depth. As a result, the manganese ore is chiefly mined by small producers. The average is 100,000 poods ore per producer, and only 11.5 workmen. It will be easily understood, therefore, what losses the miners have suffered. But the whole industry suffers, too. The high tariff, plus different station, port, transshipment, and other charges renders the ore on board extremely expensive, thus giving advantages to competitors in India and Brazil. During the first six months of 1907 a pood of manganese in Tchiaturi cost at truck 10 to 11 copecks, and on board steamer in Poti 32 to 37 copecks. Thus the transport along a distance of 38 to 43 versts and shipment increases the cost of the product by 22 to 25 copecks—i.e., by 200 to 250 per cent. As a consequence, the miners get into bankruptcy and the mines do not work.

The Statistical Bureau of the Council of the Congress of South Russian Mineowners has compiled the following statement on the export of Russian manganese ore abroad in 1908 compared with 1907—namely, 3,111,000 poods from Nikopol and 24,084,710 poods from the Caucasus, against 5,163,240 poods and 29,621,710 poods in 1907 from the respective sources named, or a total of 27,196,670 poods in 1908 against 34,784,950 poods in 1907, the decline being 7,588,280 poods. The Nikopol manganese ore was exported through the following ports in 1908: Sosnowice, 670,800 poods; Granitza, 1650 poods; and Nikolaieff, 2,439,510 poods. The quantities of manganese ore taken per country from the Caucasus were as follows: Holland, 9,792,980 poods in 1908 against 12,778,320 poods in 1907; England, 6,926,250 poods against 10,639,020 poods; Austria, 1,126,890 poods against 2,530,910 poods; Germany, 967,020 poods against 1,727,100 poods; France, 1,840,790 poods against 1,524,950 poods; Belgium, 3,405,880 poods against 420,800 poods; Italy, nil, against 610 poods; and Turkey, 24,900 poods against nil; in each case for the years 1908 and 1907 in the order named.

According to the *Golossa Priuralya*, the mining engineer, Polyakonoff, who was sent on a geological expedition, describes a series of manganese deposits stretching over an area of 60 versts from the village of Brien to that of Obrutcheff. According to the newspaper quoted, these deposits are distinguished by extraordinary richness, resembling that of the Caucasus. It is also reported that large deposits of manganese ore have been discovered near the village of Biaely Klyutch, in the district of Tiflis. The metallic content of this ore is stated to be very high.

Value of Manganese Ore.

The Carnegie Steel Company, which is the largest consumer of manganese ore in the United States, paid during 1906, 1907 and up to the

middle of 1908 the prices quoted below for manganese-iron ores delivered at the Lucy furnaces, Pittsburg, Penn., the Edgar Thomson furnaces, Bessemer, Penn., or the South works of the Illinois Steel Company, South Chicago, Ill. The prices are per 2240 lb. and per unit of metallic manganese: Above 49 per cent. $0.30; 46 to 49 per cent. $0.29; 43 to 46 per cent. $0.28; 40 to 43 per cent. $0.27. Iron is paid for at the rate of 6c. per unit. These quotations are based upon ore containing not more than 8 per cent. silica and not more than 0.25 per cent. phosphorus, and are subject to the following deductions: For each 1 per cent. silica in excess of 8 per cent., 15c. per ton is deducted, fractions in proportion; for each 0.02 per cent., or fraction thereof, of phosphorus in excess of 0.25 per cent., 2c. per unit of manganese is deducted. Ore containing less than 40 per cent. manganese or more than 12 per cent. silica or more than 0.27 per cent. phosphorus is subject to refusal or acceptance at the buyer's option. Settlements are based on analysis of sample dried at 212 deg. F., and the percentage of water found by this drying is deducted from the gross weight of the shipment. This price list was withdrawn on July 1, 1908, and up to the end of that year no substitute was announced.

High-grade ores, used chiefly as a source of oxygen in storage batteries and as a dryer in paints and oils, command higher figures than the ordinary ones used by the steel makers. Prices of high grade ores range from 1 to 20c. per lb., depending upon the ease with which the contained oxygen may be liberated, rather than upon the percentage of this element present. Thus of two samples carrying the same amount of oxygen, one may be much more valuable than the other. The only method of determining this point is by an actual working test of the ore.

According to *Min. Journ.*, Feb. 6, 1909, Russian manganese ore is generally sold upon the basis of 48 per cent. or of 49 per cent., and sometimes of 50 per cent., minimum contents of metallic manganese. It is unusual to give a sliding scale for contents of under the basis, but if such be done it is usual for the sellers to pay a penalty, under the basis, of double the contract value of the unit. For instance, if contract is for 50 per cent. basis and it results in 49 per cent., then allowance must be made as if the test were 48 per cent. As to silica, a penalty of three pence or four pence per ton is made for each degree in excess of 10 per cent. Indian manganese ore is at present being sold upon a basis of 45 per cent., 48 per cent., and 50 per cent., with sliding scales of 6d. per unit, 50 per cent. to 48 per cent., and a farthing per unit to minimum of 45 per cent., and on the 45 per cent., a farthing per unit to minimum of 42 per cent. Silica is 8 per cent. basis, with three pence per ton penalty for each unit over 8 per cent. At present Russian ore

sells at 8¼d. to 8½d. per unit, and the Indian ore at 8¾d. on 45 per cent., 9d. on 48 per cent., and 9½d. on 50 per cent. This is for delivery c.i.f. usual ports of delivery; sampling mutually by representatives of both sides at time of discharge, and analysis by usual official chemists. All moisture to be deducted from the weight. Payment for freight as per bill of lading, 75 per cent. of shipping value against documents by 60 days' sight London bank acceptance, and balance in cash upon rectification of invoice. Phosphorus scale is a matter of bargaining. The consumers prefer to have a definite maximum fixed for this. If a sliding scale be adopted, it is generally 6d. per ton for each 0.01 per cent. over the basis that is applicable to the mineral.

MICA.

By John Tyssowski.

Owing chiefly to the financial depression, 1908 was very unsatisfactory in the mica trade. During the year the United States imported mica to the value of only $266,056, a falling off of $649,203 from the figure for 1907. As usual about two-thirds of the total importation came from India and practically all of the remainder from Canada. The accompanying table shows the production and importation of mica in the United States for a period of years.

STATISTICS OF MICA IN THE UNITED STATES.
(In pounds and tons of 2000 lb.)

Year.	Production. (a)			Imports.			
	Sheet. (b)	Scrap.		Unmanufactured.		Cut or Trimmed.	
	Pounds.	Tons.	Value.	Pounds.	Value.	Pounds.	Value.
1897	118,852	2,882	$28,820	722,939	$161,334	226,771	$41,068
1898	110,918	3,529	39,537	877,930	115,930	78,567	34,152
1899	97,586	6,917	50,596	1,709,839	233,446	67,293	42,538
1900	127,241	5,417	42,889	1,892,000	290,872	64,391	28,688
1901	360,600	2,171	19,719	1,598,722	299,065	78,843	35,989
1902	373,266	1,400	25,006	2,149,557	419,362	102,299	46,970
1903	619,600	1,659	25,040	1,355,375	288,783	67,680	29,186
1904	668,358	1,096	10,854	1,095,343	241,051	61,986	22,663
1905	851,800	856	15,255	1,506,262	352,475	88,188	51,281
1906	1,423,100	1,489	22,742	2,984,719	983,981	82,019	58,627
1907	1,060,182	3,025	42,800	2,227,460	838,098	112,230	77,161
1908	972,964	2,417	33,904	497,332	224,455	51,040	41,601

(a) Statistics for 1901 to 1908 inclusive are those of the U. S. Geological Survey. (b) The value of sheet mica being so widely variable, and so little indicative of commercial results, and all previous statistics being of doubtful accuracy, they have been omitted from this table.

Market Conditions and Prices.—At the opening of 1908 most dealers had accumulated large surplus stocks, and the anticipated increase of demand failing to materialize, a generally inactive and depressed market resulted. During the closing months of the year the uncertainty over tariff revision checked all strengthening tendencies of the market despite the gradually improving conditions in general trade.

The prevailing prices during 1908 were on the average at least 10 to 25 per cent. below those of 1907. It is difficult to make any general statement as to the average price per pound of mica as, according to size and quality, the prices have a wide range. Average list figures, not considering discounts, may be stated as follows: Cut electrical, $1.75@20 per lb.; India uncut sheet, $0.35@3; amber uncut sheet, $0.65@2.50; ground mica up to 15c. per lb. in small lots, varying with the fineness. The wholesale

quotation on ground mica was 1¼@3c. per lb. Scrap mica was marketed at $7@15 per ton. A remark made by one of the principal New York dealers illustrates the general condition of prices. In speaking of the market in 1908 he said: "The other day I was called on to give my best quotation on some stuff that I would have sold for $3.50 per lb. last year. I offered it at $1.75 and lost the sale."

Mica in the United States.

Mica is found in almost every State in the Union but in commercial quantities only in Colorado, North and South Dakota, Alabama, Georgia, North Carolina, New Hampshire, Vermont and Maine. The product of these States is 95 per cent. muscovite, or white potash mica, which is harder than the Canadian phlogopite, or magnesia (amber) mica. There are also in the United States, and Canada as well, deposits of black magnesia-iron mica (biotite), which have commercial value.

Alabama. (By E. A. Smith.)—Mica is widely distributed in the region of the crystalline rocks, occurring in pegmatite veins which cut the other rocks of the region. One well known belt extends from Chilton, through Coosa, Clay, Cleburne and Randolph counties, the most important locality being near the adjoining corners of Clay, Cleburne and Randolph. Other occurrences of note are in Tallapoosa, Lee and Chambers counties. During 1908 about 40 tons of merchantable mica were produced. Owing to the distance of the mines from the plant at Heflin in Cleburne county, the absence of railroad facilities and the expense of hauling the raw material over the dirt roads, one company engaged in the business recently ceased operations. A change in these unfavorable conditions would bring about a great increase in the amount of mica produced, for the quality is good and the quantity ample.

New Hampshire.—Most of the mines were not operated during 1908, but a small production was made, chiefly from working over old mine dumps. In some cases the material was mined 90 years ago. The total yield of the State probably did not exceed 700 tons.

North Carolina. (By Joseph Hyde Pratt.)—There was a large falling off in the production of mica in North Carolina during 1908, due to the general business depression. The production[1] reported for 1908 was 390,548 lb. of sheet mica, valued at $110,266, which, compared with the 1907 production of 645,221 lb. of sheet mica valued at $290,956, is a decrease of 244,673 lb. in quantity and of $99,690 in value. Besides the sheet mica, there was also produced 1027 tons of scrap mica, valued at $10,326, which is 344 tons in quantity and $4974 in value less than the production of 1371

[1] The statistics of the mica production were obtained by the N. C. Geological Survey in coöperation with the U. S. Geological Survey.

tons, valued at $15,250, in 1907. There were 69 operators who reported a production of mica in 1908. Most of these, however, sold their mica in the crude state to other operators, who prepared the material for market. At present, a large proportion of the mica produced in North Carolina is manufactured and prepared for the electrical and stove trade within the State. This is also true of the scrap mica, a considerable portion of which is now being ground in North Carolina.

Mica occurs in many counties in western North Carolina and has been produced in at least 20 of these counties. In 1908 mica was mined in Mitchell, Yancey, Macon, Jackson, Haywood and Madison counties, although the larger proportion of the production was obtained from the first two.

South Dakota.—The Westinghouse Electric and Manufacturing Company owns and operates four of the largest mica mines in the United States. They are situated near Custer in Custer county. The company operated only its No. 1 mine in 1908. The Westinghouse mines are considered the most valuable ones in the State; as far as has been ascertained the No. 1 mine was the only one operated to any extent during 1908. The production of rough mica was 1,250,000 lb.

Mica in Foreign Countries.

Canada.—The extensive demand in the electrical trade for Canadian amber mica has resulted in an important development of the mica mines. Practically all the mica occurs in soft rock formations and is usually worked by quarrying methods. In some cases the country rock is a soft phosphate rock, the mica being a by-product. These conditions make cheap mining possible. The production in 1908 was valued at $191,602, as against $312,597 in 1907, a falling off of 38 per cent. This decrease is no doubt a result of the trade depression which was prevalent during 1908, and in greater measure, of the decline of American demand.

In Quebec little work was done in the mines and there was little activity in the trimming shops in Ottawa. The chief mines which were operated and shipped mica were those of Kent Brothers, W. Argall, H. E. Flynn, Blackburn Brothers and the Calumet Mining Company. The total production of the province was 106 tons of thumb trimmed mica valued at $95,311. About $48,000 was paid out in wages by the mica companies, which employed less than 200 men.

Canadian phlogopite (amber mica) is characterized by its softness and is classified for commerce according to size. It is designated as 1x1, 1x2, 1x3, 2x3, 2x4, 3x5, 4x6, 5x8, All classes are slightly flexible and are generally graded by the area, 5x8 grade being 40 sq. in. and over; 4x6 grade, 24 to 40 sq. in.; 3x5 grade, 15 to 24; 2x4 grade, 8 to 15; 2x3 grade, 6 to 8; 1x3 grade, 3 to 6; 1x2 grade, 2 to 3; 1x1 grade, 1 to 2.

China.—W. T. Gracey, United States Consul at Tsingtau, reports that at Tschoutschong, in the province of Shantung, there are mica deposits of good quality.

France.—A new variety of mica, with exterior characteristics similar to those of muscovite, but resembling paragonite in composition, has been discovered at Mesvres by P. Barbier. The mineral has a lithia content comparable to that of the lepidolites and has been named "hallerite" by its discoverer.

India.—India has for many years been the leading producer of mica, turning out more than half the world's supply. The two principal areas of production are the Nellore district in the Madras presidency, and a belt 12 miles broad and about 60 miles long, stretching obliquely across the junctions of the Gaya, Hazaribagh and Monghyr districts in Bengal. During the last three years Indian mica has averaged in value £5 per cwt., and although the production for 1907 was less than that for the previous year by 28 per cent., the average value of the mica was higher by 20 per cent. It is white and intermediate in hardness between the Canadian and United States varieties. In the sheet it is sold largely in grades, viz.: Extra A1, A1, 1, 2, 3, 4, 5, and 6. The several grades are made up of rough-trimmed sheets of the following areas: Extra A1, 48 sq. in. and larger; A1, 36 to 48 sq. in.; No. 1, 24 to 36; No. 2, 15 to 24; No. 3, 10 to 15; No. 4, 6 to 10; No. 5, 3 to 6; No. 6, 1 to 3. During 1908 the general trade depression affected the industry, but as the demand for mica was fairly well maintained by British and Continental buyers the slump was not serious. The export to England amounted to 21,309 cases and bags averaging 1 cwt. each, as compared to 19,925 in 1907.

PRODUCTION OF MICA IN INDIA.
(In metric tons.)

1899	479	1902	806	1905	1,174
1900	1,025	1903	1,002	1906	2,463
1901	1,505	1904	828	1907	1,772

South Africa.—The most important discovery of mica during 1908,—one which, if reports are not exaggerated, bids fair to place South Africa among the foremost mica-producing countries of the world—is reported from the northern Transvaal. The new fields are about 25 miles east of Loydsdorp, near the Olifants river. The strike of the deposit is approximately east and west, and it is claimed that the micaceous formation, pegmatite, has been traced over a distance of 50 or 60 miles. The mica is stated to be of remarkably pure quality, to contain no ferruginous matter, and to be fairly free from vegetable stains. Excellent sheets 16x8 in. in size, have been taken out, and tests show the mica to be capable of with-

standing an exceptionally high voltage. Being situated in the "low country," most of the work has to be carried on in the winter season. Already some mica has been shipped and the quality reported as good. The new region is known as the Macutsi mica fields.

Economics of Mica Mining.

Two factors aid in determining the economic value of a mica deposit; namely, the extent and persistence of the deposit, and the physical character, hardness and transparency of the mica. Of course the first consideration should always be thoroughly investigated before mining operations are undertaken. The character of the deposit will naturally determine the method of mining most advantageous for its exploitation. Open-cut work is cheapest and necessitates the least original expenditure and in New Hampshire where portable shelters have been used has proved satisfactory. Underground work is however the preferable method where the deposit is of sufficient size and quality to warrant the outlay needed for working through shafts. In general at least 1250 lb. of mica must be extracted from every 100 tons of rock mined by underground methods and in open-cutting 800 lb. per 100 tons of rock quarried. A mine yielding 5 per cent. of mica sheets cutting 4x6 in. and 50 per cent. only 3x1 in. must yield mica, so that at least 1000 lb. of trimmed material can be obtained from 100 tons of rock at a daily cost of about $85.

As to the physical characteristics of the mica, flexibility is essential, as soft weathered mica can only be used in making paint. Transparency, another very desirable attribute of the sheets, seems to be affected by the conditions prevailing at the time of crystallization. In some cases quartz or feldspar is deposited in films between the cleavage planes; in others the impurities form a portion of the mica sheets. Disturbances at the time of crystallization also cause the mica to be opaque. In some instances the impurities are easily removed by washing without affecting the nature of the product. The size of the sheets which may be obtained is an important consideration, as the larger sizes are always in demand at the best prices.

Uses of Mica.

Of late years, paralleling the growth of the electrical industry, the demand for mica has been increasing. Mica is still used for the same purposes as formerly but now the demand for the electrical trade far overshadows all others. There are two varieties of mica commercially valuable, muscovite and phlogopite. The former, white mica, is used in the stove, electrical and lamp-chimney trades, and, when ground, in the wall-paper and other decorating trades; also for roofing, as a lubricant, as an annealer,

and in built-up electrical-composition work. The latter variety, commercially termed amber mica, is used almost entirely in the electrical trade, and when ground is employed for the same purposes as the muscovite, with the exception of decorative work. It excels all other varieties for use in the electrical trade as its hardness is such as to make it exactly suitable for use as an insulating medium in commutators where it wears evenly with the copper. Made up of sheets of hydraulically pressed mica, as described in Vol. XVI of THE MINERAL INDUSTRY, are extensively used in all electrical work.

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Description of the newly discovered French mica, Hallerite. *Comptes Rendus de l'Académie des Sciences,* June 9, 1908.

Cirkel, Fritz. The Mica Industry in Canada: *Eng. and Min. Journ.,* April 19, 1908, p. 801.

The following patents of note were issued during 1908:

Method of Treating Mica. J. P. W. Beckman, Parnassus, Penn. (U. S., No. 903,949 of 1908.)

Process for Production of Mica Powder. Arthur B. C. Rogers, Agra, British India. (Brit., No. 12,570 of 1908.)

Machine for Splitting Mica. Bevahood Wolchner, New York, N. Y. (U. S., No. 901,130 of 1908.)

Mica Trimmer. Edmund G. Shepard, Ottawa, Ont., Canada. (U. S., No. 888,197 of 1908.)

Machine for Folding Mica Sheets. Edward Cooper, Newton, Mass., assignor to American Mica Company. (U. S., No. 890,500 of 1908.)

MINERAL WOOL.

The mineral wool industry of the United States in 1908 was practically the same as in 1907, except that prices were lower. No new producers entered the field and no changes or innovations in manufacture were recorded. In 1908 the production was 9197 short tons valued at $77,228 as compared with 9008 tons worth $81,769 in 1907. The accompanying table gives statistics for 11 years except for 1903 and 1904 when no statistics were collected.

PRODUCTION OF MINERAL WOOL IN THE UNITED STATES.
(In tons of 2000 lb.)

Year.	Amount.	Value.	Per Ton.	Year.	Amount.	Value.	Per Ton.
1898	6,560	$70,314	$10.72	1904	(a)	.....	.....
1899	7,448	85,899	11.53	1905	6,164	$69,560	$11.28
1900	6,002	60,320	10.05	1906	5,375	55,550	10.33
1901	6,272	68,992	11.00	1907	9,008	81,769	9.08
1902	10,843	105,814	9.67	1908	9,197	77,228	8.40
1903	(a)	.....	.....				

(a) No statistics collected.

There were but five companies making mineral wool in 1908. These were: United States Mineral Wool Company, 140 Cedar street, New York; Columbia Mineral Wool Company, 112 Clark street, Chicago, Ill.; Pennsylvania Mineral Wool Company, Norristown, Penn.; Union Fibre Company, Yorktown, Ind., and Banner Rock Wool Company, Alexandria, Ind. The first three make their product from furnace slag and the others from sandstone or other silicious rock. In the West the Union Fibre Company is the largest producer with the Columbia company second. In the East the United States company leads in production. About 70 per cent. of the wool produced from furnace slag is used to insulate the floors and walls of buildings, its non-conducting properties making it valuable for this purpose. The remainder is used for covering water pipes, and for use in refrigerators. The principal use for rock wool is in the form of compressed blocks used in cold storage plants. About 65 to 75 per cent. of the production is thus utilized. The remainder is used in building construction.

MOLYBDENUM.

A small amount of molybdenite was produced in 1908, the total for the United States being about 15 tons. This was obtained chiefly in Oregon. Several companies in Washington were engaged in prospecting and mine development, and a small production was made there. Several companies are interested in working the deposits at Cooper, Tunk Pond, and near Machias, Maine. The small production of this ore is not due to the absence of deposits, but rather to the difficulties of concentration and the smallness of the demand. The use of molybdenum steel does not appear to make any considerable headway. But little molybdenum ore was used in 1908. Some of the principal purchasers had enough ore on hand at the beginning of the year to last through the year and consequently they did not buy any.

Molybdenum ore, guaranteed to contain 90 to 95 per cent. MoS_2, was quoted in 1907 at $400@450 per ton. In 1908 the price was lower. The price for this ore is widely variable and usually is a matter of negotiation between buyer and seller. The Primos Chemical Company, of Primos, Delaware county, Penn., and De Golia & Atkins of San Francisco, Cal., are the principal dealers. Friedrich Krupp, Essen, Germany, is a large user of molybdenum.

MONAZITE.

The monazite industry in the United States fell off badly in 1908, the production being but little over three-quarters of that for 1907. This is accounted for in a measure by the general business depression, but the fundamental reason is that thorium nitrate was imported at such a greatly reduced cost that it was cheaper in some cases to buy that compound than to manufacture it. The following figures represent, in order, the year, number of pounds, total value, and value per pound of thorium nitrate imported: 1903, 72,990, $244,258, $3.35; 1904, 7,159, $261,232, $3.42; 1905, 38,274, $200,238, $5.23; 1906, 57,892, $254,858, $4.41; 1907, 88,653, $240,128,$3.71; 1908, 65,289, $173,239, $2.66.

In the United States monazite sand is found in commercial quantities only in the Carolinas, but deposits also occur in Georgia, Idaho, Oregon and in the other States on the Pacific slope. The accompanying table shows the production of monazite in the United States and in North Carolina for a series of years.

MONAZITE PRODUCTION IN THE UNITED STATES.
(In pounds.)

Year.	United States. (a)			North Carolina. (b)		
	Pounds.	Value.	Per Pound.	Pounds.	Value.	Per Pound.
1897	44,000	$ 1,980	$0.045	44,000	$ 1,980	$0.045
1898	250,776	13,542	0.054	250,776	13,542	0.054
1899	350,000	20,000	0.057	350,000	20,000	0.057
1900	908,000	48,805	0.054	908,000	48,805	0.054
1901	748,736	59,262	0.079	748,736	59,262	0.079
1902	802,000	64,160	0.080	802,000	64,160	0.080
1903	862,000	64,630	0.075	773,000	58,694	0.076
1904	745,999	85,038	0.114	685,999	79,433	0.116
1905	1,352,418	163,908	0.121	894,368	107,324	0.120
1906	846,175	152,312	0.180	697,275	125,510	0.180
1907	547,948	65,754	0.120	(c)456,863	54,824	0.120
1908	422,646	50,718	0.120	(c)310,196	37,224	0.120

(a) Statistics of the United States are those of the U. S. Geological Survey. (b) The figures for North Carolina, from 1897 to 1906, inclusive, are from "The Mineral Industry of North Carolina." (c) The figures for 1907 and 1908 were collected jointly by the U. S. Geological Survey and the N. C. Geological and Economic Survey.

MONAZITE IN THE CAROLINAS.*

Thus far North and South Carolina are the only States which have put monazite on the market. The deposits in the Carolinas are in the

* Abstract from a paper by Joseph Hyde Pratt, Chapel Hill, N. C., and Douglas B. Sterret, Washington, D. C., in *Trans.* A. I. M. E.

south-central part of western North Carolina and in the extreme northwestern part of South Carolina, in an area covering about 3500 square miles. This region is a belt from 20 to 30 miles wide and over 150 miles long, lying wholly within the Piedmont plateau and bordering close to the foot of the Blue Ridge.

Physical Properties.—The thoria content in the monazite sand of North Carolina varies from 0.01 to 7 per cent. The monazite varies in color from light yellow to reddish brown or greenish yellow, is vitreous or resinous in luster, has a hardness of 5 or 5.5 and a specific gravity ranging from 4.64 to 5.3. The mineral is brittle with a conchoidal fracture and crystallizes in the monoclinic system. It is incompletely soluble in hydrochloric but is readily attacked by sulphuric acid.

Commercial Deposits.—The original occurrence of monazite is in gneiss, pegmatized gneiss, schist, pegmatite and granite. So far, however, the placers are the only deposits of commercial importance, although saprolite, or rotted rock, down to depths of a foot or so, is sometimes sluiced along with the overlying gravel. The gravels range from 1 ft., including over-burden, to 8 ft. in thickness, and, as would be expected, the monazite is concentrated in the bottom of the deposits. The extent of the deposits varies with the topography of the country and the nature of the gravel. In some places the bottom lands containing monazite-bearing sand are over 300 ft. wide and extend over half a mile along the stream. Experience has shown the best class of deposits to be associated with light-colored sand and gravel, that contain quartz and fragments of the rock in which monazite originally occurs. When such material is wanting, the deposits are usually of a low grade. The presence of black sand is not necessarily indicative of the grade of the deposits.

Treatment.—Almost without exception all the monazite mined in the Carolinas has been obtained from gravel deposits, being recovered in sluice boxes fed by hand. Wilfley tables have been introduced for treating the concentrates from the sluice boxes. These concentrate up to 50-90 per cent. of monazite, depending upon the amount of black sand present. Magnetic separators are used to concentrate the Wilfley product up 90 to 98 per cent., which is the grade of material shipped to manufacturers of incandescent mantles.

Manufacture of Incandescent Mantles.—These mantles are generally made of selected cotton fiber, which is purified so as to remove all possible traces of mineral matter. The knitted fabric is saturated with lighting fluid, composed of one part of a mixture of approximately 99 per cent. thorium nitrate with 1 per cent. cerium nitrate, and three parts distilled water, then dried and ignited. After this the mantles are shaped, fire-

tempered and tested, finally being covered with collodion, when they are ready for the market.

By-Products.—Didymium, used in the form of the nitrate for indelible branding, is also obtained from monazite. Zircon, which is a by-product in monazite mining, has a commercial value of from 20 to 25c. per lb. for the zirconia content and is used in the glower of the Nernst lamp. It has also been suggested to use the garnet, obtained in concentrating the monazite sand, for manufacturing abrasive garnet paper for the boot and shoe trade. The magnetite and ilmenite may also possibly be used in the manufacture of magnetic electrodes.

Monazite in Brazil.

The greater part of the world's supply of monazite comes from Brazil, where it is found in commercial quantities only in the alluvial deposits in the states of Rio de Janeiro and Minas Geraes, and along the coast from Rio to Bahia. Its original occurrence is in certain felspathic rocks, pegmatite, syenite, and gneiss, in which it constitutes from 0.07 to 0.2 per cent. of the weight. The present deposits have been derived by erosion and concentration of such rocks.

The interior deposits are found in the beds of dry water courses in the form of gravel and sand containing not more than 2 per cent. monazite, together with quartz, garnet, ilmenite and other heavy minerals. The deposits sometimes lie on the surface, and sometimes under a bed of clay as much as 3 m. thick. The sand is mined, concentrated roughly, and finished on electro-magnetic machines which make a 95 to 96-per cent. monazite product, containing, on the average, 3.7 per cent. thorium oxide. The exports of monazite sand from Brazil in recent years have been as follows: 1903, 1862 metric tons; 1904, 2901; 1905, 1039; 1906, 4352; 1907, 4437.

During 1908 the monazite deposits in the provinces of Bahia and Espirito Santo were worked chiefly by Messrs. Gordon, Israelson and Freytas & Co. The deposits in Bahia are stated to have given a low yield but the importance of those in Espirito Santo has been increased by new discoveries. Messrs. Israelson are reported to have been engaged in negotiations with the provincial government of Espirito Santo for the acquisition of a monopoly, in return for a considerable premium and the erection of a thorium works in the province. A Franco-Brazilian company—the Société Minière Industrielle Franco-Brésilienne worked the deposits at Itabapoana (on the borders of the provinces of Espirito Santo and Rio de Janeiro), concentrating the monazite sand to .95 per cent. grade. The site of these operations is a disputed strip of territory

between the two provinces, and consequently there has been a dispute as to the concession, resulting in the closing down of the works.

Mr. Freise (*Montanistische Rundschau*, Nos. 16 and 17, of 1909), reports on the monazite deposits on the upper reaches of the Rio Muriahé, Minas Geraes, which he states to be of considerable importance, both from their extent and unsurpassed richness in thorium minerals. The sands are in a very accessible position, could be worked systematically owing to the uniform deposition, and are capable of yielding marketable by-products under normal conditions, while the labor question and transport arrangements present no special difficulties.

NATURAL GAS.

In spite of the immense drain upon the supply of natural gas in the United States, the comparatively short life of any single pool, and the complete exhaustion of the resources in many districts of Pennsylvania, Ohio, and Indiana, the production continues to increase. This is due undoubtedly to the more extensive piping of the gas to comparatively remote points, and the connection to trunk lines of many isolated pools. Formerly, natural gas was used only in the district of its production where manufacturing industries were established to take advantage of a very cheap fuel. This condition has changed even in such new regions as Kansas, and natural gas is now piped for distances as great as 200 miles, being delivered from West Virginia to points so distant as Cleveland, Ohio, and from southeastern Kansas to Kansas City. This has introduced a great change in the industry, and in many districts has raised the value of natural gas so that it is no longer a very cheap fuel; indeed, in many industries that once employed natural gas there is no longer any advantage over coal, all things considered,

PRODUCTION OF NATURAL GAS IN THE UNITED STATES IN 1906 AND 1907. (a)

State.	1906			1907		
	Quantity, M cubic feet.	Cents per M cu. ft.	Value.	Quantity, M cubic feet.	Cents per M cu. ft.	Value.
Pennsylvania	138,161,385	13.4	$18,558,245	135,516,015	13.9	$18,844,156
Ohio	45,426,020	15.7	7,145,809	52,040,996	16.8	8,718,562
West Virginia	119,400,392	11.5	13,735,343	122,687,236	13.6	16,670,962
Kansas	69,322,633	5.8	4,010,986	74,526,300	6.5	4,843,019
Missouri	33,560	21.5	7,210	106,090	15.7	17,010
New York	2,547,769	26.4	672,795	3,287,974	23.3	766,157
Indiana	7,861,140	22.2	1,750,715	6,624,208	23.7	1,572,605
Kentucky	789,154	36.4	287,501	1,303,158	29.2	380,176
Oklahoma	3,520,396	7.3	259,862	4,867,031	8.5	417,221
California	153,021	87.9	134,560	230,344	73.1	168,397
Alabama, Louisiana, Texas	1,038,569	14.5	150,695	1,287,734	13.8	178,276
Illinois	409,556	21.3	87,211	1,154,344	12.4	143,577
Colorado	23,567	96.7	22,800	(c)	.....	.....
Arkansas, Wyoming	120,500	28.6	34,500	766,988	16.5	126,582
South Dakota	22,900	67.2	15,400	37,500	52.0	19,500
Tennessee	2,000	15.0	300	2,000	15.0	300
North Dakota	(b)	.....	.....	940	25.0	235
Oregon	(b)	.....	.....	400	25.0	100
Total	388,842,562	12.1	46,873,932	404,441,254	13.07	52,866,835

(a) Statistics of U. S. Geological Survey. (b) Not reported. (c) Included with Wyoming and Arkansas.

and to a more or less extent there is already a reversion to the use of coal. The State of Oklahoma in its constitution forbade the piping of natural gas outside of its limits, wherefore, the supply in that State is still preserved for local manufacturing purposes.

Kansas and Oklahoma. (By Erasmus Haworth.)—But little change occurred in the gas situation in the Mid-continental field during 1908. At the present time the consumption in Kansas is exceedingly large, while that in Oklahoma is but little more than a year ago. In Kansas the Kansas Natural Gas Company is the largest producer. The principal addition to its pipe lines during 1908 was in a greater extension of the service in Kansas City, Mo. At the close of 1907 less than half of Kansas City was supplied with natural gas, but before the winter season had passed the major part of the city was reached with the supply pipes. Pipe-laying was continued during the summer so that at the present time practically all of Kansas City and its suburbs are supplied with natural gas from the Kansas fields. Here is a city which, including Kansas City, Kan., and all suburbs, has a population of nearly 400,000 using natural gas for domestic purposes. Nearly all of the smaller factories use it, while some of the larger factories employ the gas during the summer season.

The supply pipe lines were also considerably extended in the Joplin zinc mining district, thus reaching hundreds of consumers which were not supplied during 1907. The Kansas Natural Gas Company's pipe lines extend east from the gasfields as far as Joplin, Mo., and supply gas to every town and village along the line, including Parsons, Oswego, Cherokee, Pittsburg, Baxter Springs, Galena, Joplin, Webb City, Carterville, Carthage and many other places. The company's pipe line also extends north from the gas fields to Lawrence and Topeka, to Leavenworth, Atchison, St. Joseph, Mo., and Kansas City. During 1907, according to official statements of this company, the income from sale of gas was a little more than $3,000,000. For 1908 it will aggregate about $4,000,000.

A great deal of complaint was made in the winter of 1907 at Kansas City about the insufficiency of the supply of gas. Similar complaints were made from St. Joseph and Topeka, but to a less degree. The gas company answered with the statement that large pumps that would guarantee an abundant supply were in process of being installed; during the latter part of the winter and throughout the summer of 1908 a large pumping plant was installed at Scipio, a few miles north of Garnett, Kan., and another large one in the heart of the Montgomery county gasfields, five or six miles south of Independence. The months of November and December, 1908, were so mild that no opportunity offered for testing the adequacy of these pumps.

The second largest producer of gas in the Mid-continental field is the Wichita Gas Company, which has laid a pipe line from the Montgomery county field westward to Winfield, Arkansas City, Wichita, Hutchinson and other towns in south central Kansas. This company has holdings in Oklahoma but obtains the main part of its gas from the Montgomery county fields, picking up a very considerable amount from various wells in Chautauqua and Cowley counties. In addition to supplying practically every residence and business house in the various towns and cities reached by its line, the company also supplies the smaller manufacturers and to a considerable extent the salt plants at Hutchinson and Sterling, where large quantities of evaporated salt are produced.

The zinc smelters and portland cement plants come next in quantity of consumption, particularly the cement plants. The year 1908 was not prosperous for zinc smelters, but most of the cement plants were in operation practically all the time, and consumed more gas than in 1907. Three new plants were added to the list of consumers, one at Fredonia, Kan., which started late in 1907; one at Dewey, Okla., which began operations in the spring of 1908; and one at Chanute, Kan., which began during the latter part of the summer of 1908. With these additions it is evident that the total gas consumed by cement plants was fully equal and probably greater than that in 1907. The glass plants, brick plants and other factories remained about as in 1907.

The rate at which natural gas is sold is almost uniformly 25c. per 1000 cu.ft. for domestic consumption. This is the price throughout practically the entire extent of the Kansas Natural Gas Company's pipe line service. In a few places a rate of 30c. is made. The Wichita Gas Company likewise charges 25c. in most places, but in the western part of its territory the rate is 30c. Each pipe line makes a great reduction for factories, the rate of 10c. per thousand being most common. One or the other of these companies has bought out the local company in almost every town and village in the gas region, so that at the present time citizens of Topeka, Lawrence, Leavenworth, etc., obtain gas as readily and at the same rates as do the little villages in the heart of the field.

In Oklahoma conditions are different from what exist in Kansas. The rulings of the secretary of the interior as to leased Indian land and the enforcement of the new law in the State prohibiting the piping of gas out of the State, thus far have prevented any considerable amount of gas being taken outside Oklahoma. One of the results is that no strong company has yet laid extensive pipe lines anywhere in that State for the purpose of supplying gas to the market. Each town in a measure, therefore, has its own pipe line, and obtains its supply from nearby

pools. Very recently the Henry Oil and Gas Company arranged to build a 12-in. pipe line from the Hogshooter district into Bartlesville, a distance of from 12 to 15 miles. This became desirable on account of the rapid growth of Bartlesville, particularly in the manufacturing plants established. Likewise, Tulsa is preparing to increase its gas supply by laying a large pipe line from the strong gas wells in the vicinity of Collinsville, a distance of about 20 miles.

Rates paid by consumers of gas in Oklahoma differ greatly, not only for factory purposes, but for domestic purposes, as well. A number of the communities are supplied by the old-fashioned flat rates at from 5 to 10c. a gas jet per month for lighting purposes, and from $1 to 2.50 on stoves per month for heating purposes. Elsewhere, but not very commonly, meter rates are established. For manufacturing purposes gas is still offered at very low rates at a number of different places. Bartlesville, for example, immediately upon completion of arrangements to bring gas from the Hogshooter district, published far and wide that it would supply factories using large quantities of gas at the rate of 2½c. per 1000 cu.ft. for a number of years with an increase to 3c. later, which price was to rule throughout the life of the gasfields. Such offers are made in good faith and are used by different cities as inducements to draw factories.

Little attempt was made to obtain gas in Oklahoma during 1908, such wells as were obtained being incidental to oil developments. The only important exception to this statement lies with the Kansas Natural Company which did some drilling in Oklahoma expressly to develop gas. No new gasfield appeared, but here and there throughout the entire district frequently while drilling for oil, good gas wells were obtained. In this way the supply was increased faster than the consumption; so that at the end of 1908 the portion of the Mid-continental field lying in Oklahoma is capable of producing a larger supply of gas than ever before.

In Kansas matters are different. Practically in every place where there was much hope for obtaining gas, drilling was prosecuted with zeal. The new field to the southwest of Chanute has been pretty well circumscribed on three sides, and a large amount of gas was obtained by various private individuals and small companies. The Kansas Natural Company also was alert and got hold of some exceedingly valuable territory.

To the northeast of Chanute, and farther north by way of Humboldt toward Iola many wells were drilled, the great majority of which were successful, some of them developing from 10,000,000 to 15,000,000 cu.ft. per well. In the vicinity of Altoona also some excellent gas wells were

obtained by the Altoona Portland Cement Company and private individuals. It is claimed by the cement company that it has one well with a capacity of 20,000,000 cu. ft. per day. Its territory virtually, however, is a southern extension of the pool southwest of Chanute already mentioned.

Farther north, in the vicinity of Colony, a small field of shallow gas was opened with eight or nine wells ranging from 750,000 to 1,500,000 cu.ft. each. This, by the way, is an interesting locality, particularly on account of its non-productiveness. Colony is situated on the very summit of the most pronounced anticlinal ridge in the entire Kansas gasfield, and yet every deep well drilled in that vicinity has been a failure. The shallow field just mentioned lies east of town from two to three miles, and forms a narrow band trending a little northeast and southwest. The wells here are from 500 to 600 ft. deep, while the horizon supplying gas at Iola and elsewhere so abundantly in the State should be found at from 900 to 1000 feet.

The small gasfield developed more than a year ago in the vicinity of Rantoul near Ottawa is about the same as it was then, and in many respects is about like the Colony pool just mentioned. Here also may be named the Bonner Springs and the Eudora pools, each of which furnishes a small amount of gas, but neither of which thus far has been able to supply large quantities. The development during the past year lends but little encouragement to hope for great production from any of those light fields.

Farther west in the vicinity of Elmdale, Chase county, a small gasfield was developed more than a year ago, which remains about the same as it was when first developed. To the south and southwest from this place a number of small wells were found, particularly in the vicinity of Augusta and Arkansas City. It is to be hoped that future prospecting may increase the supply. Likewise, a little northeast of Elmdale in the same range 12 or 14 miles southwest of Council Grove light wells were found, but none of any particular promise for a strong gasfield.

Ohio. (By J. A. Bownocker.)—No marked advance was made in this industry during 1908. As for some time past the only large fields have been in the central part of the State. These are usually known as the Sugar Grove and Licking-Knox county fields. The former is the older and its limits have been definitely established. Many of the earlier wells have been abandoned and the production of nearly all notably reduced. Drilling is still active in the Licking-Knox territory but no large extensions are confidently looked for. The quantity of gas which these two fields have yielded is almost incredible. Scores of cities

and towns are supplied, and not a few use it to a greater or less extent for manufacturing purposes. Occasionally it is used in brick and tile mills, where only the more common fuels should find a place. This use of the finest fuel known is simply an outrage. A few small reservoirs of gas are in use in the eastern and southeastern parts of the State. The best of these are in Monroe and Jefferson counties. Their market is local and the supply apparently not large. Probably the production of natural gas in Ohio is now at the zenith. Certain it is that unless new fields are found or existing ones notably extended the decrease will be rapid.

West Virginia. (By I. C. White.)—Many good gas wells have been developed as far south as Dunlow, in Wayne county, and eastward into Kanawha and Boone. The Roane county field began its history about eight years ago, when a prolific gas field was developed in the Big Injun sand at the southern end of the county. When the state geologist visited this field in 1902, he predicted that an oil field would be found down the slope of the rock toward Walton, and stated that in Vol. I (A), page 469, published July 1, 1904. Guided probably by this information, a year or so later the first oil well was found in the Big Injun sand between the gas wells and Walton. Since 1905 the Roane county field has gradually extended both northeast and southwest until now its only rival is the Lincoln development. The gas field has also extended southward into Kanawha and Clay counties until it promises to prove one of the largest in the State. The initial rock pressure of the gas wells in this pool was about 600 lb., and in spite of large consumption, it is reported as only slightly reduced. A new gas field was opened in 1908 near Spencer, in Roane county, where a well in the Berea Grit is recorded with a daily capacity of 5,000,000 cu.ft., a feature of the year which was duplicated in Upshur county, near Buckhannon. The latter town, as well as Spencer, has been supplied with gas for many years from fields 10 to 15 miles distant.

Lewis county still maintains its lead as the greatest gas county of West Virginia, the Hope Company having one well shut in there with a capacity of 36,000,000 cu.ft., as determined by its open pressure when struck several years ago. This county has produced only a small quantity of oil as yet compared to other counties of West Virginia, but it will probably surpass any of them in gas production, inasmuch as a large and broad anticlinal fold (Chestnut Ridge) passes almost through the center of its area, thus giving the necessary relief for the accumulation of an enormous gas pool which also extends northeastward over a large portion of Harrison county, making it nearly equal to Lewis in gas production.

NICKEL AND COBALT.

The production of nickel from domestic sources in 1908 was insignificant, the North American Lead Company, of Fredericktown, Mo., being the only producer. For that reason we are unable to report the amount of the production, but we estimate it at 500,000 lb. This company produces electrolytic nickel and cobalt oxide. Its appearance in the market for the latter substance in 1907 led to the great decline in the market price for it. Outside of Missouri, developments in nickel mining were made by W. S. Adams, at Webster, N. C., who began, early in 1909, the construction of a 20-ton smelting furnace. The American Nickel Compony did some work at its mines near Boyer, Churchill county, Nev. Some attention was directed also to the occurrence of nickel in Virginia, which was described in a paper by Prof. T. L. Watson, at the Toronto meeting of the American Institute of Mining Engineers, July, 1907.

The Orford Copper Company, Balbach Smelting and Refining Company, and North American Lead Company are the only producers of metallic nickel in the United States. We are unable to report precise statistics of the production, but on the basis of the importations of nickel in matte into the United States and other data, we estimate the production of the metal in this country at 14,000,000 lb. in 1908, against 17,500,000 in 1907 and 14,300,000 in 1906.

Market.—The market for nickel was unchanged in 1908, the control of the industry being so concentrated in the hands of the International Nickel Company that it can practically demand what it pleases. The schedule of prices that prevailed in 1908 is given in the accompanying

PRICES OF NICKEL FOR RETAIL PURPOSES IN 1908.
(In cents per pound.)

Not less than	Shot.	Grain.	Electrolytic
2000 lb	50	52	55
1500 lb	51	53	56
1000 lb	52	54	57
500 lb	55	57	60
100 lb	60	62	65
50 lb	65	67	70

table. Nickel that is disposed of to the producers of nickel steel under contract realizes somewhat less than the minimum price for shot nickel.

Cobalt oxide fetched $2.50 per lb. throughout 1907. In March, 1908, a violent rate war occurred between the two producers and the price was cut at frequent intervals until on April 1, it was $1.45 per lb. Later on sales were made at $1.40 per lb. The production of cobalt oxide in the United States in 1908 was about 100,000 pounds.

The nickel industry suffered in 1908 along with nearly every branch of metallurgical production. The International Nickel Company, which practically controls the nickel business in the United States, and also a large part of the foreign business, in its report for the year ended March 31, 1908, stated that its efforts to introduce nickel into new commercial arts had been vigorously continued. Owing, however, to the generally adverse conditions of industry, its own business suffered from the general depression. The net earnings of the constituent companies in America amounted to $2,434,952, from which is to be deducted $149,583 for general expenses, taxes, etc.; $215,975 for depreciation of plant; and $94,352 for depreciation of mines; making the actual net earnings $1,975,042.

UNITED STATES IMPORTS AND EXPORTS OF NICKEL AND COBALT.

(In pounds, and tons of 2240 lb.)

Year.	Imports.							Exports.	
	Nickel Ore and Matte.		Nickel Alloys. (a)		Nickel Mnfrs.	Cobalt Oxide.		Nickel. (b)	
	Long Tons.	Value.	Pounds.	Value.	Value.	Pounds.	Value.	Pounds.	Value.
1897..	12,420	$781,483	(c)	.....	.....	24,771	$34,773	4,255,558	$997,391
1898..	26,826	1,534,262	(c)	.....	.....	33,731	49,245	5,657,518	1,350,609
1899..	19,857	1,216,253	(c)	.....	.....	46,791	68,847	5,004,377	1,151,923
1900..	25,670	1,183,884	455,188	$139,786	.....	54,073	88,651	5,869,906	1,382,727
1901..	52,111	1,637,166	635,697	209,956	$2,498	71,969	134,208	5,809,355	1,521,271
1902..	14,817	1,156,372	752,630	251,149	30,128	79,984	151,115	3,228,807	925,579
1903..	15,936	1,285,935	521,344	170,670	37,284	73,350	145,264	2,414,499	703,550
1904..	8,548	915,470	589,555	203,071	2,950	42,352	86,925	7,519,206	2,130,933
1905..	13,451	1,626,920	941,966	331,920	3,291	70,048	139,377	9,550,918	2,894,700
1906..	15,156	1,816,631	210,000	77,373	8,963	41,084	83,167	10,620,410	3,493,643
1907..	(d)16,888	2,153,873	180,025	80,994	9,159	42,794	73,028	8,772,578	2,845,663
1908..	(e)16,322	2,396,217	241,868	91,388	10'010	1,550	3,095	9,770,248	3,297,988

(a) Includes nickel, nickel oxide, and alloy of any kind in which nickel is the material of chief value, in ingots, bars and sheets (b) Comprises domestic nickel, nickel oxide and matte. (c) Not separately enumerated; included in "Nickel Ore and Matte." (d) Contained 18,418,305 lb nickel; not reported previous to 1907. (e) Contained 16,586,423 lb. nickel.

Nevada.—A deposit of ore similar in some respects to those at Sudbury, Ont., has been the subject of considerable interest in southeast Nevada. This deposit occurs in the Key West mine, situated at the west end of the Virgin River mountains, about five miles west of the Utah line. The district is reached from Moapa, on the San Pedro, Los Angeles & Salt Lake railroad, by a 35-mile stage ride, passing through Bunkerville. The ore is confined to rounded masses of diabase intruded into crystalline schists, the former being very much like the basic rock

of Sudbury, Ontario. The diabase weathers rapidly, forming a greenish black, powdery mass. Chalcopyrite is the prevailing mineral, the ore averaging about 3.5 per cent. copper, 2.5 per cent. nickel, 1 to 3 oz. silver, and 0.25 to 0.30 oz. platinum per ton. It does not contain gold. The first work on the property was an incline dipping 12 deg. into the outcrop of one of the lenses; a second incline, at right angles to the first, and starting about 150 ft. from it, was sunk at a dip of 30 deg., developing a body of ore which varied greatly as to outline and metal contents. At 300 ft. from the first incline a shaft was sunk 110 ft., encountering a mass of ore which was cut off by what seemed to be a fault but proved afterward to be only the smooth exterior of the mass. A second shaft was sunk 150 ft. from the first and encountered two additional lenses. A long crosscut was then driven to get under the first shaft, but after passing through the first lens it found nothing but the undisturbed crystalline schists. There were two reasons why the property was not further developed at that time, viz., (1) the distance from transportation and the high cost of operating, and (2) the difficulty in disposing of a copper-nickel-platinum matte. The mine is now within a reasonable distance of the railroad, and it is rumored that a matting furnace is to be erected in the vicinity.

According to A. Maurice Thompson, in *Eng. and Min. Journ.*, July 1, 1908, deposits of this nature are of wide extent in the district, and, were it not for the distance from nickel-refining centers, a large output of ore could easily be made. The platinum seems to be uniformly distributed throughout, as every sample shows that metal. The country consists of mica schists and limestones, into which lenses of diabase have been injected. The lenses range from 10 to 50 ft. thick, and from 50 to 600 ft. long, and have a northeast-southwest strike. At one place, about 5000 ft. of development work has been done, reaching a maximum depth of 400 ft., and blocking out 150,000 tons of ore. An average of 25 samples taken throughout this development shows copper, 3 to 5 per cent.; nickel, 2.5 per cent.; and platinum, 0.3 oz. per ton.

Nickel in Foreign Countries.

Canada.—With the exception of the nickel contained in the ores shipped from the Cobalt district, the production of nickel in Canada is derived entirely from the well-known nickel-copper deposits of the Sudbury district. Previous to 1906 the output had been increasing steadily for a number of years. During the last three years, however, the production has not varied very greatly. About 815 tons less matte was shipped in 1908 than in 1907. The nickel contents were also somewhat lower in 1908, averaging about 45.1 per cent. as compared with

48.1 per cent. in 1907. On the other hand the copper contents were higher in 1908. Two companies are carrying on active operations: the Mond Nickel Company, at Victoria Mines, and the Canadian Copper Company, at Copper Cliff. The ore is first roasted and then smelted to a bessemer matte containing from 77 to 80 per cent. of the combined metals, copper and nickel, which is shipped to the United States and Great Britain for refining. The accompanying table gives the results of the operations on the nickel-copper deposits of Ontario in 1906, 1907 and 1908, in short tons. According to customs returns exports of nickel in matte, etc., were, for 12 months ending Dec. 31, as shown in an accompanying table. These figures do not include the nickel contents of the silver-cobalt ores from the Cobalt district, of which it is difficult to obtain satisfactory returns. The shippers of silver-cobalt ores receive little or no return for the nickel contents, although this metal forms an important constituent of the ore.

PRODUCTION, EXPORTS AND IMPORTS OF NICKEL IN CANADA. (a)

Year.	Production.		Exports.		Imports.
	Pounds. (b).	Value. (c)	Pounds. (d)	Value. (e)	
1900	7,090,227	$3,327,707	13,493,239	$1,040,498	$6,968
1901	9,189,047	4,594,523	9,537,558	958,365	12,029
1902	10,693,410	5,025,903	3,883,264	834,513	15,448
1903	12,505,510	5,002,204	9,032,554	878,159	26,177
1904	10,547,883	4,219,153	14,229,973	1,237,307	16,330
1905	18,876,315	7,550,526	11,970,557	1,185,056	19,076
1906	21,490,955	8,948,834	20,653,845	.....	(f)15,808
1907	21,189,793	9,535,407	19,376,335	.....	(g)
1908	19,143,111	8,231,538	19,419,893	.....	(g)

(a) Statistics for production and imports cover calendar years, and are taken from the Annual Reports of the Geological Survey of Canada. Figures for exports cover the fiscal years ending June 30, and are taken from the Statistical Year Book up to 1905 inclusive. Subsequent figures are for calendar years as reported by the Canadian Geological Survey. (b) Pounds metallic nickel contained in copper and nickel matte exported. (c) On the basis of refined nickel at New York, from the *Engineering and Mining Journal* average annual quotations. (d) Pounds of nickel contained in ore, matte or speiss. (e) Spot value, to the producer, of the exported material; the variety of stages at which the material is shipped, as well as the different periods of time covered, lead to the apparent discrepancy in value when it is known that practically the entire production is exported. (f) Anodes only. (g) Not reported.

NICKEL EXPORTS FROM CANADA.

	1906	1907	1908
To Great Britain	2,716,892	2,518,338	2,554,486
To United States	17,936,953	16,857,997	16,865,407
Total	20,653,845	19,376,335	19,419,893

(By Thos. W. Gibson.)—Silver and nickel dominated the mining industry in Ontario in 1908. The shipments of ore from Cobalt for the year amounted to 24,485 tons; consequently the average silver content of the ores shipped was 792 oz. per ton. This is an improvement

over 1907, when the average had fallen to 677 oz. as compared with 1013 oz. in 1906. The improvement is, in part, due to the great richness of some of the veins opened during the year, and in part to the fact

PRODUCTION OF SILVER FROM COBALT.

Year.	Production, Oz.	Value.
1904	206,875	$111,887
1905	2,451,356	1,360,503
1906	5,401,766	3,667,551
1907	10,023,311	6,155,391
1908	19,400,640	9,115,818
Total	37,483,948	$20,411,150

ONTARIO NICKEL STATISTICS.

(In tons of 2000 lb.)

Schedule.	1902	1903	1904	1905	1906	1907	1908
Ore raised	269,538	152,940	203,388	277,766	343,814	351,916	409,551
Ore smelted	233,388	220,937	102,844	251,421	340,059	359,076	360,180
Per cent. nickel	2.55	3.16	4.58	(b)	(b)	(b)	(b)
Per cent. copper	1.78	1.81	2.41	(b)	(b)	(b)	(b)
Ordinary matte	24,691	30,416	19,123	17,388	20,364	22,041	21,197
Bessemerized	13,332	14,419	6,926				
Nickel content	5,945	6,998	4,743	9,438	10,745	10,602	9,572
Copper content	4,066	4,005	2,163	4,386	5,265	7,003	7,503
Value of nickel (a)	$2,210,961	$2,499,068	$1,516,747	$4,019,814	$4,629,011	$3,291,355	$2,930,989
Value of copper (a)	616,763	583,646	297,126				
Wages paid	835,050	746,147	570,901	(b)	$1,117,420	$1,278,694	$1,286,265
Men employed	1,445	1,277	1,063	(b)	1,417	1,660	1,690

Note.—The quantities reported in 1902 and 1903 under "bessemerised matte" include both bessemerised matte and high-grade matte, the former being the product of the Mond Nickel Company's works and the latter of the Ontario Smelting Works, which re-treat the low-grade matte produced by the Canadian Copper Company. (a) Value based on nickel and copper in matte and not on refined metals. (b) Not available.

that the shipments in 1908 included about 480 tons of concentrates, the product of the mills which several of the companies have erected to treat their low-grade ores.

The sensation of the camp in 1908 was furnished by the Crown Reserve mine. A portion of the bed of Kerr lake, containing about 23 acres, had not been staked out as a mining claim notwithstanding it was in juxtaposition to such mines as the Drummond, Kerr Lake, Lawson and Foster. In 1906 the Ontario government offered the parcel for sale to the highest bidder. The best offer received was $52,000, which was not considered high enough. Offers were invited a second time with the condition added that the purchaser should pay 10 per cent. royalty on the value of the output at the mine. The bids this time went up to $178,500. The buyers (a syndicate of Montreal brokers) formed a company, and for a time the shares sold for 15 to 20c. A diamond drill discovered a vein which for size and richness in native silver is probably not surpassed in the camp. Shares are now worth

about $2.70, and the government's decision to reserve a royalty will undoubtedly put a large sum in the treasury of the Province.

The leading mines of 1907 continued their production in 1908, and most of them increased their output. Among those were Nipissing, La Rose, Coniagas, O'Brien, Kerr Lake, Trethewey, Silver Queen, Temiskaming, Temiskaming & Hudson Bay, Townsite, Right-of-Way and others. Crown Reserve was the chief producer added during the year. Temiskaming came well to the front, and is of interest from the fact that its leading vein is in the Keewatin formation, which at Cobalt is less prolific in silver-bearing ore than the conglomerate.

Most of the ore is still shipped out of the camp for treatment, the low-grade or "screenings" going principally to Denver, Colo., where it has been found suitable for mixing with ores from certain of the Western mines. Low-grade ore in Cobalt contains 100 to 150 oz. per ton. The high-grade ore which carries up to 7000 and 8000 oz. per ton is treated principally at the Canadian Copper Company's smeltery at Copper Cliff, and the Deloro Mining and Reduction Company's works at Deloro, both of these plants being in Ontario. The latter is owned chiefly by the proprietors of the O'Brien mine. The Coniagas smeltery at Thorald on the Welland canal, went into commission during 1908, and operated on the product of the Coniagas mine.

There were no labor troubles at Cobalt in 1908, but the declining value of silver undoubtedly had a tendency to lessen production. The cost of producing silver is put by some of the leading Cobalt mines at 9 to 10c. per oz., so that even at 48 or 49c. there is a good margin of profit. That the mines of Cobalt yielded nearly 100 per cent. more silver than those of any one State in the United States even at low silver prices is proof of the remarkable character of the deposits.

Extensions of the Cobalt silver field continue to be eagerly sought for and several have been found. The chief of those are: (1) South Lorrain; (2) Montreal river; (3) Everett and Miller lakes; (4) Gowganda lake.

South Lorrain is a patch of unsurveyed territory on the west shore of lake Temiskaming, and south of the township of South Lorrain proper. The formations are similar to those in the vicinity of Cobalt. Most of the promising veins have been found in the Keewatin but good silver ore has also been obtained from veins in the diabase. The chief properties in South Lorrain are the Keeley (H.R. 19), and the Haileybury Silver Mining Company.

The Montreal river runs in a southeast course for many miles before falling into lake Temiskaming, and about 40 miles northwest of Cobalt, in James township and vicinity, some good finds of silver have been

made. The geology is similar to that of Cobalt except that the diabase is more prevalent, and in this the silver-bearing veins chiefly occur. Numerous claims have been staked in James, Smyth and Tudhope townships and also on Hubert, Silver, Bloom and other lakes. The Otisse, Downey, Gifford, Currie and other claims in the district and the Darby and White groups at Maple Mountain have promising showings.

Miller and Everett lakes lie farther west in the Temagami forest reserve, and a number of excellent discoveries were made here and also at Gowganda lake, still farther west, whence some very fine samples of native silver were brought during the year, resembling the bonanza finds made at Cobalt in 1903 and 1904. Gowganda claims most attention at the close of the year as the new silver field. "Snowshoe" staking is actively going on at present, and the coming of spring will undoubtedly see a big rush. Careful observers, comparing conditions with those at Cobalt, doubt whether the veins in any of these fields will prove so rich or permanent.

The cobalt-oxide industry suffered from the excessive supply of cobalt ore thrown on the market as a by-product from silver smelting. The world's consumption of oxide annually falls far short of the production from the mines of Cobalt, and consequently the price of oxide fell from $2.50 to $1.40 per lb. The only relief is in a widely extended use for cobalt, which is, at present, chiefly employed in making the color known as cobalt-blue.

The output of the nickel-copper mines of the Sudbury district for 1908 was slightly less than in 1907. The general slackening off in the iron and steel trades has lessened the demand for nickel, and during the last quarter of 1908 mining and smelting operations were considerably curtailed. The nickel production of 1907 was 10,972 tons; for 1908 it was 10,175 tons. The copper production was slightly in excess of that for 1907, say 7561 tons. The leading producers continued to be the Canadian Copper Company and the Mond Nickel Company. The former's works at Copper Cliff were completed and well equipped. Ore was raised mainly from the Creighton mine, which is high in nickel and from the Crean Hill, which is rich in copper. The Mond company's plant is at Victoria Mines, and its orebodies are at the latter place and in the township of Garson. A new concern, the Dominion Nickel Copper Company, has begun work on the northern range, where it controls a number of important deposits some of which, notably the Whistle, rival in size any yet opened on the southern range. The construction of the Moose Mountain branch of the Canadian Northern railway has provided the necessary shipping facilities, and the com-

pany proposes to develop water power on the Wahnapitae river for use in its mines. The parties principally interested are Canadian capitalists.

Norway.—The Evje nickel mines were sold to a company represented by Admiral Börresen and Sam. Eyde. These well-known mines are situated in the valley of Sötersdalen, about two hours' journey by rail from the town of Christiansand, about 5 km. from the station of Evje. In 1908 their output was 5781 tons of ore, of which 5493 tons were smelted, giving 200 tons of matte containing 81 tons of nickel and 53 tons of copper. It is now intended to modernize and enlarge the plant and increase the production.

New Caledonia.—The production of this island is given in the accompanying table. At present all of the nickel ore is exported to France, Great Britain and Germany for smelting, but plans for smelting on the island are again under consideration. Cargoes of nickel ore are now regularly imported from New Caledonia into the Clyde for the Glasgow Nickel Company, whose works are at Kirkintilloch, on the banks of the Forth and Clyde Canal, which affords cheap transportation. About 300 men are employed at Kirkintilloch. Practically all the ore used is imported from New Caledonia, and is brought mostly by French sailing vessels. During the 12 months ended June 30, 1908, 50,604 tons of nickel ore were landed, as compared with 42,277 tons in the previous year.

SHIPMENTS OF NICKEL AND COBALT ORES FROM NEW CALEDONIA. (a)
(In metric tons.)

	1899	1900	1901	1902	1903	1904	1905	1906	1907	1908
Nickel ore	103,908	100,319	133,676	129,653	77,360	98,655	125,289	130,688	101,708	120,028
Cobalt ore	3,200	2,400	3,110	7,512	8,292	8,961	7,919	2,487	3,943	3,405

(a) Reported by *Le Bulletin du Commerce*, Noumea.

Monel Metal.

During the last three years the International Nickel Company has been investigating the properties of a new alloy of nickel and copper which is now coming into commercial use under the name of Monel metal. This consists of nickel and copper in the proportion of three parts of nickel to one part of copper, this being the natural proportion in which these metals exist in the deposits in the Sudbury district. Throughout its entire manufacture this alloy acts as a single metal. It is sold in New York at 2c. above the market price for copper. A typical analysis of Monel metal is as follows: Nickel, 68 to 72 per cent.; iron,

0.5 to 1.5 per cent.; sulphur, 0.014 per cent.; carbon, 0.073 to 0.15 per cent.; copper to balance. This alloy is silver-white and takes a brilliant finish, which it retains indefinitely. On prolonged exposure the surface assumes a grayish cast, which may be easily removed with a polishing cloth. In the rolled sheets, the surface assumes on heating a coating of oxide which has great resistance to acids. Samples of the rolled metal show no loss in 56 days' test in pumping 40 deg. sulphuric acid.

Monel metal melts at 1350 deg. C. It can be rolled perfectly from 900 to 1200 deg., and its annealing temperature is above 875 deg. It can be finished hard or soft, like sheet copper. Its specific gravity, as cast, is from 8.86 to 8.87, and when rolled from 8.94 to 8.95. Cast Monel metal is made in two grades, with qualities that compare very favorably with the specifications for carbon-steel castings. In the rolled and annealed condition Monel metal is stronger than nickel steel.

During 1908 about 300,000 sq.ft. of Monel metal sheets were used to roof the Pennsylvania tunnel station in New York City. The manufacturers, who have tested this material for roofing, state that they find it as easily worked as copper. The sheets are silvery white, exceedingly flexible and malleable, and are handled in exactly the same way as sheet copper. The rolled metal has been drawn into wire of all sizes down to 0.004 in. This latter is as soft and pliable as a silk thread. In the form of wire exhaustive tests have been made of the metal for window screens, bed springs, filter cloths and other purposes where its incorrodibility makes it especially effective. In small articles, such as tacks, bolts, screws, angle braces and such stamped work as door and window trimmings, hinges, gas and electric fittings, and other small articles too numerous to mention, it is rapidly finding its way into the market. The metal works on the lathe as well as soft steel. It has been spun and pressed into hollow-ware, kitchen utensils, watch cases, finger bowls, etc., with the same ease as copper or silver. One of the most satisfactory uses of Monel metal is in seamless tubes for condensers and boilers for automobiles and motor boats. The metal draws perfectly into a smooth tube and its high elastic limit and tensile strength, combined with its incorrodibility, are of especial value in light machinery.

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NEW PATENTS.

Complex Ore Treatment.—Process for treating complex cobalt ores and for refining cobalt from nickel, arsenical and silver-bearing ores. Ernest E. Armstrong, Niagara Falls, N. Y. (U. S. No. 881,527 of 1908.)

Metal Extraction.—Method of extracting metallic ores and matte. Emil Günther and Rudolf Franke, Eisleben, Germany. (U. S. No. 875,259 of 1907.)

Ore Treatment.—Process of recovering nickel from silicious ores. Adolphe Chalas, Philadelphia, Penn. (U. S. No. 887,735 of 1908.)

Ore Treatment.—Process of treating metallic ores or mattes. Emil Günther and Rudolf Franke, Eisleben, Germany. (U. S. No. 879,633 of 1908.)

Purifying Precipitates.—Process of purifying precipitates. Herbert H. Dow, Midland, Mich., and Walter S. Gates, Worthington, Ontario, Canada, assignors to Ontario Nickel Company, Ltd., Worthington, Canada, a corporation of Canada. (U. S. No. 883,792 of 1908.)

Refining Process.—Electrolytic process for refining nickel. John N. Pring, New York, N. Y., assignor to Elmer A. Sperry, Brooklyn, N. Y. (U. S. No. 874,864 of 1907.)

Separation from Matte.—Art of separating metals from matte. Albion J. Wadhams, Riverton, N. J., and Robert C. Stanley, New Brighton, N. Y., assignors to International Nickel Company, Bayonne, N. J., a corporation of New Jersey. (U. S. No. 900,452 and 900,453 of 1908.)

Separation of Nickel from Matte.—Process for separating the metals contained in copper-nickel matte. James M. Neil, Toronto, Ontario, Canada, assignor to Frederic C. Norris, trustee, Detroit, Mich. (U. S. No. 882,075 of 1908.)

Separation from Speiss.—Art of separating metals from speiss. Albion J. Wadhams, Riverton, N. J., and Robert C. Stanley, New Brighton, N. Y., assignors to International Nickel Company, Bayonne, N. J., a corporation of New Jersey. (U. S. No. 900,454 of 1908.)

PETROLEUM.

By Edwin Higgins.

The production of petroleum in the United States in 1908 was 184,711,413 bbl., which is the largest in the history of the industry. This figure compares with 164,347,930 bbl. in 1907 and 131,771,505 bbl. in 1906. The exports of crude petroleum, naphtha, illuminating oils, lubricating oils and paraffin, and residuum during 1908 also exceeded those of any previous year.

PRODUCTION OF CRUDE PETROLEUM IN THE UNITED STATES.

(In barrels of 42 gal.)

Field.	1902	1903	1904	1905	1906	1907	1908
California (a)	13,973,500	24,337,828	28,476,025	35,671,000	30,538,000	40,085,000	45,000,000
Colorado	396,901	483,925	(b) 501,763	(e) 550,000	600,000	400,000	411,836
Gulf Texas	18,083,658	17,955,572	21,672,111	30,354,263	12,666,000	12,350,000	11,220,000
Gulf Louisiana	548,617	917,771	6,611,419	9,672,015	7,100,000	4,620,000	5,700,000
Illinois					4,900,000	24,540,024	38,844,899
Lima Indiana	7,535,561	9,177,122	10,744,849	22,102,106	25,680,000	8,030,000	7,287,000
Lima Ohio	15,877,730	14,893,853	13,350,060				
Mid-Continental (c)	359,123	1,157,110	5,617,527	12,000,000	21,929,905	47,556,906	50,741,678
Ken.-Tennessee	(f)	(f)	998,284	(e) 1,200,000	1,000,000	1,250,000	1,250,000
Pennsylvania (d)	32,018,787	29,897,815	(b)30,410,183	28,324,324	27,345,600	25,500,000	24 240,000
Wyoming	6,253	8,960	11,542	(e) 12,500	13,000	13,000	(e)13,000
Others	957	3,000	2,572	(e) 3,000	4,000	3,000	(e)3,000
Total	87,801,087	98,832,956	118,396,335	139,889,210	131,771,505	164,347,930	184,711,413

(a) Reported by the California Producers' Association, except the statistics for 1907 and 1908, which are of our own co'lection. (b) Statistics of the U. S. Geological Survey. (c) Kansas and Oklahoma. (d) Pennsylvania, New York, West Virginia, Eastern Ohio, and, until 1904, Kentucky and Tennessee. (e) Estimated. (f) Included in Pennsylvania.

EXPORTS OF MINERAL OILS FROM THE UNITED STATES. (In gallons.)

(1=1000 in quantities and values.) (a)

Year.	Crude Petroleum.		Naphthas.		Illuminating.		Lubricating and Paraffin		Residuum.		Totals.	
1897	121,864	$5,044	13,704	$1,020	804,446	$46,876	52,659	$6,732	(b)12,247	$ 335	1,004,920	$60,007
1898	120,436	5,016	17,258	1,071	764,823	38,895	65,526	7,626	(b)30,436	815	998,479	53,423
1899	117,690	5,958	18,210	1,597	733,382	49,172	71,116	8,658	(b)21,609	658	962,007	66,043
1900	133,161	7,341	18,570	1,681	739,163	54,693	71,211	9,933	(b)19,750	845	986,855	74,493
1901	127,008	6,038	21,685	1,742	827,479	53,491	75,306	10,260	(b)27,596	1,255	1,079,059	72,786
1902	145,234	6,331	19,683	1,393	773,801	49,079	82,200	10,872	(b)38,316	922	1,064,234	68,597
1903	126,512	6,782	12,973	1,519	691,837	51 356	95,622	12,690	(b) 9,753	282	936,697	72,629
1904	111,176	6,351	24,989	2,322	761,358	58,384	89,738	12,389	34 904	1,174	1,022,165	80,620
1905	126,185	6,086	28,420	2,215	881,450	54,901	113,730	14,312	70,728	2,128	1,220,513	79,641
1906	148,045	7,731	27,545	2,488	878,274	54,858	151,269	18,690	64,645	1,971	1,269,788	85,738
1907	126,306	6,334	34,625	3,676	905,924	59,635	152,029	19,210	75,775	2,528	1,294,660	91,383
1908	149,190	6,520	43,890	4,542	1,129,005	75,988	147,769	18,971	77,552	2,793	1,547,406	108,815

(a) In addition to the above, the following quantities of paraffin and paraffin wax were exported (1=1000): 1897, 136,069 lb. ($5,284); 1898, 166,317 lb. ($6,363); 1899, 181,861 lb. ($7,650); 1900, 157,108 lb. ($8,186); 1901, 151,695 lb. ($7,960); 1902, 175,268 lb. ($8,398); 1903, 204,120 lb. ($9,596); 1904, 174,582 lb. ($8,273); 1905, 160,836 lb. ($7,873); 1906, 173,504 lb. ($8,463); 1907, 207,504 lb. ($10,209); 1908, 141,667 lb. ($6,923). (b) Reported in barrels of 42 gallons.

Oklahoma headed the list of producers in 1908, with California and Illinois following in the order named. California made a large increase in production, and the market there was strong throughout the year. Over-production in Oklahoma was the cause of depressed market conditions in that district. There was a significant extension of the West Virginia oilfields. A decrease in production was recorded in the Lima field, while the Appalachian field, the production of which has steadily declined since 1904, showed a further decline. No new discoveries of significance were made during 1908 in Texas or Louisiana. In general, the market was strong, the top price recorded for oil of the Pennsylvania grade being $1.78 per barrel. The industry in the different producing fields is reviewed in detail in the following pages.

PETROLEUM OUTPUT OF THE WORLD. (a)
(In metric tons.)

	1902	1903	1904	1905	1906	1907	1908
United States.....	11,906,000	13,402,000	16,055,000	18,999,000	17,862,000	22,287,985	25,054,931
Russia.....	10,445,536	9,759,214	10,058,968	7,505,637	8,167,934	8,435,708	7,631,800
Dutch East Indies...	732,000	662,767	955,957	1,062,224	1 186,907	2,200,000	2,348,000
Galicia.....	576,000	675,518	826,077	794,862	739,885	1,176,000	1,612,000
Rumania.....	310,000	384,303	497,000	614,870	887,000	1,129,097	1,147,727
India.....	209,000	352,848	475,869	581,519	564,470	587,000	661,000
Other countries.....	270,000	250,000	250,000	350,000	367,000	(b) 590,000	(c) 880,000
Total	24,448,536	25,486,650	29,118,871	29,878,112	29,775,196	36,405,790	39,335,458

(a) In the above table the statistics for the United States are computed from the production reported in barrels as given in the first table of this article. As a gallon of the crude petroleum found in the United States varies in weight from 6.41 to 7.83 lb., the oil in a barrel varies from 269.22 to 328.86 lb. The arithmetical mean of these figures is 299.04 lb., which figure has been used as a factor in converting the output stated in barrels into metric tons. This is not strictly correct, because in the period from 1900 to 1908 the proportion of petroleum production of various gravities has altered materially, especially because of the largely increased production in California, Texas and Louisiana. However, the method adopted is as near an approximation as can be made at this time. (b) Includes 294,000 tons from Mexico. (c) Includes 587,000 tons from Mexico.

Review of the Oilfields of the United States.

Appalachian. (By Harold C. George.)—The production of petroleum in the Appalachian oilfields has been steadily declining for years. In 1908 there was a further decrease. In addition to the natural falling off in production in the Allegheny field of New York and practically all of the oilfields of Pennsylvania, developments and operations were retarded for several months by the drought and the resulting lack of water. Large numbers of pumping wells stood idle for the last three or four months of 1908, and consequently where such conditions existed no new wells were drilled. It is only by the successful drilling operations in southeastern Ohio and West Virginia that the decrease in petroleum production in the Appalachian field has been gradual and slight rather than rapid and great.

West Virginia is the only district in the Appalachian field that holds out any great possibilities of increasing the supply of high-grade petro-

leum; in the southwestern part of that State there is a large unprospected area. This region is now attracting the attention of prospectors. In both Jackson and Lincoln counties some prospecting was done during 1908, but as yet no large wells have been drilled. In Lincoln county nine wells were completed during April and May. These wells averaged 40 bbl. a day. In November, 12 wells were completed which averaged 35 bbl. a day. Brooke county continues to lead all other districts in "gushers." The Follansbee pool in Brooke county and the Holidays

PRODUCTION OF WELLS DRILLED IN THE APPALACHIAN OILFIELD DURING 1907 AND 1908.

Field.	Number of Wells Drilled.		Daily Production in Barrels.		Daily Production per Well Drilled in Barrels.		Per Cent. of Dryholes.	
	1907	1908	1907	1908	1907	1908	1907	1908
Allegheny, New York	575	493	1,114	880	1.9	1.8	16.0	13.4
Pennsylvania	3,611	7,748	12,176	9,532	3.3	2.5	21.0	19.0
West Virginia	1,320	1,329	21,300	27,304	16.1	20.6	38.0	32.5
Southeastern Ohio	1,335	1,344	6,793	13,798	5.9	10.3	39.5	39.3
Kentucky and Tennessee	212	205	2,006	2,519	9.4	12.3	32.0	33.6
Total	7,053	7,119	43,389	54,033	6.1	7.6	27.0	25.0

Cove pool in Hancock county attracted much attention during 1908, but it is generally believed that these fields are now on the decline. This section of the State remained unprospected and undeveloped for years, but finally, after much drilling, oil was discovered in that section of West Virginia which lies across the Ohio river from Steubenville, Ohio. Hancock county has long been noted for its large oil pools, but it is only recently that Brooke county has become a factor in the petroleum production of West Virginia. In Roane county operations continued actively and with profitable results. The best wells are found in the "Big Injun" sand.

The operations in southeastern Ohio attracted some attention owing to the development of the Mingo pool, but like the Follansbee pool across

MONTHLY AND YEARLY AVERAGE PRICE OF PIPE-LINE CERTIFICATES PER BARREL OF CRUDE PETROLEUM AT THE WELLS IN THE APPALACHIAN FIELD.

Year.	Jan.	Feb.	March.	April.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Average
1899	$1.17	$1.15	$1.13	$1.13	$1.13	$1.13½	$1.22½	$1.27½	$1.44½	$1.50½	$1.57½	$1.65½	$1.29½
1900	1.66½	1.68	1.68	1.55	1.30½	1.25½	1.25½	1.25½	1.23	1.10½	1.06½	1.09½	1.35¼
1901	1.09½	1.25	1.29	1.20½	1.07½	1.05	1.13½	1.25	1.25½	1.30	1.30	1.21	1.21
1902	1.15	1.15	1.15	1.17½	1.20	1.20	1.22	1.22	1.22	1.28	1.38	1.49	1.24
1903	1.52	1.50	1.50	1.51½	1.51½	1.50	1.53	1.56	1.56	1.69½	1.79½	1.88½	1.59
1904	1.85	1.82½	1.73	1.65	1.59½	1.57	1.53½	1.50	1.54½	1.56	1.58	1.55	1.62½
1905	1.43½	1.39	1.37½	1.32½	1.28	1.27	1.27	1.27	1.39	1.58½	1.59½	1.58	1.31½
1906	1.58	1.58	1.58	1.60	1.64	1.64	1.63½	1.58½	1.58	1.58	1.58	1.58	1.5969
1907	1.58	1.63	1.68	1.78	1.78	1.78	1.78	1.78	1.78	1.78	1.78	1.78	1.7425
1908	1.78	1.78	1.78	1.78	1.78	1.78	1.78	1.78	1.78	1.78	1.78	1.78	1.78

the Ohio river in Brooke county, W. Va., of which this pool is a part, the Mingo field also is declining in production.

All recent additions to the areas producing oil of Pennsylvania grade have been too small to have more than a temporary effect in checking the waning production. There was no change in the price of the high-grade petroleum during 1908. The price remained at $1.78 per bbl. throughout the year; in 1907 the average price was $1.74 per bbl.

West Virginia. (By I. C. White.)—The new developments in petroleum during 1908 were confined principally to the counties of Brooke, Pleasants, Ritchie, Roane, Kanawha, Lincoln and Cabell, the most active regions being in Roane and Lincoln. Both of these fields were developed very slowly. The Lincoln field is principally in the Berea Grit. Since 1907 this field has extended in most every direction, and a new (Big Injun) producing horizon has been developed farther west along the Guyandotte river region. The Lincoln field, which had such a modest beginning more than four years ago, is today giving promise of proving one of the largest and richest oil and gas pools of the State. None of the oil wells has proved large, ranging in size from only 20 to 50 bbl., but as the wells from this horizon are noted for their staying qualities, the field promises finally to surpass in daily production any of the existing pools of the State.

Since 1905, the Roane county field has gradually expanded both northeast and southwest until now its only rival is the Lincoln development. A new oil pool was found in Harrison county, near Shinnston in the latter half of 1908, but as yet only one well has been drilled. It is in the "50-ft. Sand," and although it produces only 30 to 40 bbl. daily, it may lead to important developments, since there are no oil wells nearer than 8 to 10 miles. Some new spurs from old pools were opened in Ritchie, Pleasants, Doddridge, Wetzel, Marion, and Monongalia counties during the year. The Brooke county field was very active during 1908, and several attempts were made to extend it southward into Ohio county, but as yet without success, although a small showing of oil has been found in the Big Injun sand near Wheeling. The Brooke and Hancock oilfields occur in the Berea Grit, and hence the showing of oil near Wheeling can have no connection with the fields in the prolific pools to the north. The production of oil in West Virginia for 1908 was somewhere near 10,000,000 barrels.

California.—The production of oil in California in 1908 was 45,000,000 bbl. In 1907 the yield was 39,748,375 bbl., valued at $14,699,956 or about 37c. per bbl. In 1908 the average value at the wells was between 48 and 50c. per bbl., and at the end of the year it was perhaps higher than 60c. per bbl. All the old contracts were "worked up" and

the new ones being made vary from 50 to 63c. per bbl. Light oil from Coalinga sold as high as 87c., and in the Santa Maria field prices at the end of the year advanced to 70 cents.

In 1908 some of the companies nearly doubled the yield of the previous year. Much prospecting was done in undeveloped territory and new wells are being sunk in proved areas. The Kern river field heads the list in amount of daily production, followed in order by Coalinga, the Lompoc and Santa Maria, and the Los Angeles-Salt Lake fields. In total output Coalinga forged to the front in the last two months of the year, leaving Kern river second, with Santa Maria third in importance. In the Kern river field there are 1900 producing wells which average 35 bbl. a day per well. In Coalinga there are 385 wells yielding an average of 80 bbl. a day per well. In the Santa Maria field the wells are averaging a daily product of 200 bbl. a day per well. In other fields the flow from the producing wells varies from 20 to 70 bbl. per day. In 1907 there were 2630 wells producing; the figures for 1908 are not yet available.

The increase in consumption of California fuel oil in 1908 was general and was from 20 to 25 per cent. over the amount in 1907, both for industrial and railroad uses, and the output is expected to keep pace with this. In one field alone—the Kern river—there was in 1908 an increase of 25 per cent. in the number of productive wells compared with 1907; while the increase is not so large in other fields, yet there is an apparent increase in all, with the possible exception, perhaps, of the Santa Maria. In that field a large amount of water is being pumped with the oil.

The pipe line 280 miles long, from the Kern river field to Port Costa, at tidewater, was completed in 1908. It is an 8-in. rifled pipe line owned equally by the Associated Oil Company and the Southern Pacific railroad. Between Bakersfield and Port Costa there are 12 pumping stations, for both water and oil, on this line. The "rifling" of the pipe gives it a much greater carrying capacity for heavy fuel oil than an ordinary smooth-bore pipe has. This is the second pipe line from Kern county to tidewater. The Associated Oil Company intends building another pipe line from Sunset through the Midway, the McKittrick, and the Devil's Den fields to Coalinga. The fields in Santa Barbara county have their pipe lines to the ocean, and there is one from Fresno county to Monterey bay. The Japanese are large consumers of California fuel oils and have extensive contracts, mainly with companies owning wells in the Santa Maria field.

The various oilfields are gradually being enlarged as new wells are sunk and developments progress. All have now good transporta-

tion facilities to tidewater, and large refineries are numerous. A great amount of prospecting is being done in new sections which promise good results, and much money is thus being expended in finding new oil-yielding areas. Meantime, in the older fields more extensive operations are being carried on, and deeper explorations are being conducted.

Colorado and Utah.—The Florence and Boulder fields of Colorado produced steadily during 1908. The total production was 411,836 bbl. The new Rangely field in the northwestern part of Colorado attracted much attention. Early in June the Star-Crescent Company completed its Downey well, near Virgin City, Utah. The depth obtained was 517 ft., at which point some oil showed. About the same time the Wood No. 1 well, of the Virgin River Oil and Development Company, was brought in at a depth of 725 ft. It is estimated that this well will yield 40 bbl. per day of light gravity green oil.

(By A. P. Rogers.)—A new oilfield has been developed in the southeastern part of Utah, a section heretofore practically inaccessible. The district lies to the east of the Colorado river, where it cuts across the State, and along the course of the San Juan before the deeper portions of the cañon have been reached, that is to say, between the settlement of Bluff City on the east and Moonlight on the west, and south of the Blue and Henry mountains.

The country is sandstone, of the Jura-Triassic and Cretaceous ages, with occasional igneous ranges breaking through it, north or south of the known oilfield. A series of anticlines and synclines have produced ideal conditions for oil deposits, and it is here all present prospecting is taking place. The nearest railroad station to Bluff City is at Dolores, Colo., on the Denver & Rio Grande Railway, 80 miles away. Ten miles west of Bluff City lies the Comb Wash, extending both north and south of the San Juan river for many miles; from its western side an anticlinal fold rises over 1000 ft. above the river at its highest point. This is the eastern boundary of the field as at present developed. It can be followed west from this point more than 60 miles across two synclines and succeeding anticlines until the formation finally disappears below the surface near the Colorado river. Through these anticlines the San Juan river has cut deep cañons, and at various places oilfields occur, especially from the lower sands close to the water. These cañons have exposed seven different oil-bearing sands in a depth of 1200 feet, varying in thickness from 3 to 60 ft., each one of which is sealed from the other by a stratum of hard blue limestone. The oilfields proper are about 65 miles from east to west and perhaps 75 miles north and south.

Four wells were sunk up to December, 1908, the deepest of which is 320 ft., and the shallowest 120 ft. Two of these struck oil and were gushers for a short time. The other two struck water in the upper sands, but as these were primarily sunk for assessment work, they are not classed as completed wells. The first and deepest well was sunk at the mouth of Grand gulch, encountering a fine grade of oil upon penetrating the fifth sand; this oil carried a higher percentage of volatile matter than any so far found in the west, and if it is obtained in quantity in other wells it will prove most valuable. No continued pumping was attempted, however, up to the close of 1908. The country has been located for miles around and large drilling machines, some with a capacity of 3000 ft., are being taken in to prospect the field in a systematic manner.

Illinois. (By H. Foster Bain.)—The oilfields of Illinois produced 38,844,899 bbl. in 1908. The detailed pipe-line runs and miscellaneous receipts of the Ohio Oil Company for each month of 1908 are given in an accompanying table. In addition to these figures, the following receipts of other active refineries may be noted: Pipe-line runs, 1,081,000 bbl.; other receipts, 464,712 bbl.; total, 1,545,712 barrels.

RECEIPTS OF THE OHIO OIL COMPANY, 1908.

Month.	Pipe-line Runs, Bbl.	Other Receipts, Bbl.	Total, Bbl.
January	2,497,354.21	98,433.07	3,595,787.28
February	2,464,914.17	236,157.20	2,701,071.37
March	2,591,911.33	191,963.25	2,783,874.58
April	3,089,417.23	439,709.81	3,529,127.04
May	3,084,815.94	1,015,263.40	4,100,079.34
June	2,965,786.03	947,741.70	3,913,527.73
July	2,597,977.26	291,096.39	2,871,073.65
August	2,690,931.05	1,059,941.36	3,750,872.41
September	2,555,871.53	744,119.98	3,299,991.51
October	2,582,561.09	395,818.90	2,978,379.99
November	2,356,385.48	318,352.72	2,674,738.20
December	2,512,704.66	212,473.49	2,725,178.15
Total	31,972,629.98	5,951,071.27	37,923,701.25
Daily Average	87,356.91	16,259.76	103,616.67

The bulk of the miscellaneous receipts consists of oil purchased in tanks when leases were transferred. No additional lines have been laid into the field though the Pure Oil Company has purchased a right of way, and the Tidewater line is to be completed by July 1, 1909. The Indian Refining Company, the Sun Oil Company, the Cornplanter Oil Company, the Central Oil Company, and the Robinson Oil Refining Company continue to ship in tank cars. As in past years production, except for a short time in May and June, ran ahead of the capacity of the pipe lines. For most of the season approximately 60 per cent.

of the oil offered was taken. The price remained fixed at 68c. per bbl. for all oil above 30 deg. B, and 60c. per bbl. for that below 30 deg. B. The great bulk of the oil was marketed at the higher rate. On Dec. 31, the gross stock of the Ohio Oil Company amounted to 25,323,612 barrels.

The principal development work of 1908 was in Crawford, Lawrence and Clark counties, the largest wells being found in the deepest sands. Within the year Coles county, after repeated failures, became a producer. Drilling was active in the early summer, but the severe drought crippled operations in the fall. On Jan. 1, 1908, it was estimated that 9772 wells had been put down. Of these 1260 were dry holes. In 1908, 3574 wells were drilled, with 555 dry holes and 77,413 bbl. of new production. The record by months, taken as heretofore from the *Oil City Derrick*, is given in an accompanying table.

WELLS DRILLED IN ILLINOIS, 1908.

Month.	Average Initial Production, bbl.	Wells Completed.	New Production, bbl.	Dry Holes.
January	24½	303	6144	55
February	24½	157	3329	22
March	27½	187	4133	37
April	26	197	4285	33
May	29	264	6628	35
June	29½	390	9856	54
July	23	474	9475	65
August	23	417	8322	55
September	26½	344	7848	49
October	26½	290	6091	51
November	29½	273	6242	51
December	29½	283	5060	41

WELLS COMPLETED IN ILLINOIS IN 1908.

County.	Completed.	New Production, bbl.	Dry Holes.	County.	Completed.	New Production, bbl.	Dry Holes.
Crawford	2,161	43,116	313	Miscellaneous	38	155	33
Clarke	334	6,231	68	Coles	7	91	2
Lawrence	652	21,295	69	Edgar	4	15	2

Outside the old oil district "wildcatting" continued active throughout the year. The principal result was the bringing in of enough wells to start production at Sparta in Randolph county. Here the Sparta Oil and Gas Company has a group of wells a short distance east of the old gasfield. The oil comes from the Chester formation. After finding several 5- and 10-bbl. wells, one was shot which is said to have come in at 135 bbl. a day. The oil is shipped in tank cars and through field lines; approximately 50 bbl. a day are going out. Numerous other wells are being drilled in the vicinity.

On Nov. 1 a well was brought in at Centralia in Marion county. Oil had been found here seeping into a coal mine along a fault plane. The well was drilled a short distance east of the mine; at about 600 ft. a good sand was found in the coal measures. After shooting, the well settled down to an 18-bbl. pumper. Numerous other tests are now being drilled in the vicinity. The significant points in this find are that Centralia is well toward the center of the great structural basin of southern Illinois and that the sand is of approximately the the age of the Casey sand in which oil was first found in Clark county. This geological horizon is widespread, but as yet it has not been found productive in many places.

Small amounts of oil and gas continue to be found at various points along the western edge of the basin, but so far without any important commercial result. The year as a whole was quietly prosperous, and the outlook for 1909 is excellent.

Kansas and Oklahoma. (By Erasmus Haworth.)—During 1908 no remarkable developments were made anywhere in the Mid-continental field. Drilling was greatly diminished compared with either of the two previous years. Nevertheless, many rigs were in operation almost constantly and in the aggregate a great many new wells (more than 1600) were brought in. In Oklahoma drilling was confined principally to a search for oil, and, therefore, gas developments in the main were incidental, while in Kansas more than half of the drilling was done expressly in a search for gas.

Drilling was continued practically in every district in Oklahoma, particularly in the Glenn pool area and the Alluwe shallow pool area. The Glenn pool remained the largest producer, aggregating 50,000 bbl. daily throughout the year. The Alluwe area was greatly extended northward and its total production materially increased. Production in the vicinity of Dewey, and generally throughout the Osage, declined. Scarcely a well was drilled for oil in Kansas, and the production in this State was greatly reduced. Numerous wells are standing idle which would produce in the aggregate large amounts of oil were prices better, but with the ruling low prices they would not pay to operate. No change whatever occurred in the prices of crude oil throughout 1908; oil of 32 deg. B. or lighter brought 41c. regularly.

The Prairie Oil and Gas Company (Standard) again was the principal purchaser of oil, although not so emphatically so as previously. The company's total runs aggregated 33,181,951 bbl., the December runs being estimated, which gives a daily average of 90,661 bbl. Of this amount 25,273,941.54 bbl. were delivered to the refineries, the remainder going to storage. At the end of the year the Prairie

Oil and Gas Company, therefore, had a total storage of 38,971,913 barrels.

The largest independent purchasers of oil were the Gulf Oil and Pipe Line Company and the Texas Oil Company, each of which purchased more than 5,000,000 bbl. The independent refineries are on the increase and consumed perceptibly more oil than the same class of refineries did in 1907. A number of other firms shipped large quantities of oil by rail from different points in Oklahoma, the more important shipments being made by W. G. Barnes, Haywood Oil Company, Quaker Oil Company, W. H. Milliken, Laurel Oil Company, Benson Oil Company, Kansas-Oklahoma Oil Company, Superior Oil Company and a few others shipping smaller amounts.

To all of these sums must be added the amount of crude oil used for fuel, which was fully as great as in any previous year. No account is taken of the amount of oil stored by the producers. It is probable that the total storage in the field at the end of 1907 was as great if not greater than at the close of 1908. The figures included in the accompanying tables, therefore, are for the actual deliveries to purchasers. These tables summarize the entire statistics for the year. It should be noted that there is a slight discrepancy in the figures given out by the Prairie Oil and Gas Company, the amount put in storage plus the deliveries not being quite equal to the total runs. The error evidently was an unintentional one, but sufficient data were not at hand to make the slight corrections desirable.

CRUDE OIL BOUGHT BY PRAIRIE OIL AND GAS COMPANY DURING 1908.

Month.	Total Runs, Bbl.	Daily Average, BBl.	Deliveries, Bbl.	Stored, Bbl.
January	2,914,843	94,027	2,332,098	582,745
February	2,773,592	95,641	2,005,285	768,308
March	2,985,097	96,313	1,994,992	990,705
April	2,979,844	99,328	1,941,284	1,038,561
May	2,487,426	80,240	1,987,441	499,985
June	2,395,326	79,844	2,162,765	232,561
July	2,950,757	95,186	2,326,524	624,233
August	2,858,651	92,215	2,353,609	505,042
September	2,674,264	89,142	2,405,831	268,433
October	2,773,996	89,484	2,297,333	476,333
November	2,620,725	87,358	2,310,282	310,443
December	2,766,829	89,248	2,192,497	574,333
Total	33,181,951		25,273,942	6,871,681

OIL PRODUCTION, MID-CONTINENTAL FIELD, DURING 1908.

	Bbl.		Bbl.
Prairie Oil and Gas Company	33,181,951	Independent refineries	3,107,000
Texas Oil Company	5,298,545	Fuel oil, crude	1,750,000
Gulf Oil and Pipe Line Company	5,654,182	Independent shippers	1,750,000
Total for year			50,741,678

Of the refineries using oil from the Mid-continental field those of the Standard at Neodesha, Kan., and at Sugar creek, near Kansas City, are much more important. They consumed the principal part of the 25,000,000 bbl. delivered by the Prairie Oil and Gas Company, the remainder going to the large refinery at Whiting, Ind. At present 25 independent refineries are in existence, 17 in Kansas and eight in Oklahoma, all of which are in operation excepting four; two of these are closed down temporarily and the other two not quite completed. These 25 refineries have an aggregate annual capacity of more than 3,500,000 bbl. consumption, and are situated as follows: Oklahoma: Big Heart, one; Chelsea, one; Muskogee, one; Oklahoma City, one; Okmulgee, one; Sapulpa, one; Tulsa, two (one closed down). Kansas: Atchison, one (closed down); Caney, one; Chanute, four; Cherryvale, one; Coffeyville, three (two not yet completed); Erie, one; Humboldt, one; Longton, one; Moran, one; Niotaze, one; Paola, one; Petrolia, one. The refinery at Paola changed hands late in 1907, at which time it was reported that it was bought by the Standard Oil Company. Since then this report has been denied, and at present the public is uninformed as to where the ownership lies. These independent refineries consumed a little more than 3,000,000 bbl. of oil during 1908.

A large amount of fuel oil is used in the great-plains area of western Kansas, Oklahoma and Texas, and eastern Colorado and New Mexico, a large proportion of which is drawn from the Mid-continental field. Some of it is taken directly from the wells, the remainder being supplied by the refineries after the lighter oils have been removed. The plains area in general consumes but little fuel; yet in the aggregate it amounts to 4,000,000 to 5,000,000 bbl. annually. It is used by almost every flouring mill, ice plant, electric light plant, and other plants generating steam throughout the entire territory of the plains area where coal is high priced, it having been found that oil is cheaper than coal, especially on account of the lower labor cost involved in its use. Such a large consumer even as the portland cement plant at Yocemento, Ellis county, Kan., finds it advantageous to use oil for fuel.

Ohio and Indiana. (By Harold C. George.)—The oilfield of northwestern Ohio and northeastern Indiana, known as the Lima oilfield, showed a falling off in petroleum production. The total production in 1908 was 7,287,000 bbl., as compared with 8,030,000 bbl. in 1907. A large number of wells were abandoned in this field in 1908 and the production was maintained only by the increased activity in operation due to the higher market price prevailing in 1908 than in 1907. The average price paid for North Lima was $1.03 per bbl., as compared with $0.93½ in 1907, and for South Lima $0.98, as compared with

$0.88½ in 1907. There was a much smaller percentage of dry holes drilled in northwestern Ohio in 1908 than in 1907, as shown in the accompanying table, which gives the production of only those wells that were drilled during the year referred to.

WELLS DRILLED IN THE LIMA OILFIELD IN 1907 AND 1908.

	Number of Wells Drilled.		Daily Production in Barrels.		Daily Production per Well Drilled in Barrels.		Per Cent. of Dry Holes.	
	1907	1908	1907	1908	1907	1908	1907	1908
Northwest Ohio	930	837	8,100	9,252	8.7	11.0	15	9
Northeast Indiana	682	413	5,673	3,405	8.3	8.2	20	19
Total	1,612	1,250	13,733	12,657	8.5	10.1	17	13.5

Ohio. (By J. A. Bownocker.)—The petroleum industry was quiet during 1908. No new pools were found in the Trenton limestone territory of northwestern Ohio. This field has long passed its zenith, and the end is certain, though it may be a long distance ahead. Much the same may be said of southeastern and eastern Ohio, though the decrease has not been so notable, in fact, has only set in. However, no large pools were opened in that territory during 1908, and the great number of wells already drilled notably diminishes the chances of finding large reservoirs. During the summer a pool was opened at Mingo on the Ohio river. Good-sized producers were secured, and there was the usual excitement and enthusiasm. In March, 1907, a well was completed near Pleasantville, in southeastern Fairfield county, and began flowing about 75 bbl. per day. In July a well was completed about three miles northeast of Bremen and began flowing about 40 bbl. per day. Since that time the drill has been active in both areas and a number of good wells have been secured, the largest starting at 250 bbl. The producing sand is the Clifton. It is found usually at a depth of 2340 ft. in the Pleasantville field and at nearly 2600 ft. in the Bremen. The cost of a completed well in the latter pool is about $6000, and although the oil commands the Pennsylvania price, the heavy expense is a great burden. The limits of the field have not been as yet definitely established.

Texas and Louisiana.—Field operations declined in the Gulf, Costal and North Texas fields during 1908. While two new gusher fields were expected, they did not materialize. The Anse Le Butte district, which in 1907 seemed to promise a large output, simmered down, after the expenditure of a large sum in development, to the original discovery well and a few others in the immediate vicinity. Markham and Goose

creek discoveries sent declining crude-oil prices to the lowest point of the year, only to prove a false alarm. Caddo slowly increased its output and prevented the coastal field from showing a decline in production. Old districts were in some cases extended in area, and held up their production better than expected, so that the year's output was satisfactory, in spite of the fact that Jennings during November reached its lowest output for many months. While prices declined greatly they were a fair reflex of conditions and above the normal when compared with the Mid-continent field.

The 1907 production in Texas was 12,322,696 bbl., of which 11,410,078 bbl. was derived from the coastal region. The Louisiana production for 1907 was 5,000,221 bbl. and the total coastal field output, 16,410,299 bbl. The 1908 estimates for Texas are 11,220,000 bbl., of which about 10,520,000 bbl. is credited to the coastal field. The Louisiana output is placed at 5,750,000 bbl., making the total for the coastal field 16,220,000 barrels.

The selling price of crude varied greatly and was so much lower than in 1907 that the total value declined from $14,000,000 in 1907 to about $10,500,000 in 1908. Wildcat drilling was generally confined to local owners of land and State corporations, for oil operators who make a regular business of exploiting prospective oil territory were engaged in the Mid-continental field, where the percentage of dry holes proved less than in Texas. According to the *Oil Investors' Journal,* the number of completed wells in the Coastal field in 1908, was 833, of which 616 were oil producers, 16 gas wells and 201 dry holes. The record for a similar period in 1907 was 733 producing oil wells, 22 gas wells and 244 dry holes. In north Texas, 81 wells were

PRODUCTION OF GULF COAST FIELDS (a).
(In barrels)

District.	1906	1907	1908
Spindletop	1,077,492	1,699,943	1,741,070
Sour Lake	2,156,010	2,353,740	1,580,655
Saratoga	2,182,057	2,130,928	1,700,968
Batson	2,289,507	2,164,453	1,584,500
Humble	3,571,445	2,929,640	3,777,316
Dayton	92,850	106,038	39,901
Matagorda	8,000	4,500	2,000
Hoskins Mound	72,591	12,000	15,875
San Antonio (Mission field)	.....	5,000	5,000
Markham (b)	.....	.....	60,869
Piedras Pintas	.....	8,354	16,019
Corsicana	332,622	226,311	211,335
Powell	763,221	596,897	398,649
Henrietta	111,072	83,260	82,639
South Bosque	1,300	8,000	.....
Jennings	9,025,174	4,895,905	4,856,889
Welsh	23,996	36,000	.....
Anse La Butte	23,708	76,938	219 265
Caddo	4,650	48,266	513,504
Goose Creek (b)	.....	.....	11,160

(a) *Oil Investors' Journal.* (b) New field in 1908.

completed during 1908, of which 21 were dry. These figures do not include those wildcat wells outside of the Markham and Goose creek fields, and the gas wells were mainly in the Caddo field though Humble and Henrietta contributed a few. The production of the different districts is shown in an accompanying table.

The crude market opened weak in 1908 and the use of weathered crude from Oklahoma and northern Texas further depressed prices early in the year, so that the posted credit-balance prices (which may be taken as the base price paid by consumers) ranged from 67 to 75c. The quotations compared favorably with those of 29 to 41c. per bbl. paid in Oklahoma, when the quality of the crude is considered, although it must be borne in mind that for fuel purposes (practically the sole use of coastal crude) an oil of 18 to 20 deg. B. is better than that of 30 to 32 deg. B. The undoubted effect of high crude-oil prices in 1907 was shown in the 500,000-ton increase in the coal output of Texas for that year where previous years had shown a decline.

Quotations declined in March, April and May, until the highest price posted was 60c., and in June the Gulf and Texas companies suspended quotations while the Sun Company prices ranged from 38 to 42c., a decline of more than 100 per cent. Stocks were increasing, demand was poor, and there was general uneasiness regarding the capacity of the new pools at Markham and Goose creek. Buying by cotton compressors, sugar planters, and by the railways, combined with the failure of the new pools and the operations of the Oil Producers Marketing Company raised the posted prices gradually to 55 to 57c. in October. Subsequently the market remained weak. There is no valid reason for any advance in the near future unless production declines greatly, when local conditions may warrant higher prices.

If the proposed Prairie Oil and Gas Company pipe line from Oklahoma to New Orleans is constructed, a market now exclusively supplied by coastal crude will be divided and consumption further reduced. The preliminary survey of 520 miles for this line has been made, and all the interested Oklahoma municipalities have consented to the crossing of their highways. Under existing and probable future conditions coastal producers will be fortunate if the average price for 1909 equals that for 1908, which was about 60c. per bbl., compared with 84c. in 1907. These prices do not apply to the Caddo field, the oil from which is not taken by the Southern pipe lines, but has been shipped by tank cars and for which recent prices have ranged from 35 to 38c. per bbl. Careful investigation by the *Oil Investors' Journal* placed crude stocks on Oct. 1, at 3,130,000 bbl., and the amount of stocks on Dec. 31, was approximately 3,020,000 barrels.

The suits instituted by the State against the Standard Oil Company, Security Oil Company, Union Tank Line Company, and other alleged affiliated corporations resulted in a receiver taking charge of the Union Tank Line Company cars, which were promptly leased to one of the railways. The Security Oil Company succeeded in obtaining relief from the vexatious conditions of the injunction proceedings, and its large refinery near Beaumont was put in operation again in January. The suits, which are for ouster and for penalties amounting to many millions with the customary collateral garnishee proceedings, drag along subject to the usual legal quibbles and delays. The protests against the burdensome Kennedy gross tax laws continued, but most operators paid the tax. Only one or two, including the Southwestern Oil Company of Houston, are testing the constitutionality of the act.

Production in Louisiana was confined entirely to fields located prior to 1908, although wildcats were drilled in Rapides, Webster, Terrebone, Winn, Saint Landry, and Calcasieu parishes during the year. The Welch pool remains small and practically no effort was made to develop it. The Jennings pool was extended about a half mile westerly, the most important development since the Latrille 40-acre extension in 1907. Many of the new wells had an initial flowing output of 3000 to 5000 bbl. daily. After a brief period compressed air was used on them giving a much larger output than they would have yielded under the pump. The pool is not only the largest producer in Texas or Louisiana, but also leads in average production per well.

More than 100 new wells were drilled in 1908, but the last months of the year showed a rapid decline in field operation and production. The maximum output of Jennings was 557,000 bbl. in March, and the minimum, 320,000 bbl. in November, while the 1908 output was practically the same as in 1907.

The Anse La Butte pool proved a disappointment, despite active work and a very large expenditure of money. Salt water is the greatest detriment encountered, and the few producing wells are confined to those in the vicinity of the original well drilled by the Lake Oil Company. The average daily output does not exceed 1000 bbl., and the production for 1908 was about 220,000 barrels.

Caddo field operations were gradually extended during 1908, and this pool will certainly be the most active one in Louisiana during 1909; 58 wells were completed up to Dec. 31, 1908, and the total output is estimated at 515,000 bbl. The proved field now extends over an area of many square miles and both the U. S. and State Geological Surveys are investigating its resources and geology. Public lands in the Caddo district have been withdrawn from sale pending regulations

to restrain the enormous waste of natural gas. Conditions are radically different from those of the large gusher fields of the coast (which are of small area) and will be more slowly developed. While the gas wells are of large capacity the oil wells are small producers when compared with those of Spindletop, Saratoga, or Humble. Drilling is expensive in addition to being slow and hazardous on account of the enormous gas pressure usually found in the overlying strata at 820 to 940 ft. Most of the producing oil wells are in the 2200- to 2275-ft. sand, although there are several in a sand at 1580 ft. The gravity of the oil varies from 24 to 39 deg. B., and the initial production runs from 100 to 400 bbl. per well. The output in January, 1908, was only 8000 and gradually increased to 90,000 bbl. in December.

In Texas, Humble and Spindletop were the only pools showing an increased output as compared with 1907. Humble's large production was caused by active development in the northern extension of the proved field. Completed wells to Dec. 31, aggregated 295, a number three times as large as in any other Texas field. The average initial capacity of the wells was also greater than in any pool outside of Jennings. Spindletop production increased slightly, and field work was more extensive than at Batson or Saratoga. The only feature at Sour Lake was the expanding of the proved area 450 ft. easterly, while at Saratoga drilling into the deeper sand at 1490 ft. prevented a more serious decline than would otherwise have occurred. The small pools at Mission, Liberty, Dayton, and Piedras Pintas were only notable for inactivity and their output was confined to a few wells of limited capacity.

In Clay county, northern Texas, some excitement was caused by the bringing in of a few oil wells of larger flow than usual. Gas wells of great volume were also located and are being utilized by the Navarro Refining Company, to supply the cities of Henrietta and Wichita Falls, with a probable extension of the pipe line to Dallas and Fort Worth.

None of the wildcat wells scattered through Texas created anything more than local interest other than those at Markham and Goose Creek. The first-named field is in Matagorda county, 6½ miles northwest of Markham, a station on the Victoria division of the Southern Pacific Railway. The topography of the district is similar to that of other coastal fields. Gas seepage induced the Hardy Oil Company to drill a well which was brought in early in June at a depth of 1370 ft. The pay sand resembles that at Jennings, the gravity of the crude is 22 deg. B., and the initial capacity of the well was about 80 bbl. per hour. The Rio Bravo Oil Company (Southern Pacific Railway) promptly laid

a pipe line to the railway and contracted for the output at 50c. per bbl. Another well of smaller capacity was soon brought in, but subsequent field operations by the Guffey company and others extended over a wide area failed to find any extension of the pool. Production soon fell off and in consequence of the failure to "make good" the field is practically idle at the present time, although there is a bare possibility of its developing into a fair producer. The output to date has been about 60,000 barrels.

Goose Creek, the other discovery which gave promise of being a gusher field, was found about the same time as Markham. The territory lies in Harris county, 25 miles east of Houston at an altitude of 30 ft. above the level of Galveston bay, and only a mile distant from the barge channel. The discovery well was the third test hole drilled by Armstrong, Peden & Co., the second well having yielded a small amount of oil which was utilized for fuel. Well No. 3 was finished at 1600 ft. (much deeper than usual in similar formations along the coast) and yielded about 700 bbl. daily, of 18 deg. B. gravity crude. The Producers' Oil Company acquired the producing property and drilled about 10 wells to determine the area of the field. All these holes proved dry except one or two in the immediate vicinity of well No. 3. Several wells are now going down, but most operators have formed an unfavorable opinion regarding their success, although their opinion may be fallacious. The Texas Company has laid a pipe line to the barge channel, so that water shipments can be made, but the output so far is merely nominal.

It is unlikely that the Texas-Louisiana production will decline materially in 1909, for while the old fields are falling they are still

PRODUCTION OF LOUISIANA AND TEXAS. (a)
(In barrels of 42 gal.)

	Texas.			Louisiana.		
Year.	Production.	Value.	Avg. value per bbl.	Production.	Value.	Avg. value per bbl.
1896	1,450	$ 1,050	$0.720			
1897	65,975	37,662	.570			
1898	546,070	277,135	.508			
1899	669,013	473,443	.708			
1900	836,039	871,906	1.043			
1901	4,393,658	1,247,150	.284			
1902	18,083,658	3,998,097	.221	548,617	$ 188,985	$0.344
1903	17,955,572	7,517,479	.418	917,771	416,228	.453
1904	22,241,413	8,156,220	.367	6,718,958	2,438,952	.363
1905	28,136,189	7,552,262	.268	8,910,416	1,601,325	.180
1906	12,567,897	6,565,578	.522	9,077,528	3,557,338	.392
1907	12,322,696	10,401,863	.844	5,068,425	4,114,561	.812
1908	11,217,155	6,730,298	.600	5,633,634	3,492,753	.620
Totals	129,036,785	$53,830,223		36,875,349	$15,810,642	

(a) From *The Oil Investors' Journal.*

capable of development and will be actively worked. Caddo will surely increase its output and make up largely for the deficiency in other fields. There is also the possibility that Markham and Goose Creek will develop into fairly good fields, although it must be admitted that operators are pessimistic. It is to be hoped that no very large pools will be discovered in the near future, for the present supply fully equals consumption in local markets and a new gusher field would cause a fall in crude prices demoralizing to producers in the old pools.

Wyoming. (By H. C. Beeler.)—More drilling was done in 1908 than for a long time in the oilfields of Wyoming. In the Salt Creek field in Natrona county, one gusher was struck, and the rigs were moved to another location where drilling is now going on. The Popo-Agie Oil and Power Company is drilling near the Murphy wells for oil, for fuel for the new power plant. A pipe line is contemplated from the wells to Hudson, the nearest railroad point, and additional rigs will be put into commission as soon as they can be advantageously placed. Drilling is going on near Ft. Washakie, Shoshone Indian Reservation, on leased Indian lands. A gas gusher burned for a year near Basin, Big Horn county, but was finally capped. Additional wells were drilled in the vicinity to furnish local consumers and further to test the fields. At Byron and Garland, in the same county, oil and gas were struck; drilling still goes on. The Uinta county fields were more active than for several years; indications are that production is to be permitted, and these high-class oils put on the market. High freight charges have hitherto kept the production down. All things considered, Wyoming held its own during 1908, and while location work and patenting of mineral, oil and coal lands all were seriously hindered and the miners were put to much unnecessary expense and trouble by the overzealous agents of the National Forest and Land Departments, whose various reserves and "withdrawn" areas cover most of the State, general prospecting was active and productive of good results.

Petroleum in Foreign Countries.

Africa.—The Orange River Colony will probably become the center of the oil industry in South Africa. Investigations of the field have been going forward for several years, but were conducted with the greatest energy during 1908. The result of this work has been the mapping out of a broad oil belt extending across the Colony in a northeasterly-southwesterly direction, and including the following districts: Harrismith, Bethlehem, Senekal, Ficksburg, Ladybrand, Thaba 'Nchu, Bloemfontein, Wepner, Smithfield, Edenburg and Fauresmith. In the Ladybrand district are found oil springs, oil shales, blue clay, ozoker-

ite and quartz-sandstone impregnated with oil and bituminous tufa. In every case the basins are clearly defined, the inclosing ground consisting of the undulating country, which gives the synclines and anticlines of petroleum fields. The whole area is intersected by huge igneous dikes that stand vertically. These still retain on their sides and in their cleavage, in many cases, flakes of crude oil, and a strong odor of paraffin throughout the subterranean reservoirs. Oil discoveries have also been made in Nigeria, West Africa.

Austria Hungary.—The petroleum industry in Galicia during 1908 again suffered from over-production. The production of the Boryslaw-Tustanowice field was 1,581,080 tons. The over-production amounted to about 450,000 tons; this, added to the unmarketed oil of previous years, taxed the storage room at Boryslaw to its utmost. Thus, a crisis has been brought about, to deal with which measures must be adopted to increase the market for crude oil and products, to provide additional storage room, and possibly to curtail production. In 1908 there were 45 producing properties in the Boryslaw-Tustanowice district whose output exceeded 10,000 tons. The largest producer was the Karpathen company, with a total of 113,530 tons. Other producing Galician fields are the Schodnica, Krosno, Urycz, Rogi, Potok, etc. The total yield from these fields, however, was small and had little effect on the general position of the Galician petroleum industry.

The highest prices for petroleum prevailed during February and March, when quotations ranged from 16 to 17 kronen per ton. After this there was a gradual decline down to August, during which month prices ranged from 7 to 9 kronen per ton. During December the lowest quotation was 10.50 kronen, and the highest 12.50 kronen per ton.

PRODUCTION OF BORYSLAW-TUSTANOWICE FIELD.
(In metric tons.)

Year.	Tons.	Year.	Tons.	Year.	Tons.	Year.	Tons.	Year.	Tons.
1899..	18,000	1901..	132,000	1903..	373,000	1905..	546,500	1907..	(a) 1,011,590
1900..	55,000	1902..	226,000	1904..	546,000	1906..	562,000	1908..	(a) 1,581,080

(a) Includes the Tustanowice field in 1907 and 1908.

Canada.—Practically the whole production of crude petroleum in Canada is obtained from the Ontario peninsula. The production for 1908, estimated on the basis of the bounty of 1½c. per gal., paid by the Dominion Government, was 527,987 bbl., valued at $277,193. The decrease as compared with 1907 is about 33 per cent.

China. (By T. T. Read.)—China possesses no extensive fields of petroleum so far as is known. Those known to exist are undeveloped

and but little is known regarding them. The most important district is in Ssu-chuan; this was described by a Jesuit father in a paper in the *Annales des Mines* some years ago. The production is probably small, as the natives do not know how to refine the petroleum, but use it in the crude state. Recently newspaper accounts have appeared of the discovery of an important field in Shensi, and the formation of a Chinese company to exploit it, but I have been unable to get any definite information. Petroleum is said to occur at Kansu, but the statement is unconfirmed.

Dominican Republic.—According to the British Vice-Consul at Santo Domingo, the petroleum belt measures over 190 square miles. Oil is found in abundance in the province of Azua, and the fields are said to extend from a point near the town of Azua for many miles into the interior. At present this area is under exploitation by an American company known as the West Indian Petroleum Mining Company.

Germany.—The production of petroleum was 141,900 metric tons in 1908, as compared with 106,379 tons in 1907. Petroleum products imported amounted to 1,416,807 metric tons in 1908, as compared with 1,381,773 tons in 1907. These imports included illuminating and lubricating oils, crude, refined, and heavy benzine, crude oil, gas oil, residuals, patent turpentine and other mixtures. The exports of petroleum products for 1908 amounted to 17,345 tons, as compared with 17,151 tons in 1907. Exports were made to the United Kingdom, Holland, Sweden, Switzerland, Denmark, Norway, France and other countries.

Japan. (By T. T. Read.)—An important petroleum field has been opened on Saghalien island. A company has been formed, chiefly with German capital, and work is under way. The crude oil is to be sent to Japan for refining.

Mexico.—Important developments were made in the petroleum industry during 1908. Exploitations, conducted at widely separated points, seem to indicate that a large territory, bordering on the Gulf of Mexico, is underlaid with oil. The oil industry is in the hands of a few large companies, and little reliable data is obtainable as to the extent of the operations of the different concerns. Although the industry is still in the formative stage, the existence of oil in large quantities has been proved, and Mexico may be expected to figure largely as a producer within a few years. For the most part, the wells now producing furnish a heavy variety of oil with an asphalt base. While not possessing the refining qualities of the lighter oils, the product has been found valuable on account of its by-products.

Persia.—Extending along the valleys of the Euphrates and Tigris in a northwest and southeast direction, down the Persian Gulf and into Beluchistan, is a broad belt that shows an abundance of petroleum indications. Prospecting so far has been limited and in most cases unsuccessful. However, the area is considered promising, and further prospecting is expected to bring forth better results.

Rumania.—The petroleum production during 1908 was 1,147,727 metric tons, distributed as follows: District of Prahova, 1,095,821 tons; Dambovitza, 26,272; Buzeu, 10,766; Bacau, 14,868. Of this amount, 460,704 tons were exported. About 250,000,000 francs is invested in the petroleum industry in this country. The future outlook for the industry, as far as increased production is concerned, is not bright.

PRODUCTION OF PETROLEUM IN RUMANIA.

Year.	Metric Tons.	Year.	Metric Tons.	Year.	Metric Tons.	Year.	Metric Tons.
1897.....	110,000	1900.....	260,000	1903.....	384,202	1906.....	887,094
1898.....	180,000	1901.....	270,000	1904.....	497,000	1907.....	(a)1,120,097
1899.....	250,000	1902.....	310,000	1905.....	614,870	1908.....	1,147,727

(a) Reported by the *Petroleum Review.*

Russia. (By I. I. Rogovin.)—The oil industry in Russia during during 1908 was characterized by a small yield and high prices. The accompanying table shows the yield of oil in the Baku region during

OUTPUT OF OIL AT BAKU AND GROSNY FROM 1901 TO 1908.
(In millions of poods.)

	1901	1902	1903	1904	1905	1906	1907	1908
Baku.....	671	636	600	615	410	448	476	} 466
Grosny.....	35	34	33	40	43	38	40	
Total.....	706	670	633	655	453	486	516	466

a period of years. Whether 1909 will be more favorable with respect to production in this region is difficult to state, but judging from the number of wells in operation, and their capacity, the outlook is far from favorable. With the exception of 1905, the average daily yield has shown a decrease for the last five years, whereas the number of wells has steadily increased.

During the first half of 1908 there was a decrease in the price of oil; the lower figures obtained during the latter part of the year. The average prices for oil products in Baku during 1908 were as follows: The first figure is for light oil on hand, the second for residue in schooners, and the third for kerosene in schooners (prices are in copecks

per pood). Jan. 1 to 15, 24.87, 25.6 (.....); Feb. 1 to 15, 26.375, 27.165, 34.95; Feb. 15 to 29, 27.56 (.....) 30.00; March 1 to 15, 27.35 (.....), (.....); March 15 to 31, 25.25, 26.5, 30.75; April 1 to 15, 22.75, 25,925, 28.66; April 15 to 30 (.....), 23.74, 28.5; May 1 to 15, 22.146, 23.00, 27.61; May 15 to 31, 22.42, 23.00, 28.625; June 1 to 15, 21.42, 21.25, 27.67; July, 18.92, 20.12, 24.66; August, 18.76, 20.67, 24.55; September, 19.596, 20.61, 26.597; October, 19.656, 20.253, 26.522; November, 18.967, 19.65, 25.375; December, 18.75, 19.50, 25.00.

The increase in the price of fuel oil led to a decrease in the demand for oil and residue, and coal, timber, and even peat were largely used in their stead. The crisis in the cotton industry, which is the chief consumer of oil in the central region, will probably cause a decrease in the home market for oil.

Exports of illuminating oils showed a slight increase in 1908 as compared with 1907. Shipments from Batum to Europe amounted to 16,700,000 poods, as compared with 14,445,000 poods in 1907. Exports of residues in 1908 were 2,093,000 poods, as compared with 185,000 poods in 1907. Exports to the far East markets in 1908 were 9,265,000 poods, as against 11,954,000 poods in 1907.

Extensive prospecting was carried on during 1908 in a new area in the north of Russia, Province of Archangel, in the basin of the Uhta river. Indications are that this region will become of great importance.

Bibliography of Petroleum in 1908.

The following articles and papers, which are only a small percentage of the total, appeared in the technical press during 1908:

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PHOSPHATE ROCK.

By Edwin Higgins.

The production of phosphate rock in the United States in 1908 was 2,375,031 tons of 2240 lb., as compared with 2,251,459 tons in 1907. The principal producing States were Florida, Tennessee and South Carolina, named in the order of their importance. Arkansas and Idaho contributed a few thousand tons to the total. The accompanying tables afford interesting comparisons of the statistics of phosphate rock in the United States and foreign countries.

STATISTICS OF PHOSPHATES IN THE UNITED STATES. (a)
(In tons of 2240 lb.)

Year.	Production	Imports.	Exports. (b)	Consumption.	Year.	Production	Imports.	Exports. (b)	Consumption.
1899.....	1,663,476	118,613	867,790	914,299	1904.....	1,874,428	166,090	842,484	1,198,034
1900.....	1,527,711	144,006	619,995	1,051,722	1905.....	1,933,286	82,072	934,940	1,080,418
1901.....	1,483,723	180,714	729,539	934,898	1906.....	2,052,742	46,228	904,214	1,194,756
1902.....	1,600,813	145,793	802,086	944,250	1907.....	2,251,459	55,017	1,018,212	1,288,264
1903.....	1,581,576	153,972	785,259	950,289	1908.....	2,375,031	58,203	1,188,411	1,244,823

(a) Production statistics of 1901 and subsequent years, except 1905-1908, are those of the U. S. Geological Survey and are based on marketed products. (b) Neglecting the insignificant re-exports of foreign product.

IMPORTS OF FERTILIZERS INTO THE UNITED STATES.
(In tons of 2240 lb.)

	1905		1906		1907		1908	
	Quantity.	Value.	Quantity.	Value.	Quantity.	Value.	Quantity.	Value.
Guano.....	25,651	$365,823	22,947	$320,565	29,141	$365,257	31,469	$420,724
Crude phosphates.....	56,421	275,889	23,281	147,547	25,876	163,944	26,734	175,365
All other fertilizers.....	.....	4,048,403	.....	4,231,723	.....	4,994,346	.....	4,394,434

Phosphate in the United States.

Arkansas. (By A. H. Purdue.)—As in past years, only one company, the Arkansas Fertilizer Company, operated in Arkansas. The phosphate rock is quarried on Lafferty creek in Independence county, about 12 or 15 miles northwest of Batesville. From this point it is shipped to Little Rock, and ground and made into acid phosphate, from which as a base is manufactured fertilizers of a number of different formulas.

In 1908 the output of the quarries was somewhat in excess of 6000 tons, from which about 10,000 tons of acid phosphate was manufactured.

Florida.—The phosphate industry during 1908 was characterized by a decline in prices. The year opened with high-grade quoted at $10.25

PRODUCTION OF PHOSPHATE ROCK IN THE UNITED STATES. (a)
(In tons of 2240 lb.)

Phosphate.	1905 (a)		1906 (a)		1907 (b)		1908 (c)	
	Tons.	Value.	Tons.	Value.	Tons.	Value.	Tons.	Value.
Florida hard rock..	579,228	$3,011,986	561,370	$3,312,063	589,217	$3,714,767	642,259	$6,262,025
Florida land pebble	401,997	803,994	603 382	1,810,146	721,028	2,523,598	1,026,392	5,131,960
Florida river pebble	90,065	216,156	41,742	116,878	36,729	139,570	5,000	17,500
Total Florida....	1,071,290	$4,032,136	1,206,494	$5,239,107	1,346,974	$6,377,935	1,673,651	11,411,485
S. Car. land rock...	221,712	$731,650	270,000	$990,000	228,354	$890,581	250,000	1,687,500
S. Car. river rock...	30,284	87,824	45,000	144,000	37,303	126,830	40,000	240,000
Total S. Carolina..	251,996	$819,474	315,000	$1,134,000	265,657	$1,017,411	290,000	1,927,500
Tennessee.....	610,000	$2,074,000	520,381	$2,029,486	626,683	$3,008,078	403,180	1,673,197
Other States.....	.....	....	10,867	61.942	12,145	47,098	8,200	28,700
Total United States	1,933,286	$6,925,610	2,052,742	$8,464,535	2,251,459	$10,450,522	2,375,031	$15,040,882

(a) Statistics of 1905 and 1906 are those of J. M. Lang & Co., Savannah, Ga., with respect to quantity, and are based upon shipments. (b) As compiled by *The Mineral Industry*, the tonnage figures for Florida being supplied by J. M. Lang & Co. (c) Statistics of J. M. Lang & Co., except for Tennessee and Other States.

PRODUCTION OF PHOSPHATE ROCK IN THE WORLD.
(In metric tons.)

	1901	1902	1903	1904	1905	1906	1907
Algeria.....	269,878	260,859	301,112	344,969	334,784	333,531	315,000
Dutch W. Indies.....	(c)	10,530	15,511	22,764	22,940	26,138	.....
Belgium.....	(a)222,520	(b)135 850	(b)184,120	(b)202,480	193,305	152,140	180,000
Canada.....	937	776	1,205	832	1,180	(c)	680
Christmas Island.....	42,125	61,179	70,096	71,757	97,052	90,561	200,000
France.....	535,676	543,900	475,783	423,521	476,720	469,408	375,000
Norway.....	738	2,295	1,795	1,456	2,522	3,452	1,830
Russia.....	21,276	13,709	14,635	20,282	(c)	(c)	(c)
Spain.....	4,220	1,150	1,124	3,305	1,370	1,300	3,078
Sweden.....	(c)	3,895	3,219	2,929	(c)	.....	5,317
Tunis.....	178,018	263,482	332,888	455,789	559,645	796,000	1,040,300
United States.....	1,507,548	1,514,159	1,606,881	1,904,419	2,135,449	2,085,596	2,251,459

(a) Cubic meters. (b) Metric tons of phosphate of lime; in addition there were 315,200 cu.m. of phosphatic chalk in 1902, 350,250 cu.m. in 1903, 311,640 cu.m. in 1904, 80,380 cu.m. in 1905 and 119,450 cu.m. in 1906. (c) Statistics not available.

@10.50 per ton, f.o.b. shipping point, and land pebble at $8.25@8.50 f.o.b., Port Tampa. February quotations were $9.50@10 for high-grade, f.o.b. Florida and Georgia ports, and $5.25@5.50 for land pebble. Prices declined still further during March. Few sales were made in the spring, the manufacturers holding off expecting that prices would go lower, while the miners held out for better prices. There was little or no recession in the price of high grade, but by the end of August, land pebble was reduced from $4.25 to $3.50@4. September showed a further reduction in land pebble, the quotation being $3.35@3.50.

High grade was practically unchanged toward the close of the year. During October and November, the land pebble market stiffened slightly, quotations being $3.50@3.75. At the close of the year, in order to rehabilitate the weak market, phosphate miners closed down plants that represented a production of 200,000 tons per annum.

Early in 1908, Edward Holder, who has been active in Florida phosphate for many years, organized the Holder Phosphate Company. This company began the erection of two large plants near Inverness. The Saunders Phosphate Company began mining operations with a new plant on the Herron property, near Hernando. After opening up the mine, the Saunders company leased the property to the Mutual Mining Company. The Mutual company also has producing properties at Luraville and Floral City. Later in the spring it began the construction of two other washers at Wade. The Corona Phosphate Company was organized for the purpose of mining a high-grade pebble rock, near Plant City. The Medulla Phosphate Company completed its plant at Christina and began operations on June 1. This company mines a high-grade pebble. At this period of the year the pebble miners had demands for their entire output. From shipments made to domestic ports, it became apparent that the domestic consumption of pebble was increasing rapidly. Discovery of a good grade of phosphate was reported near Lake Helen, Volusia county, by N. P. Roberts. If this discovery comes up to the expectations of those interested, the Florida phosphate fields will be greatly extended.

The new drying and loading plant of the Cummer Lumber Company at Jacksonville was completed and phosphate shipments begun Nov. 1. Through the opening of this port another Florida shipping point has been added to the list. It is expected that the Cummer Lumber Company will ship at least 75,000 tons annually through this new plant. This company now has six mines in operation at Newberry.

South Carolina.—The production of phosphate during 1908 showed only a slight decrease as compared with that of 1907. The following-named companies operated land rock deposits, the first named being the most important: Charleston Mining and Manufacturing Company, Charleston; Bolton Mines Company, Charleston; Runnymede Phosphate Company, Charleston; and Bulow Mining Company, Charleston. The Stono Mining Company, Charleston, and the Central Phosphate Company, Beaufort, operated river rock deposits. The price of ground rock, f.o.b. Charleston, was practically unchanged throughout the year, the quotations being $6.75@7 per 2000 lb. South Carolina, hot air dried, f.o.b. Ashley river, was quoted at $7@7.25 per 2000 lb. during January and February. These prices were slightly lowered during March, but

remained practically unchanged during the latter part of the year. In January, South Carolina, undried, f.o.b. Ashley river, was quoted at $5.75@6 per long ton. In February prices were lowered to $5.50@5.75. During the remainder of the year there were slight concessions from these figures.

Tennessee. (By H. D. Ruhm.)—The situation in the Tennessee phosphate field during 1908 was fairly well predicted in these pages last year, when I said: "The only possible salvation of the situation as to present prices is in the policy of curtailing production, and, unless the miners get together and consolidate in some way, their shrift is short for the maintenance of recent prosperity." No such consolidation was effected, and in the meantime the manufacturers increased the efficiency of their mining organizations to such an extent that they now seem amply able to take care of their own needs for several years to come.

This almost entirely cut out the selling of rock on the domestic market, and, with the exception of a few old contracts and a few new sales (mostly for export), shipments in 1908 were largely confined to the operations of the Virginia-Carolina Chemical Company (Charleston Mining and Manufacturing Company), the Federal Chemical Company, the American Agricultural Chemical Company and the Independent Phosphate Company with its allied interests. The shipments from points in Tennessee on the Louisville & Nashville railroad for the years 1905 to 1908, inclusive, are given in the accompanying table:

SHIPMENTS FROM POINTS IN TENNESSEE ON THE LOUISVILLE & NASHVILLE RAILROAD.

Year.	Domestic.	Export.	Total.	Year.	Domestic.	Export.	Total.
1905.....	285,321	110,407	395,728	1907.....	523,686	102,996	626,682
1906.....	379,894	110,514	490,408	1908.....	320,634	81,123	401,757

Inasmuch as the large bulk of the phosphate rock of 1908 was mined and shipped by manufacturers for their own consumption, it can well be understood that the year was a very lean one for the miners who are dependent upon the general market. There was a marked falling off in prices. In September, 1907, the prices were for 72 per cent. domestic, $5.25; for 75 per cent. domestic, $6.25; for 78 per cent. domestic, $7.; 75 per cent. export, $7.50, f.o.b. mines. In September, 1908 (for what few sales could be made), the prices ranged around: 72 per cent., $3; 75 per cent., $3.50; and 78 per cent., $4.50 for local buyers, with some small contract buying at 25@50c. in advance of above prices. Very little phosphate rock was sold, however, at these prices.

The Independent Phosphate Company, under the management of T. C. Meadows, made a considerable reduction in the cost of output in comparison with that of 1907, so that after paying sinking fund, interest, and providing a surplus, it paid a 10-per cent. dividend on the basis of $3.60 per ton for rock containing 73 to 75 per cent. calcium phosphate. The organization of the Independent Fertilizer Company was announced during November, 1908; its officers are Waldemar Schmidtman, president; T. C. Meadows, vice-president and general manager; J. H. Carpenter, treasurer. It is understood that this company has obtained control, by purchase, of about 50 of the independent fertilizer companies in the United States.[1]

During 1908 the Swan Creek railroad to the Mayfield mines of the Virginia-Carolina company was completed, and that company began supplying a large number of its factories with blue phosphate which is said to be much superior to the Florida pebble rock, and almost as good as the high-grade Mount Pleasant rock. The Independent Phosphate Company completed its railroad to Estes Bend and has the machinery on the ground for its plant there. The railroad to Leatherwood starts from Nashville; enough grading has been completed to warrant the expectation that the company will commence operations at Leatherwood by June 1, 1909.

The Middle Tennessee Phosphate Company completed its washer during 1908, and now has one of the best washing and drying plants in the field. The Blue Grass Phosphate Company completed its second washer and dryer, and the Jackson Phosphate Company got its dryer in operation. T. C. Meadows & Co. completed their drying plant. The Federal Chemical Company is installing washers of improved construction at both the Century and Tennessee branches. The Richland Phosphate Company's drying plant was completed and plans for a washer are under consideration. The Columbian Phosphate Company, after having paid 10-per cent. dividends every six months for a number of years, finally worked out to the point where it was not desirable to maintain a separate organization, so it was taken over by the Petrified Bone Mining Company, an affiliated concern. Both the Maury Phosphate Company and France & Co. have plans under consideration for a washer and dryer. A new organization known as the Franco-American Phosphate Company has under option a large amount of undeveloped phosphate property and one going concern, the Volunteer Phosphate Company, of Centerville; it is said that this company plans to build two large plants.

The demand for phosphate properties is decidedly strong, and owners

[1] According to latest reports (May, 1909) this consolidation appears likely to fail of consummation. *Editor.*

of phosphate land are still holding at high prices, although at considerably lower than were asked in 1907.

The most noteworthy development of the phosphate business during 1908 was the marked tendency to use washers for preparing all grades of rock for market. In the first stages of the phosphate business, washers were extensively used, but as the method of recovery was over screens entirely, and as nothing was recovered smaller than the oversize from screens having ⅛x⅜-in. openings, it was found not only that an enormous percentage of the rock itself was ground up by the logs and screens and carried off with the waste into the settling ponds, but also that a considerable percentage of the clay formed into balls and went with the phosphate oversize from the screens. This clay, which baked into hard lumps during drying, not only decreased the resulting percentage of bone phosphate, but also increased the amount of deleterious iron oxide and alumina in the rock.

The first departure was to abandon all the machinery and to return to the old-fashioned method of simply drying in kilns on cordwood. As the mining progressed and more effort was made to save the portion of the deposit formerly wasted, mechanical dryers of the cylindrical type were brought into use and the entire deposit saved. In this method the rock was also badly pulverized, and much of it was lost with the screenings when the making of high-grade rock was attempted. By screening or by proper selection in the mine, the entire product of the dryer can be saved, but it is of only an average grade, still having a considerable amount of the clay, while a large amount of rock is left in the rejects, or in the waste bank.

Producers then set about designing machinery that would save all of the rock and eliminate all of the dirt, with the result that practically everyone has turned to the combination of washers and dryers, the finest grains of phosphate being recovered by settling in water. It has been demonstrated that the very fine granules can be concentrated so as to contain 75 per cent. or better in phosphate of lime, and that by this method the saving is nearly double that made with the old wasteful method.

The best results so far have been obtained by the Federal Chemical Company, which has perfected methods to the point where it has felt justified in spending many thousand dollars in the erection of plants to do this work. The method consists in washing the lump rock over drag conveyers, with hose pipes, passing the small pieces and clay through submerged screens and then through pug mills; by this method the small clay balls are entirely eliminated. The fine waste and water is then pumped into a series of settling tanks from which the

fine settlings are washed out and pumped into wet storage bins; the water is then allowed to drain out for several hours, and finally the fine material is passed through cylindrical dryers.

The Charleston Mining Company uses log washers and passes the waste through hydraulic classifiers, the phosphate granules discharging at the bottom to the dryers and the clay passing out at the top to the waste ponds. The Independent Phosphate Company and its allied companies are proceeding on a combination of the above principles, and have accomplished good results, with a much smaller outlay of money.

The bad reputation achieved for Tennessee rock during the hurry-scurry methods of 1907 has been almost wiped out by the high-grade product shipped in 1908, and by another year it will be as easy to depend on getting uniform high-grade rock from Tennessee as it is now to get uniform low-grade rock from the Florida pebble district. The use of Mount Pleasant raw pulverized phosphate rock for direct application to the soil and for mixing with barnyard manure received a great impetus during 1908, and more was shipped for this purpose than ever before. Farmers who have used phosphate rock in this manner for a number of years seem to get good results, and this may prove an outlet for the producer not connected with the fertilizer manufacturers.

Western Fields.—The series of phosphate deposits found in the mountain ranges from central Utah to east-central Idaho and in western Wyoming, mention of which was made in Vol. XVI of THE MINERAL INDUSTRY, are believed to constitute the largest known area of phosphate-bearing beds in the world. Acting under instructions from President Roosevelt, the Secretary of the Interior, on Dec. 10, withdrew from entry, selection and location all public lands in Wyoming, Idaho and Utah, believed to contain phosphate rock, pending appropriate action by Congress. This action was taken largely because of facts brought out at the conservation meetings in Washington. It was pointed out at them that the known available supply of high-grade phosphate rock in the United States will last only about 50 years. The absolute necessity of utilizing these deposits for the benefit of the farms of the United States was strongly emphasized. The question of how to dispose of these phosphate lands proved exceptionally troublesome to Congress; no decision was arrived at during 1908. Two plans for settling the issue have been proposed. One is that of retaining the land in Government ownership, leasing the phosphate mining rights to individuals; the other is that of restoring them to private ownership subject to distinct provision as to the law under which they may be patented.

Idaho. (By Robert N. Bell.)—Idaho produced 5700 short tons of high-grade phosphate rock during 1908. This came from the property of the San Francisco Chemical Company at Montpelier, Bear Lake county. Recent development of the phosphate deposits of this district tends strongly to confirm the opinion that this field will prove the richest and most important source of phosphate rock ever discovered in the United States. Its area has lately been greatly extended and the Government withdrawals of land known to be phosphate-bearing extend as far north as the Freemont county coalfields, fully 100 miles north of Montpelier.

PHOSPHATE ROCK IN FOREIGN COUNTRIES.

Germany.—During 1908 there was imported into Hamburg a total of 248,641 metric tons of phosphate rock. Of this amount 158,500 tons came from Florida; the remainder was made up chiefly of shipments from Algeria, Gafsa, and Christmas island. The imports into the port of Stettin during 1908 amounted to 201,853 metric tons. Of this amount, 86,361 tons was from the United States; the remainder was made up of shipments from Algeria, Tunis, and South Pacific islands. The 1907 imports into this port from the United States were 56,421 metric tons.

Italy.—The total imports of phosphate rock into Italy during 1908, as reported by U. Vedovelli, Milan, were 567,449 metric tons, of which 420,780 tons came from Africa, and 146,669 tons from the United States. This compares with a total of 430,526 tons imported in 1907. The increase in consumption for 1908 amounted to nearly 32 per cent. Of the phosphate rock imported from the United States, 128,010 tons was Florida land pebble, 1685 tons Florida hard rock, and 16,974 tons Tennessee phosphate. Of the African phosphate imported the bulk came from Gafsa and Kalaa Djerda; Constantine shipments showed an increase.

Japan.—The imports of phosphate rock into Japan during 1908 were 118,263 metric tons, valued at $1,675,479, as compared with 126,585 tons, valued at $1,950,126, in 1907.

Russia.—According to the *American Fertilizer,* Russia has some important deposits of phosphate. Their exploitation, however, is little extended as yet, many of the deposits being badly situated for economic exportation, and not convenient for supplying the Russian manufacturers of superphosphate, of which the number is somewhat limited. One factory is situated at Riga, four in Poland, one at Odessa, and another at St. Petersburg. The Russian phosphate deposits occur in widely separated regions. The quality of the rock is extremely variable,

ranging from that so rich in phosphorus as to be immediately transformed into superphosphate, to others of such small value as to require the most advanced methods of concentration. Geologically, the deposits may be distinguished as follows: (1) The phosphorites of Podolia. These are found between Staraja, Uschtschiza and Mohileu, as well as in the valley of Fschorny Potok, an affluent of the Dniester, district of Chotin, on the Galician frontier. (2) The Wologda phosphorites. These were discovered in 1896 on the banks of the rivers Lyssola and Wisinga, in the district of Ustj-Syssolsk. (3) The phosphorites of the Center. These are found in the governments of Moscow, Wladimir, Kostroma, Jaroslaw, Nijni-Novgorod, and Smolensk.

South Sea Islands.—As development progresses the great importance of the phosphate deposits of the islands of the southern Pacific is becoming manifest. According to American consular reports, it is estimated that there are on Ocean and Pleasant islands, in the Gilbert group, a total of 50,000,000 tons of phosphate rock; from 250,000 to 300,000 tons per year are now being marketed. The deposits are controlled by a London company, and the material is shipped to Japan, Australia, and European points. Phosphate of a good quality has been discovered on three islands in the French colony of Tahiti and dependencies. These islands, Makatea, Matahiva and Niau, lie in the northwestern part of the Tuamotu archipelago of 80 atoll islands. The high-grade deposits in Makatea are estimated at 10,000,000 tons, with many million more tons of inferior quality. An important new source of phosphate rock has also been found on the island of Augaur, to the west of the Caroline islands, and about 600 miles east of the Philippine island of Mindanao. This island is under German rule, and the exploitation of its deposits has been undertaken by a Bremen company. In addition to the islands mentioned above, discoveries of greater or less importance have been made on many other neighboring islands, and indications are that the phosphate industry will assume large proportions within the next few years.

Modern Land-Pebble Phosphate Mining in Florida.[1]

By H. D. Mendenhall.

Overburden.—Removal of overburden is accomplished by hydraulicking where the material is comparatively free from boulders; the steam shovel is used in strata containing large quantities of boulders. In the former method the removal is accomplished by forcing a stream of water against the banks with a hydraulic giant. Obviously, the hydraulic

[1] Digest of an article in *Engineering News*, Oct. 15, 1908.

method, when conditions permit of its use, affords the cheapest and most expeditious means of removing overburden.

Mining.—There is but one method in the mining operations, and that is the simplest and cheapest known—hydraulicking. Throughout the district the matrix in which is found the pebble phosphate is composed almost wholly of soft sand, intermixed with a small amount of yellowish-white clay, easily disintegrated and moved with a hydraulic giant. The method of removal is virtually the same as that for hydraulicking overburden. By a proper manipulation of the giant, impossible to describe and gained only by experience, small ditches in which the disintegrated deposit is washed to the sump hole are kept abreast of the giant. The material passes to the sump hole, whence it is taken up by powerful sand pumps, sent along to similar pumps at the washer and thence to the washing machinery. Each giant can mine on an average about 200 tons of washed rock per 24 hours where the proportion of pebble to matrix is about 20 per cent. The matter of water supply is of great importance; local conditions, such as proximity to a stream or artesian supply, affect the arrangement, and in consequence the variations are infinite. At the property of the Medulla Phosphate Company, at Christina, the amount of water required is 4000 gal. per min. Two artesian wells furnish one-half the amount required; the remainder is obtained through the use of the circulating method; that is, by using the water over and over again. This is accomplished by the use of a dam and a series of pumps.

Washer.—The general design of washers throughout the district is similar, but in details of construction they vary widely. They all maintain the general scheme of pumping the rock-bearing matrix into a separator, where the pebble is separated from the clay balls, large sand rocks and other large débris, passing the rocks through log agitators and finally passing it over or through rinsing screens. In practice, the variations are in the number of sets of logs, number and type of screens, and final method of disposing of the rock. In the land type, as installed at the plant of the Medulla company, the matrix is pumped up into the separator by tandem centrifugal sand pumps. This separator is a cylinder 12 ft. long by 48 in. in diameter, punched "hit and miss" with 1-in. round holes. It is inclined on a slope of 1 : 6 and makes about 12 r.p.m. As none of the pebble phosphate rock is of such a size as to be retained by a 1-in. hole, approximately all of the pebble passes through, the larger clay balls, etc., being retained and worked down into a trough, whence they are carried off by a stream of water continually supplied from pipes. The pebble rock is caught on an adjustible screen and washed down into the logs. Material such as sand, dirt,

etc., that passes through the screen, is carried to the same trough that receives the debris from the separator. The logs in this type of washer are four in number, arranged in pairs, and may be described as simply blades or flukes bolted to 8-inch by 18-ft. cast iron pipe, and caused to rotate in opposite directions by means of spur gears. The logs are spaced 36 in. apart and revolve at 25 r.p.m. From the first pair of logs, the rock, now partially cleaned, passes into a revolving rinsing screen. The debris passing through these screens is caught in a wooden trough and carried away. To facilitate the passage of the rock through the screen, it was found necessary to introduce flights to assist gravity down the slope. From the screen the rock falls into another set of logs, the same in every detail as the first, passes from them into another rinsing screen the same as the last, and thence into the receiving hopper. From the receiving hopper, the now thoroughly cleaned rock is discharged either into a receiving bin or into railroad cars.

For a washer of the dredge type, the machinery down to the discharge of the rock from the last screen is the same as for the land washer above described. In place of having the whole machinery stationary, it is constructed on a barge and a conveyer transfers the washed rock from the receiving hopper to the cars on shore.

Mill.—From the washing plant the rock goes to the central wet storage and drying plant. The washed rock is usually run up on an elevated track and discharged into a receiving hopper back of the wet bins. It goes from here into elevator boots, whence it is taken by the elevators up into the wet bins. The rock is drawn from the bins by chutes leading to another elevator boot, whence it is elevated to another hopper that feeds directly into the two dryers. The dryers are of the revolving cylinder type with diaphragms, flights and showering shelves, similar to, although smaller than, the ones used in cement manufacturing. They are 30 ft. long by 52 in. inside diameter and make 10 r.p.m. This type of dryer has a capacity of about 150 tons per 12 hours. The furnaces have no features out of the ordinary other than that they are designed to burn oil, coal, or wood, by making simple changes. From the dryers the rock, now thoroughly dried, passes through chutes to a concrete boot, whence it is elevated to the top floor and discharged into a conveyer to be taken to the dry storage bins. After the rock has been thoroughly washed and dried, nothing remains for it but shipment.

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PHOSPHORUS.

Phosphorus is produced in the United States by the General Chemical Company, the American Phosphorus Company, and a few other concerns. This product is used largely in the manufacture of lucifer matches and in the preparation of certain alloys, especially phosphor tin and phosphor copper. Phosphor tin and phosphor copper are both employed for the manufacture of phosphor bronze. At the present time phosphor tin and phosphor copper, containing 15 or more per cent. of phosphorus is prepared. A few years ago, 5 per cent. of phosphorus was the limit that could be introduced into copper and tin. The domestic trade in phosphor copper and phosphor tin amounts to as much as $1,000,000 per annum; the business in phosphor bronze is much larger. That alloy is employed to an important extent as a bearing metal.

In the recent tariff hearings, the Damascus Bronze Company of Pittsburg, Penn., submitted the following figures showing the cost of making phosphor copper. It is necessary to introduce 20 per cent. of phosphorus in order to secure a manufactured product containing 15 per cent. phosphorus and the balance copper; it is necessary to introduce 13 per cent. phosphorus in order to secure a manufactured product containing 10 per cent. phosphorus, and it is necessary to introduce 6 per cent. in order to secure a manufactured product containing 5 per cent. of phophorus.

15 PER CENT. PHOSPHOR COPPER.

American cost per 100 lb:		European cost per 100 lb:	
20 per cent. phosphorus, 35c.	$7.00	20 per cent. phosphorus, 20c.	$4.00
80 per cent. copper, 14c.	11.20	80 per cent. copper, 14c.	11.20
	$18.20		$15.20
Loss in smelting	2.91	Loss in smelting	2.43
Crucible and fuel	.60	Crucible and fuel	.42
Labor	1.10	Labor	.70
Total	$22.81	Total	$18.75

The price of phosphorus in Europe is 20 to 22c. per lb. In the United States it is 35 to 40c. In 1908 it could be freely obtained at 35 cents.

PLATINUM.

The production of platinum in the United States in 1908 was 510 oz. According to the U. S. Geological Survey, the production in 1907 was 357 oz. Our report for 1908 is based upon the answers of all of the refiners in the United States to the inquiry as to the amount of crude platinum of domestic origin refined by them during the year. However, it is impossible to pronounce definitely that the metal reported was entirely of domestic origin.

Most of the domestic platinum passes through the U. S. mints. These do not make a business of refining platinum, but considerable of that metal is recovered as a by-product in the electrolytic refining of gold and silver bullion. The platinum is recovered in the slime, principally in the gold cells. There is no way of identifying its origin. It is associated with the gold bullion in such small quantities that its presence is not always noted; and inasmuch as the crude bullion comes into the mints from foreign as well as from domestic mines, it is probable that a portion of the platinum recovered is of foreign origin. The platinum is recovered at the mints in the form of sponge, furnaces for reducing it to bars not yet having been installed. Between 750 and 1000 oz. of platinum is recovered per annum. In 1908 only 500 oz. were sold, the price received being $16.32 per ounce.

The refiners of platinum in the United States are the following:

Baker & Co., Inc., Newark, N. J.
Croselmire & Ackor, Newark, N. J.
S. S. White Dental Mfg. Co., Prince's Bay, N. Y.
Welsbach Company, Gloucester, N. J.
J. Bishop & Co., Malvern, Penn.
American Platinum Works, Newark, N. J.
Roessler & Hasslacher Chemical Company, Perth Amboy, N. J.

There are also some concerns which refine scrap platinum in a small way. The chief supply of the refiners is Russian crude metal, but small quantities of American crude are obtained from time to time. Domestic crude platinum is commonly bought on the quotation for scrap platinum. One of the refiners reported as follows: "We are in hopes that the day is not far distant when we may be able to obtain a dependable domestic supply, inasmuch as the manipulation of the market by foreign interests is decidedly unsatisfactory. That miners and prospectors are more than ever alive to the situation is evident by the frequent inquiries made of us and samples of domestic ore sent to us for examination, as a result of

which we hope in the near future to be able to report materially larger purchases of domestic ore."

The business of platinum refining in the United States is, of course, much more important than the statistics of domestic production indicate, inasmuch as a great deal of foreign crude platinum is refined here, while a large amount of domestic scrap is refined or remelted. Refined platinum is imported in the form of ingots about $\frac{5}{16}$ in. thick, weighing about 100 troy ounces.

STATISTICS OF PLATINUM IN THE UNITED STATES.

Year.	Production. (a)		Imports.			Consumption.
			Unmanufactured.		Manufactured	
	Troy Oz.	Value.	Troy Oz.	Value.	Value.	Value.
1896	163	$944	83,080	$926,678	$106,338	$1,033,960
1897	150	900	83,080	960,299	43,921	1,005,120
1898	225	3,375	101,018	1,178,142	52,283	1,233,800
1899	300	1,800	187,778	1,462,157	55,753	1,539,710
1900	400	2,500	118,919	1,728,777	36,714	1,767,991
1901	1,408	27,526	85,438	1,673,713	24,482	1,725,721
1902	94	1,314	105,450	1,950,362	37,618	1,989,794
1903	110	2,080	114,521	1,921,772	135,889	2,059,741
1904	200	4,160	103,802	1,812,242	106,636	1,920,478
1905	318	5,320	104,196	1,985,107	188,156	2,176,263
1906	1,439	45,189	137,556	3,601,021	187,639	3,797,460
1907	357	10,589	74,208	2,509,926	175,651	2,696,166
1908	510	11,655	49,868	1,096,615	134,119	1,108,270

(a) Statistics of the U. S. Geological Survey, except for 1908.

Market.—The imports of platinum into the United States were 74,208 oz. in 1907, and 49,868 oz. in 1908; a decrease of 24,340 oz. This indicates a large decline in demand, as was to be expected from the general depression of business. It must be noted that a large part of the platinum sold is not consumed in the ordinary sense of the word, but returns to dealers in the form of scrap, practically uninjured, and ready for use as new metal.

At the close of 1907 refined platinum was quoted in New York at $26 per oz.; during that year it had been as high as $38. During 1908 the price receded from $26 to about $22 in July. For two or three months the market was disturbed by offers of the metal at $18 and even as low as $17; the source of these offers, or at least of the supply of metal, being involved in some mystery. These offers gradually ceased, and in November and December of the year prices recovered. At the close of the year dealers quoted $24@24.50 for refined platinum; $26@26.50 for hard, which contains some iridium; and $20@22 for good scrap.

The market for Russian platinum is controlled by a syndicate of Paris and London refiners, whose usual course is to depress prices,

make long contracts with the Ural producers, and then exact from consumers all they will pay. This course is made possible by the limited quantity of the output, and the fact that there is no substitute for the metal in many of its uses, although for some purposes gold is employed when the price for platinum rises greatly in excess of $20 per ounce.

AVERAGE MONTHLY PRICES OF PLATINUM AT NEW YORK.
(In dollars per troy ounce.)

	1906		1907		1908	
	Ordinary	Scrap.	Ordinary	Scrap.	Ordinary	Scrap.
January	20.50	16.00	38.00	31.50	25.50	17.00
February	25.00	19.00	38.00	31.75	25.50	17.00
March	25.00	19.00	37.00	30.75	25.50	17.00
April	25.00	19.00	32.50	24.75	23.50	16.00
May	25.00	19.00	29.50	21.125	22.50	17.00
June	25.40	19.75	26.20	20.30	23.50	16.00
July	26.00	21.50	26.75	21.437	20.00	15.00
August	26.00	21.50	28.125	22.625	18.75	15.25
September	32.10	24.00	28.70	23.30	20.00	16.00
October	33.00	25.50	27.125	21.25	21.50	18.00
November	35.50	28.38	26.312	18.937	24.00	19.00
December	38.00	31.25	26.00	17.62	24.00	20.50
Year	28.04	21.99	28.183	22.310	22.854	16.94

Colombia.—The platinum output of Colombia is second only to that of Russia. This precious metal, washed from the gravels of the Choco, is always found mixed with gold, sometimes one or the other metal predominating. Although platinum occurs to a small extent in other parts of the Choco, its main sources are the Platina and Condoto rivers, with a few tributaries, and some of the streams running into the Cajon, all being tributary to the San Juan river. The metal is also obtained from the Agua Clara river and certain sections of the Berbera river, both of which flow into the Atrato, and from the Certegui, which is the main affluent to the Quito river.

According to an article in *La Colombie,* for June, 1908, the crude platinum of the Choco region contains from 80 to 85 per cent. of the metal. The chief platinum bearing zone is the district drained by the San Juan, l'Iro, the Condoto and the Tamana rivers, but many other streams in the region show platinum in their sands. The placers of the Condoto are probably a very large deposit of the metal. The government has granted a concession for the exploitation of the placers of the Condoto and the Andagueda, according to the terms of which the government is to receive 10 per cent. of the net profits. The mines of Dabaibe, in the upper Andagueda have been exploited with great success, and an English company is now making arrangements to exploit the mines of Certegui.

of the producers of platinum in the Ural; and (3) the low price for this metal during the major part of the year. The statistics of production are given in the accompanying tables, from which it appears that the yield of platinum decreased especially in the South Verhotursky district, where 97,879 oz. were extracted in 1908, against 121,665 oz. in 1907. The Platinum Producing Company, Ltd., which obtained 89,858 oz. in 1907, reported only 74,072 oz. in 1908. The mines of the Demidoff estate (the Nijni-Tagilsk mines) which produced 30,255 oz. in 1907, yielded only 21,381 in 1908.

PRODUCTION OF PLATINUM IN THE URAL.
(In troy ounces)

District.	1906	1907	1908
South Verhotursky	127,461	121,665	97,879
North Verhotursky	11,156	7,554	13,043
Perm	40,149	36,062	36,776
Cherdinsky	4,450	4,804	7,470
South Ekaterinburg	2,330	1,869	1,837
Total	185,546	172,064	157,005

AVERAGE PRICE PER OUNCE TROY FOR PLATINUM INGOT.

Year.	£	s.	d.	Year.	£	s.	d.	Year.	£	s.	d.
1874	1	5	2	1900	3	19	9	1906 (Jan.).	4	15	2
1888	1	13	8	1901	4	1	11	1906 (Oct.).	7	19	8
1890	1	15	8	1902	4	4	0	1907 (Jan.).	7	0	0
1893	1	17	9	1903	4	6	1	1907 (Feb.).	6	19	6
1895	2	2	0	1904	4	8	1	1907 (Mch.).	6	19	0
1898	3	13	6	1905	4	10	4	1907 (Apr.).	6	18	8

PRICE FOR CRUDE PLATINUM OF 82 PER CENT. IN THE URALS.

Year.	Rubles. Per Pood.	£	s.	d. Per Oz.	Year.	Rubles. Per Pood.	£	s.	d. Per Oz.	Year.	Rubles. Per Pood.	£	s.	d. Per Oz.
1874	4,800	0	19	0	1899	7,000	1	7	8	1906 (Jan.).	22,000	4	7	9
1888	6,000	1	4	0	1901	16,200	3	3	9	1906 (Oct.).	34,000	6	15	8
1890	6,200	1	4	8	1902	17,300	3	9	4	1907 (Jan.).	30,000	5	13	11
1893	6,500	1	5	10	1903	18,500	3	13	3	1907 (Feb.).	29,000	5	15	8
1895	6,600	1	6	7	1904	21,000	4	3	8	1907 (Mch.).	28,000	5	12	0
1898	6,800	1	7	0	1905	22,000	4	7	9	1907 (Apr.).	27,000	5	8	0

With respect to the technical side, 1908 was marked by a tendency on the part of the platinum producers to begin dredging operations, a tendency which is certainly connected with an exhaustion of the fields and the necessity to reduce the cost of production. Two dredges of English manufacture were furnished in 1908 to the Demidoff estate, and dredges were also installed by other platinum mining undertakings.

With respect to the economics of the industry, 1908 was a tragic year, especially for the smaller concerns. For many months the price

was as low as 3.7 rubles per zolotnik, i.e., 14,200 rubles per pood of crude platinum, containing 83 per cent. of the pure metal, which price was absolutely unprofitable. Having insufficient capital to be able to await higher prices, the smaller producers were obliged to cease operations, and were inspired by their difficulties to apply to the Government for help. The petition of the Ural platinum miners can be summarized as follows: (1) The Government is desired to take measures to raise the price for platinum by buying it in at a fixed price; and (2) loans at a low rate of interest from the Imperial Bank are requested upon the security of the crude platinum produced. In order to obtain such assistance, the platinum miners held several meetings in 1908. The Government appears to be disposed to help them and has decided to form a committee for the regulation of the platinum industry, wherefore, we may expect in 1909 the establishment of some important measures.

Osmiridium.—The *Torg. Prom. Gazeta* says that in 1908 osmiridium was produced as follows: In the Miask district, on the mines of the Miask Gold Company, to the extent of 55.978 oz.; the Western Ekaterinburg, on the properties of the Kysshtim mining works, made no return in 1908. Thus in 1908 the quantity of osmiridium produced in the Urals was 16.327 oz. less than in the year 1907.

According to E. de Haupick the annual production of osmiridium and iridosmium in the Ural is about 6000 oz., the major portion finding its way secretly to market and not being accounted in the official statistics.

The Russian Platinum Industry.[1]

By E. de Haupick.

The two principal centers of platinum placers are Blagodat, on the Asiatic slope, and Nijni-Tagilsk, on the European slope of the Ural mountains. The platinum area is about 80 miles. In addition to being found at these placers, platinum occurs in many Ural gold placers, and also in the Altai mountains (East Siberia). The presence of platinum on the far north of the Ural at the Kandalak Gulf (White Sea) was foretold by A. P. Karpinsky and E. S. Theodorof.

In my preceding articles on this subject, I have mentioned that the Russian official figures concerning platinum extraction are inexact, as they do not show the large quantities of the metal regularly stolen by the miners. The total output since 1890 is given in the accompanying table.

Compared with 1907 the production of platinum decreased in 1908 to the extent of 15,966 oz. (according to official figures). The decrease

[1] Abstract of an article in London *Min. Journ.*, May 8, 1909.

PRODUCTION OF PLATINUM IN RUSSIA.

Year.	Official.	Actual.	Year.	Official.	Actual.	Year.	Official.	Actual.
	oz.	oz.		oz.	oz.		oz.	oz.
1891.....	136,036	235,000	1897.....	179,879	395,200	1903.....	192,976	276,000
1892.....	146,806	260,000	1898.....	193,213	203,100	1904.....	161,950	290,120
1893.....	163,757	285,850	1899.....	191,464	380,900	1905.....	167,950	200,450
1894.....	167,268	203,250	1900.....	163,060	212,500	1906.....	185,492	210,318
1895.....	141,757	290,900	1901.....	203,257	315,200	1907.....	172,758	310,000
1896.....	158,326	200,000	1902.....	197,024	380,806	1908.....	157,005	250,000

was most marked in South Verhotur mining district, where the production fell from 121,558 oz. in 1907 to 97,750 oz. in 1908. The falling off was thus about 24,808 oz., especially on the Nijni-Turinski placers, the property of the Société Industrielle du Platine, the production of which in 1908 was 73,980 oz., against 89,760 oz. in 1907. The Nijni-Tagilsk placers produced 21,367 oz. in 1908 and 28,063 oz. in 1907. The North Verhotur mining district showed an increase in production in 1908 of 4416 oz.; this increase may be accounted for by the adoption of a new dredger on Lialensk placers and by the increase in the rate of working on Nicola-Pavdinsk placers, the property of Vorobieff. The Permski mining district, comprising the Count Shouvaloff placers, produced only 1838 oz. in 1908. Consequently, the total platinum output of the Ural, which reached its highest figure of 203,257 oz. before the Russo-Japanese war in 1901, was in 1908 only 172,758 oz., or less by 10,200 oz. than the average produceion for the last 10 years (182,958 oz.).

I give all these official figures as a matter of course, but according to private information received by me, the total platinum output of the Ural for 1908 was about 250,000 oz.

The production of the Nijni-Tagilsk district is twice as great as that of the Blagodat, and the quality of the ore is also better, as was shown by chemical analysis. The richest deposits are the Krestovozdvijensky placers of Shouvaloff, with an annual output of 36,000 oz. obtained by washing 600,000 tons of alluvium; next in order are the Nijni-Tagilsk placers of Prince Demidof-San-Donato with 33,000 oz. from 1,000,000 tons of alluvium. The Vladimirsky placer of G. J. Nekloodof on the river Toora, rented by the Société Industrielle du Platine, gives per annum 8700 oz. by washing 100,400 tons of alluvium; the Ivano Predtetshensky placers near Nijni-Turinsk, the property of M. P. Protopopof, rented also by the Société Industrielle du Platine, give 6443 oz., obtained from 80,000 tons of alluvium; placers Issakievsky, also near Nijni-Turinsk, the property of M. Shomine, also rented by the Société Industrielle du Platine, produce 6050 oz. from 70,000 tons of alluvium; and the Marinsky placers, the sole property of the Société Industrielle du Platine, supply 5826 oz. from 100,000 tons of alluvium.

The actual average yield is at present about 0.09 oz. of crude platinum per ton of gravel, with the exception of the few placers mentioned above. In olden times the placers were richer in precious metals. From 1825 to 1829 the average extraction was 2.7 oz. or more; in 1829-1838, 0.5; in 1838-1850, 0.4; in 1850-1883, 0.33; and in 1833-1894, 0.1.

The four principal platinum markets—England, United States of America, France and Germany—consume annually about 363,000 oz. This figure includes about 33 per cent. of platinum scrap, or about 123,000 oz., which is annually thrown back on the market. The annual consumption of crude platinum may be estimated at 240,000 oz. Apart from the Ural, the platinum annual production is: Colombia, 8800 oz.; United States, 530 oz.; New South Wales, 530 oz.; Sumatra and Borneo, 500 oz.; total, 10,060 oz. The Ural furnishes to the world's market about 230,000 oz. of crude platinum per annum, while Russian official statistics show 82,500 oz. of platinum exported annually. It is evident, therefore, that about 147,000 oz., or 61 per cent. of the world's consumption of crude platinum, finds its way to the market secretly.

The crude platinum is sent immediately after its extraction to the Ural assaying office at Ekaterinburg, which gives a certificate of analysis, and at the same time deducts 3 to 4 per cent. in kind as a State contribution. This last contribution is sold by auction at Ekaterinburg. The accompanying table gives the quantities and prices of the above-mentioned auction sales.

AUCTION SALES OF CRUDE PLATINUM AT EKATERINBURG.

Year.	Oz.	Per oz.	Year.	Oz.	Per oz.	Year.	Oz.	Per oz.
		£ s. d.			£ s. d.			£ s. d.
1893.....	2,700	1 5 10	1899.....	9,000	1 7 8	1905.....	4,950	3 8 6
1894.....	5,400	1 6 0	1900.....	4,900	2 8 6	1906.....	6,120	4 0 0
1895.....	4,300	1 6 7	1901.....	3,850	2 8 6	1907.....	5,250	4 2 0
1896.....	2,800	1 6 8	1902.....	5,300	3 0 6	1908.....	5,750	4 0 0
1897.....	7,000	1 6 9	1903.....	4,750	3 3 0			
1898.....	5,050	1 7 0	1904.....	6,900	3 5 6			

Concerning platinum prices, I must remark that the Ural crude platinum extracted in the Blagodat mining district on the rivers Iss and Tura contains more platinum (about 85 per cent.), and is valued higher than the Nijni-Tagilsk platinum (about 74 per cent.). The prices per ounce troy for crude platinum on the basis of 82 per cent. in the Ural, and for refined platinum ingots on the basis of 100 per cent. as quoted in Europe, were as given in the accompanying table.

Except for 2000 to 3000 oz. refined in St. Petersburg by Kolbe & Lindfors and the Tentelef Chemical Manufacturing Company, the whole platinum supply is refined abroad; in London by Johnson, Matthey & Co.,

PRICES FOR PLATINUM.

Year.	Ural Price for Crude.			European Price for Refined.			Year.	Ural Price for Crude.			European Price for Refined.			Year.	Ural Price for Crude.			European Price for Refined.		
	£	s.	d.	£	s.	d.		£	s.	d.	£	s.	d.		£	s.	d.	£	s.	d.
1819...	0	0	10	.....			1888...	1	4	0	1	13	8	1903....	3	13	3	4	6	1
1822...	0	18	1	.....			1890...	1	4	8	1	15	0	1904....	4	3	8	4	8	1
1823...	1	6	1	.....			1893...	1	5	10	1	17	9	1905....	4	7	9	4	10	4
1824..	1	12	2	.....			1895...	1	6	7	2	2	0	1906(Ja)	4	7	9	4	15	2
1825...	2	2	0	.....			1898...	1	7	8	3	13	6	1906(Oc)	6	15	8	7	19	8
1827...	1	12	2	.....			1900...	.....			3	19	9	1907....	5	12	0	6	18	6
1860-70	0	6	8	.....			1901...	3	3	0	4	1	11	1908....	4	10	2	5	2	6
1874...	0	19	0	1	5	2	1902...	3	9	4	4	4	0							

in Paris by Desmoutis, Lemaire & Co., in Germany (Hanau) by Heraus & Co. The greater part of crude platinum is exported from Russia by parcels post and not as goods. Often the despatch is made from the Ural to the Russian frontier terminus, east or west, and is carried thence across the frontier in secret. This fully explains why the official statistics do not show the true figures. During a long sojourn in the United States I often saw the Ural contraband crude platinum finding its way to America by Siberia, Vladivostock, and San Francisco. It should be noted that neither in the Ural nor anywhere in Russia are there important platinum stocks. The exploitation of platinum is being carried out on the 140 to 150 placers, and it is sold by contract before its extraction. The proprietors of placers are always short of money, and are unable to accumulate considerable reserves of the metal.

Dredging for Platinum in the Urals.

By L. Tovey.

Platinum mining in the Urals is carried on most extensively in two fields, viz., the Nijni-Tagilsk district on the river Martjan, a secondary tributary to the Tchusowaja, in the estate of Demidoff, Prince San Donato; and on the river Iss tributary to the Tura falling into the Tobol. The Tura itself is a platiniferous stream below the mouth of the Iss. Other districts of inferior importance are being opened to the north, on the eastern slope of the Urals, but as yet mining has not attained any serious scale there. However, there is reason to expect that in the near future a good many of the new districts will come into importance. All the platinum is produced from gravel deposits; the only primary deposit is in a patch of olivine rock with chromic iron near the principal camp of the Demidoff mines, but this is not worked. However, the administration of the Demidoff estate is beginning to realize the importance of finding the source of the rich placers, which are to a certain degree exhausted, and is trying to solve this question.

The Demidoff Placers.—These are worked on a very large scale. All along the Martjan, for a distance of about 15 *versts,* platiniferous gravel

is washed on stationary plants of the common Ural type, employing the *botchka,* or conical revolving screen, the *tchasha,* or puddling machine with a bottom of perforated iron plates, or the *boronka.* Details of these operations may be found in *Trans.* A. I. M. E., Vol. XXIX, in a paper by C. W. Purington, and in the *Leoben & Pribram Jahrbuch* for 1902 (Bd. 4, Heft. 3) in a paper by L. St. Rainer. The metal from the Martjan is dark and magnetic. The output in 1907 was about 28,750 ounces.

The richer portions of the deposits have been worked out. Some parts are at present worked by small groups, sometimes by families of workmen, or *starateli,* who get from the owners small patches of the gravel deposit and wash it by a kind of long tom. The water is usually pumped from the stream or pond. The work of washing is usually done by women while the men break the gravel from the bank. As this kind of work can be carried on only where water is not abundant in the deposit, a good part of the deposit is left unworked. To work the remaining gravels the best means is a dredge, and the administration of the Demidoff estate has been negotiating about building a powerful dredge.

The Shouvaloff Placers.—More important than the Martjan deposits are those on the streams Iss and the Wyia, tributaries to the Tura in the Goroblagodatski district. The upper part of the Iss with its tributaries belongs to Count Shouvaloff and is an important producer of platinum. The gravels are formed of the different varieties of gabbro, containing diallage and olivine, which crop out on the upper Iss in two high mountains, Saranoraja and Katchanar, in large masses. N. Wyssotzki calls special attention to two minor hills, the Svetli and Veressovoi Bor, containing large masses of dunite, the source of the precious metal. The main deposit is in the bed and banks of the Iss and its tributaries, and is worked by open cuts and stationary washing plants, and also by two Werf Conrad dredges with Perret patent revolving screens. The output of the whole estate in 1907 was between 75 and 90 poods, or about 39,560 to 47,500 ounces.

The Werf Conrad boats, built by a firm of Haarlem, Holland, have, on the whole, been successful, but still they do not form the most important part of the washing plant. Their output in 1907 was 3757 oz., or less than 10 per cent. of the whole. This was obtained from 260,000 cu.yd. of gravel, measured in the bank. The operation of the dredges is satisfactory. The dredging conditions are more or less favorable, the depth of dredging being about 17 ft., and the deposit being uniform and without large boulders. The loss of platinum in the tailings is reported to be quite insignificant.

The results of the dredges in 1907 were very good. Dredge No. 1 worked 114 days at an average of 16.82 hours per day, handling 1028 cu.yd. of gravel and producing about 15.5 oz. of platinum. The performance for the year was 117,268 cu.yd. of gravel and about 1735 oz. of platinum. The work of dredge No. 2 was still more satisfactory. It worked for 135 days at a daily average of 18.26 hours, handling 1058 cu.yd. of gravel and producing 24.15 oz. of platinum. The output for the year was 2022 oz. platinum.

Dredging is usually begun about the last of April or the first of May, and is continued to early October. No doubt the dredging season could be made longer if trials were made in that direction. At present the dredges represent only a small part of the operations and are closed down with the rest of the plants when washing the gravel from the open cuts becomes difficult. In other mines lower down the Iss and the Tura the working season is longer, for the climate is milder.

Compagnie Industrielle du Platine.—The next property down the Iss is that of the Compagnie Industrielle du Platine, the largest producer of platinum in the world. Besides the properties on the Iss it owns claims on the Wyia, another tributary to the Tura, and a good many claims on the Tura, itself below the mouth of the Iss. In October, 1907, it bought up the property of the Turinski Platinum Company and in the spring of 1908 a controlling interest in the company, "Platina," formed by Moscow banks. The average production of the Compagnie Industrielle has been between 125 and 150 poods.

In 1907 the placers were worked by open cuts with manual labor, the haulage being done in cars drawn by horses. The washing plants included five *butaras* (conical screens) eight *boronkas* and two sluices. These washing plants have been described by Purington and Rainer. The only alteration introduced of late has been a combination of the *butara* and *boronka,* the coarse gravel and clay passing from the revolving screen to the *boronka,* a kind of concave screen bent to about a quarter of a circle, on which the stones and clay are rubbed by a sort of double rake run from the main engine. The clay and fine sand separated from the boulders pass to a small table under the *boronka* screen. The main revolving screen (*butara*) is usually 16 to 19 ft. long with diameters of 5½ and 7 ft. The holes in the iron plates of the screen are ½, ⅝ and ¾ in. and the plates are $\frac{5}{16}$ in. thick. In one of the plants there is a combination of this screen with the Perret bar screen which does the preliminary washing.

Inasmuch as the present methods of working are rather expensive, only the richer portions of the deposit can stand the costs. When these are worked out the company will be compelled to work the poorer parts

of the river by more efficient mechanical methods. Experiments with this end in view have been carried on since 1901 and 1902, when M. Mongolfier put up two dredges which are still working on the Troitzki and Anno Tonifowski claims. According to Saytzeff, the overburden here is about 14 ft. thick, and the pay gravel from 4½ to 6 ft.; the average depth of dredging is about 16 to 17 ft. These natural conditions caused the company to adopt a special method of mining, i.e., the employment of two dredges, one to strip the overburden, the second to wash the gravel. As the overburden is about three times as thick as the pay gravel, the capacity of the stripping dredge is about three times that of the washing dredge. The dredges were not specially constructed for mining purposes, but were bought at the sale of the Kiel canal excavating machinery; this accounts for some of the defects of the machines.

The stripping dredge has buckets of more than 9 cu.ft., running at a speed of 7 to 10 buckets per minute. The capacity of the dredge is 2033 cu.yd. per 24 hours. The ground excavated is carried by a conveying belt to the side of the cut where it is dumped. The whole of the placer is worked out by successive cuts parallel to each other. After the stripping dredge has opened the pay gravel in the first cut the washing dredge follows and takes the gravel to the bedrock. In the meantime the stripping dredge takes the second cut and by means of the conveying belt dumps the overburden into the cut made by the washing dredge. As the conveying belt is perpendicular to the direction of the bucket ladder in both dredges, the dumps are more or less uniform. This work by parallel cuts suits the local conditions, for the deposit is very wide, in some places more than 1000 ft. The washing dredge is much smaller, the buckets having a capacity of about 4½ cu.ft., running at a speed of 7 to 10 buckets per minute. Each of the dredges requires 11 men per shift.

The large number of men is needed because of the antiquated construction of the boat, especially the winches, the operation of which requires three men besides the dredge master. Two men are in charge of the conveying belt, three men at the engines and boiler, one man at the ladder, and one helper for handling the fuel. Such work can be carried on only with the low labor prices prevailing in the Ural, not exceeding 50c. for laborers and $1.25 to $1.50 for skilled workmen. The average daily capacity of the dredge for the 206 working days of the season of 1907 was about 297 cu.yd. The total production for 1907 was about 61,449 cu.yd. of gravel and about 1230 oz. of platinum. Such work is possible only when the prices for platinum are as high as they were in 1907, and is quite unsatisfactory when compared not only with American practice, but also with that of the other Russian dredges.

This unfortunate experience warned the managers against weak and inadequate machinery, and they decided to build the next dredge according to the local requirements. A new 4-ft. dredge began operations in July, 1907, and during the trial days, which lasted, with considerable intervals for alterations, to the close of the season, it worked 16,556 cu.yd. of gravel, recovering about 145 oz. of platinum. A good part of the operations was stripping work. The Valerianowski claim on the Wyia on which this dredge is working, was chosen for the test with good judgment; the gravel is easy to wash and without large boulders, and the whole depth of the cut to the bedrock is about 16 to 17 ft. The bedrock is soft and easily excavated by the buckets. The company intends to build more dredges, and has increased its staff of mechanical engineers who are expected to build the dredges at the mines.

The third dredge owned by the company is a small 1½-cu.ft. prospecting boat, which worked in 1907 on the river Tura to prospect the claims intended to be worked later by more powerful dredges. It has in 10 hours' daily work the capacity of about 74 cu.yd. The production for 1907 was 16,207 cu.yd. and 342 oz. of platinum.

The nearest neighbor of the small dredge just described is a 3½-ft. boat, the strongest on the Tura river, and built after the Bucyrus type, with some alterations introduced according to the conditions of the Jerusalimski mine which it works. These modifications have not, however, been very successful.

Late in the autumn of 1907 the Poutiloff works added to the successful gold-dredging boat on the Irdel, two very good 5½-ft. dredges to work in the north of the Ural.

POTASSIUM SALTS.

By Albert H. Fay.

Germany continues to be the principal producer of potassium salts. The accompanying tables give the production of Germany and the exports from India since 1896. The exports from India showed a slight increase in tonnage for 1907, and at the same time the net value per ton was less than it had been since 1904. There are about 50,000 laborers employed in this industry alone in India. The province of Behar is the principal center.

PRODUCTION OF POTASSIUM SALTS IN GERMANY. (a)
(In metric tons and dollars; 1 mark=$0.238.)

Year.	Kainit.		Potassium. Chloride		Potassium. Sulphate.		Potassium Magnesium Sulphate.		Other Salts. of Potassium.	
	Quantity.	Value.	Quantity.	Value.	Quantity.	Value.	Quantity.	Value.	Quantity.	Value.
		$		$		$		$		$
1896	856,290	2,989,736	174,515	5,718,559	19,682	813,381	4,623	85,977	902,707	2,964,750
1897	992,389	3,486,007	168,001	5,764,423	13,774	565,720	7,812	149,079	953,798	3,030,143
1898	1,103,643	3,835,856	191,347	6,380,220	18,853	763,397	13,982	259,485	1,105,212	3,576,628
1899	1,108,159	3,838,250	207,506	6,801,250	26,103	1,027,500	9,765	195,000	1,384,972	4,202,000
1900	1,178,527	4,134,000	271,512	8,793,750	33,853	1,249,250	15,368	280,500	1,874,346	5,643,750
1901	1,500,748	4,327,250	282,750	8,782,250	27,304	1,460,000	15,612	286,500	2,036,326	5,443,250
1902	1,322,633	4,571,980	267,512	7,507,710	28,279	1,079,092	18,147	334,390	1,962,384	4,949,448
1903	1,557,243	5,208,154	280,248	8,125,320	36,674	1,389,444	23,631	441,252	2,073,720	4,903,478
1904	1,905,893	6,322,470	297,238	8,425,676	43,959	1,664,572	29,285	545,972	2,179,471	5,305,972
1905	2,387,643	7,976,808	373,177	10,580,528	47,994	1,804,040	34,222	614,754	2,655,845	6,396,250
1906	2,720,594	8,918,574	403,387	11,034,632	54,490	2,032,520	35,211	644,028	2,821,073	6,538,336
1907	2,624,412	8,579,206	473,138	12,639,704	60,292	2,216,494	33,368	631,652	3,124,955	7,314,930
1908	2,589,804	.....	508,622	.....	55,755	.....	33,149	.....	3,500,635	.....

(a) From *Vierteljahrshefte zur Statistik des Deutschen Reichs.*

EXPORTS OF SALTPETER FROM INDIA. (a)
(In tons of 2000 lb.)

Year.	Quantity.	Value.	Value per 100 lb.	Year.	Quantity.	Value.	Value per 100 lb.
1898.....	18,263	$1,164,480	$3.06	1903.....	23,105	$1,450,980	$3.14
1899.....	19,870	1,281,050	3.64	1904.....	21,894	1,331,745	3.04
1900.....	17,432	1,471,245	4.05	1905.....	17,535	1,178,615	3.36
1901.....	17,721	1,189,400	3.22	1906.....	19,446	1,352,735	3.48
1902.....	21,882	1,359,335	3.11	1907.....	20,873	1,297,221	3.06

(a) From "Mineral Production of India," by T. H. Holland, Government Geologist.

German Potash Mines.

Six new potash mines—Deutschland, Siegfried, Immenrode, Neubleicherode, Hotorf and Hansa-Silberberg—are reported to be now sufficiently developed to join the "Kalisyndikat" and of 21 other mines,

the majority have passed the early stages of shaft sinking, some of them being down into the saline formation and near the potash beds. The existence of the potash deposits has also been proved at 16 other properties.

German Potash Syndicate.—At a recent meeting of the syndicate, the representatives of a number of the companies expressed approval of the proposals at the council with regard to coming to an understanding with the works still outside the syndicate; and agreed not to sell any of their output, except through the syndicate, until the expiration of the present agreement in June, 1909. The outsiders are to be invited to take part in the future negotiations for a renewal of the syndicate. In connection with the proposed change of domicile, a report of the committee states that the Duchy of Anhalt has offered to exempt the syndicate from taxation, a concession that is not obtainable from Prussia. Some of the members are of opinion, however, that Berlin is the only suitable headquarters, should any change be decided upon.

Upper Alsace.—Borings at Niederbruck in Upper Alsace tapped an upper layer of potash salts, 154 m. thick, at a depth of 358 m. The lower salt-bearing stratum has a total thickness of 327 m. and was encountered at 620 m. Altogether, 103 deep boreholes were put down, 17 of which struck the potash deposits; the others were stopped on first reaching the salt. The proved area measures about 80 square miles, bounded on the north by a fault running between Ungersheim and Regishelm, and on the west by the Vosges, while toward the south and east the deposits thin out. A shaft 600 m. deep has been sunk near Wittelsheim.

The Refined Salt Market During 1908.[1]

Carbonate.—The demand was irregular and the situation marked by intervals of activity and dullness. At the beginning of 1908 sales were made at 3¾@4½c. for 80 to 85 per cent. calcined; 4⅜@4¾c. for 96 to 98 per cent. calcined, and 4¼@4⅝c. for 80 to 85 per cent. hydrated, according to quantity, seller and terms of sale. This price was maintained throughout the year.

Caustic Potash.—Manufacturing consumers took a fair average volume of goods and importations generally met with active distribution in the soap, paper and textile industries. At the beginning of 1908 sales were made on the basis of 6⅛c., according to grade, quantity, process and sellers. This level remained unchanged until May, when the prices were reduced to 3⅞c., which proved to be a stationary level the remainder of the year.

[1] From the *Oil, Paint and Drug Reporter*, Feb. 22, 1909.

Chlorate.—The dullness of the textile industry during the first six months of 1908 had a depressing influence on reducing stocks of potassium chlorate, but as conditions improved with the advent of summer, the early losses were largely offset. The demand during 1908 was moderate. In the latter part of October the manufacturers announced prices for 1909 as follows: Chlorate, f.o.b. works, according to quantity, 8½@8¾c. The prices for 1908 showed little fluctuation, ranging from 8¾ to 9½c.

Chloride.—Early in January large shipments arrived to fill old contracts which gave the market an active appearance. Business during the following month was somewhat stagnant. Prices were steady on spot at $1.95@1.98 per 100 lb. on the basis of 80 per cent. During March prices were upon the basis of $1.87@1.90 for 80 per cent., at which figures the market continued unchanged for the remainder of the year.

Chromate.—The aggregate business was somewhat lighter during 1908 than it was in 1907. The fact that prices for this commodity are fixed each 12 months has deprived the market of some interesting features. The price for 1908 was 8¾@9c. per lb. The market was confined almost exclusively to current requirements and a few contract deliveries. The figures to rule for delivery over the new year, beginning Jan. 1, 1909, are 8⅝@8½c. A merger of chrome manufacturers, including some of the largest concerns in the country, was completed in September and was the cause of considerable agitation among consumers.

Kainit.—There was very little inquiry for kainit when 1908 opened, and prices were nominally quoted at $8.50@9.50 for bulk material, and $9.50@10.50 in bags, according to quantity. New contracts offered by the syndicate brought out some additional buying on the basis of $8.25 per 2240 lb. minimum 12.4 per cent. potassium, for all Atlantic and Gulf ports. The amount purchased was lighter than usual. During the latter part of 1908 there was comparatively little interest shown, and the market remained at $8.50 per long ton.

Nitrate.—The year was active and prices were steady, with a good demand. The meat-packing and explosive industries took the bulk of contract deliveries. The estimated consumption of the crude material for 1908 is 78,000 bags; this compares with 82,234 bags in 1907. The prices for the crude material through the year were on a basis of 4@4½c. per lb., and for refined saltpeter on the basis of 5@7c.

Prussiate.—The market was very dull during 1908 and prussiate suffered a steady decline in price. In September the price was 13½c. per lb., where it remained during the remainder of the year.

Sulphate.—The market opened in January at the prevailing price of $2.18@2.22½ per 100 lb. for prompt deliveries and continued throughout the year without change or any feature worthy of special mention.

Imports.—The accompanying table gives the amount and value of potassium salts imported into the United States since 1904.

IMPORTS OF POTASSIUM SALTS.(a)
(Tons of 2000 lb.)

	1904		1905		1906		1907		1908	
	Quantity.	Value.	Quantity.	Value.	Quantity.	Value.	Quantity.	Value.	Quantity.	Value.
Bicarbonate	47	$4,778	38	$4,504	22	$2,192	155	$6,787	109	$11,500
Carbonate (crude)	2,405	194,339	2,634	217,041	2,472	267,865	3,732	266,502	2,971	241,996
Carbonate(refined)	6,793	397,104	6,843	440,139	7,489	451,631	9,326	583,730	8,428	575,731
Chloride	80,751	2,428,480	101,183	3,241,152	105,938	3,360,804	115,664	3,863,311	118,474	3,885,419
Chlorate	108	12,541	19	2,352	29	3,868	10	1,486	9	1,447
Chromate	19	2,554	28	3,433	21	3,442	5	685	11	15,453
Cyanide	757	279,204	812	260,206	1,054	321,867	1,535	483,789	1,644	494,915
Hydrate (crude)	2,406	194,839	2,635	217,041	2,473	267,865	3,732	266,502	2,971	241,996
Hydrate (refined)	18	4,879	11	2,537	18	3,979	20	4,545	23	4,967
Kainit	158,984	839,983	240,789	1,143,296	379,221	1,963,914	344,005	2,347,695	329,467	2,008,555
Nitrate (crude)	.....	.....	.....	.....	.....	.....	.....	.....	8,060	449,877
Nitrate (refined)	.....	.....	.....	.....	.....	.....	.....	.....	130	13,708
Prussiate (red)	28	13,508	30	14,453	26	10,300	29	11,811	26	10,697
Prussiate (yellow)	868	156,275	583	103,193	462	85,233	727	148,580	410	88,637
Sulphate	28,796	833,954	27,289	958,305	24,542	922,171	29,153	1,013,045	28,276	1,030,470

(a) From the *Oil, Paint and Drug Reporter*, Feb. 22, 1909.

PRECIOUS STONES.

By George F. Kunz.

There were fewer diamonds, precious stones, and pearls imported into the United States in 1908 than during any year since 1896. The importations for the year amounted to $4,618,999. The small import in 1908 was due to the effects of the financial disturbance of 1907, up to which time, for a number of years, the imports had increased annually. When the disturbance took place, the retail buyer ceased his purchases and many firms found their business at a standstill, stocks were disposed of to other dealers, and many private individuals sold back the jewels they had previously purchased. Moreover, those dealers who were not so seriously affected by the hard times, initiated a policy of retrenchment. Although the value of the imports only amounted to the low figure I have mentioned, yet this does not indicate the actual sales of jewels to retail purchasers, for probably two or three times the amount of precious stones imported changed hands through the shrinkage and liquidation of certain stocks, and through repurchases made from private individuals. Moreover, the policy of retrenchment to which I have alluded caused the sales to exceed the purchases in the case of many retail dealers. It will, therefore, be seen that the imports for 1909 must exceed those of 1908 by 100 to 200 per cent., even if there be no sharp recovery and conditions remain as they are at present. Already they are rapidly improving.

Diamonds.

Transvaal.—The year 1908 was eventful and critical for the diamond industry of South Africa. The enormous development and production, and the opening of new mines, that had been going on for years, were brought to a sudden stop by the panic in 1907, America being the greatest consumer of the diamond output. Retrenchment at once began; and diamonds ceased to be purchased, and became almost a drug on the market. Prices fell rapidly, and the effect was felt at every African mine. The smaller ones closed at once, and the larger ones had to face grave problems of policy in order to continue operations, with lowered prices and almost no demand.

The two great producers, viz., De Beers and Premier, after ineffectual negotiations, decided upon opposite courses. The De Beers company

adopted a policy of reduced production, and the endeavor to uphold prices as far as possible. The Premier company preferred to continue, and even increase, its output, and find its return in large sales at lower prices. The wisdom, and the results of these opposite policies occupied the minds of all concerned, and filled the columns of the press connected with the diamond industry with keen debate. Both companies, however, have weathered the storm, and conditions have now improved. The business crisis has passed; the large stocks of diamonds in the hands of dealers have been gradually reduced; and the mines are resuming some activity, and hoping for a return of the former conditions.

This, however, is not certain. The great and sudden fall in the value of diamonds has shown that they cannot be relied upon, as before was claimed, as a sure investment of capital, available at any time. Moreover, the new discoveries in German Southwest Africa, the development of the great Brazilian diamond region with modern appliances, and the discoveries in Arkansas, small as yet, but possibly important, all tend to cause some question as to the likelihood of the African mines ever regaining the control that they had in the period between the end of the Boer war and the crisis of 1907.

The whole situation is likely to be influenced greatly by the enormous output for which the Premier mine is preparing. Its "No. 4 gear," now being installed, is to raise its washing capacity to 45,000 loads per day of 24 hours. At 300 days in a year, this would yield over 13,000,000 loads per annum, much above the product of all the De Beers company's mines; and at an average of 0.25 carat per load, the annual output of diamonds would exceed 3,000,000 carats. How will this immense production affect the situation? The costs at the Premier are very low, as it is an enormous open mine, and the company can hence afford to sell at low rates and yet make a profit. It has proved this already, in the disastrous period of 1908; and there is a fear freely expressed that the result may be a cheapening of diamonds to such an extent that their traditional value will be impaired. This, however, is not entirely likely to be the case. A great quantity of the Premier material is of very poor quality and will fill a large and new field in which something cheaper than has been sold before can be placed in great quantities throughout such countries as India, Asia Minor, Northern Africa, Spain, Italy, Mexico, and some of the South American countries, as well as the West Indies. On the other hand, it will probably have a tendency to check a much further advance or gradual marking up in the price of diamonds, which it has been the custom of the Syndicate to do every six months, for the last eight or 10 years. The price now is as high as the public will freely take them at, and it is well that the advances should cease.

The De Beers mines, and the Jagersfontein, which is closely affiliated with them, produce diamonds of finer quality than the Premier, at greater cost and selling for higher prices. Their policy, therefore, was reduction of output, and it resulted well, even while the Premier declined to coöperate and went on producing lower-grade stones. But it is claimed by competent judges that the course taken by the De Beers and Jagersfontein companies saved the diamond trade from utter collapse and disaster in 1908. The reduction was made under an agreement between the De Beers and some other companies, and aimed to meet the emergency gradually, with as little hardship and strain as possible under the circumstances. Some mines were to be closed, and the others were worked on a reduced scale. The course of the De Beers company at Kimberley was as follows:

In October, 1907, it was operating five mines, six days (of 24 hours each) in the week, and producing some 30,000 tons of "blue" daily. In November, the reduction began by cutting down the working hours to 16 and the hoisting hours to 12, and later to eight, per day; this lowered the output one-half. The next step was to reduce the days to five per week; and in April, 1908, the Dutoitspan mine, which had been yielding one-third of the company's output, was closed entirely. By June, 1908, these steps had reduced the force employed—the whites to one-half, and the natives to two-fifths—as shown by the following figures:

	Whites.	Blacks.	Total.
1907	4106	25,601	29,707
1908	2162	9,889	12,051
Reduction	1944	15,712	17,656

The salaries of all employees had also been reduced one-sixth. The daily output was brought down to about 11,000 tons.

The annual report of the De Beers company presented at the meeting in December, 1908, gives full particulars of the crisis passed through in the year ending June 30, 1908, and shows also the influence of improved conditions in the subsequent half-year. These had been such that the dividend on preferred shares, due in December, was voted as usual. The previous dividends on the same shares had been also paid, although those on the deferred shares had been passed. The company has on hand a stock of diamonds conservatively estimated at £1,500,000, and a supply of "blue" on the floors that is worth £8,000,000 after washing costs are paid. The tone of the report is assuring, and it appears to be well grounded. The dynamite factory owned by the De Beers company is proving an important feature; it is supplying the great demand of the

Rand gold mines, and an agreement has been formed with them, to furnish one-half of their entire supply for nine years to come.

The exceptional conditions of 1908 led the Diamond Syndicate, which has for many years taken the product of the De Beers company for the market, to terminate that arrangement, or at least not to renew it. The syndicate has since been reorganized, however, and some similar agreement will probably be reached ere long.

Since the autumn of 1908, a revival of confidence has taken place, and plans are being formed for resuming and enlarging the diamond production at various mines. The vast preparations at the Premier have already been mentioned. Great activity is developing at the Voorspoed mine, in the Orange River Colony, which will soon be treating 9000 loads per day. The Kimberley mines have returned to six days in the week; and a general spirit of advance is apparent at all points.

In comparing the results of the opposite policies adopted by the two great companies, it would seem that while both have been justified by a measure of success, that of the Premier has proved most advantageous for itself, while that of the De Beers has operated more favorably on the diamond trade as a whole. This last point has already been referred to, and need not be further explained. The total output of all the De Beers mines was reduced, by the methods above described, from 2,061,973 carats in 1906-07, to 1,473,272 carats in 1907-08; while the value fell in a still greater ratio, viz., from £5,652,321 to £2,914,646 respectively. Notwithstanding these facts, however, the De Beers company met its obligations to the Cape government, £110,000, and paid its dividends on the preferred (though not on the deferred) shares,—£800,000,—and closed its last year with a balance of £563,810.

The Premier company in the year ending with October, 1908, produced 2,078,825 carats of diamonds, against 1,889,986 carats in the previous year, and about half as many in the year before that. The value of the diamonds produced, sold and unsold, exceeded the total cost of production and sale by £789,000, and £290,000 was paid to the Transvaal government for colonial taxes. Dividends of £100,000 were paid to shareholders; and much more would have been divided, but for the cost of installing the great new plant already referred to, which is to increase the output to 3,000,000 carats.

Naturally the De Beers company might be inclined to carry on a competition with the Premier, but as the latter can produce diamonds at so much less cost than the De Beers, some more neutral and less dangerous course will probably be pursued. Although the bulk of the material from this mine is not as fine as that from De Beers mines, some of it is finer.

RECORD OF SOUTH AFRICAN DIAMOND COMPANIES IN 1907–08.

Mine	De Beers-Kimberley.		Wesselton.		Bultfontein.	
Year ending June 30.	1907	1908	1907	1908	1907	1908
No. loads hoisted	2,103,853	1,208,974	2,104,308	1,524,099	2,320,538	1,319,720
No. loads washed	1,467,456	1,130,673	1,891,461	1,719,737	1,728,047	1,275,858
Carats of diamond found	543,752	414,121	604,915	457,028	547,485	411,386
Value of product	£1,762,080	£822,287	£1,243,360	£775,088	£1,191,551	£704,741
Carats per load	.37	.37	.32	.27	.32	.32
Value per carat	64s. 9.74d.	58s. 0.8 d.	41s. 1.3 d.	38s. 11.4d.	43s. 6.34d.	41s. 4.8 d.
Value per load	24s. 0.2 d.	21s. 5.8 d.	13s. 1.76d.	10s. 6.2d.	13s. 9.49d.	13s. 2.97d.
Cost per load	9s. 0.8 d.	6s. 8.2 d.	5s. 8.87d.	4s. 8.73d.	6s. 2.4 d.	6s. 1.86d.
Loads remaining on floors	3,213,875	3,323,237	2,102,803	1,907,165	1,802,309	1,846,191

Mine	Du Toit's Pan.		Premier Year ending Oct. 31.	
Year ending June 30.	1907	1908	1907	1908
No. loads hoisted	2,481,897	1,444,989	.....	8,439,084
No. loads washed	1,539,327	839,075	6,538,669	8,145,794
Carats of diamonds found	365,821	190,737	1,889,937	2,078,825
Value of product	£1,455,330	£612,530	£1,702,630	£1,536,719
Carats per load	.24	.23	.289	.25
Value per carat	79s. 6.78d.	74s. 5.7 d.	18s.	14s. 9d.
Value per load	18s.10.9 d.	17s. 1.4 d.	5s. 2d.	3s. 9d.
Cost per load	6s. 5.84d.	5s. 6.24d.	2s. 4d.	1s. 10d.
Loads remaining on floors	2,272,616	2,878,530	.....	.....

Points to be noted: The extremely high value of stones from Du Toit's Pan. The steady reduction in cost per load, at all the mines (though little at Bultfontein). The diminution in value per carat, and per load. Du Toit's Pan mine was closed in April, 1908; and the old De Beers mine also, for a part of the last year.

A movement has been started to develop diamond cutting in South Africa. The local market, though relatively small, would employ perhaps one thousand workmen. The cost is greater than in Holland; but this difference is offset by the present tax of 15 per cent. on imported cut stones, and a profit is possible even now. To make the industry important, however, an export tax of like proportions on uncut stones is suggested, but this would involve many complications. All agree that the industry would be advantageous, but many difficulties exist, and are strongly presented by those who fear to unsettle present conditions.

Rhodesia.—Diamonds have been found in two regions in Rhodesia, viz., in the Somabula gravels, and near the Bombesi river in true kimberlite pipes. These may prove to be the source of the Somabula occurrences, as they are within 50 miles of that region. The kimberlite is of the ordinary type, with the usual associated minerals. The gravel beds rest upon a decomposed granite, and at times the separation of the two is not easily traced; but they are distinct, and no diamonds can be found in the decomposed bed-rock. These deposits extend over a wide area, and are sometimes 100 ft. thick.

F. P. Mennell, curator of the Rhodesia Museum at Bulawayo, has published some important notes on the diamantiferous rocks of South

Africa. He accepts the suggestion of the late Prof. Carvill Lewis, attributing the abundant calcite of the blue-ground to alteration from original melilite, and finds it strongly confirmed by recent microscopic studies. The eclogite question assumes a new aspect from observations in the Rhodesian pipes. Here eclogite appears in place, resulting from contact metamorphism, and is therefore neither a segregation nor an older igneous inclusion. Mr. Mennell shows also that it contains metamorphic minerals, and cannot possibly be regarded as the original source of the diamonds.

German Southwest Africa.—The diamond discoveries in German Southwest Africa appear now to be authentic and important. The mode of occurrence is peculiar, among sand-dunes near the coast, where small diamonds abound in a gravelly surface layer, not over a foot deep, and are either picked up by hand or washed in sieves, as at the Vaal River deposits. The central point is Luderitz Bay, a town with now about 1000 white inhabitants and a number of good buildings, situated some 500 miles north of Cape Town. The diamantiferous region is a crescent-shaped belt, curving around Luderitz bay for 300 miles from Elizabeth bay northward, and extending 50 or 60 miles inland. It is perfectly arid, being merely sand reaches among granitic hills. Water for drinking is obtained by an evaporating plant, and for washing, by sinking pits or from the sea. The extent of the diamantiferous sand-layer is yet unknown, as it is irregularly distributed, and much prospecting must be done and a full survey must be made. The diamonds are brilliant octahedral crystals, from $\frac{1}{8}$ to $\frac{1}{4}$ of a carat, although the largest stone found weighed a little over two carats. A "load" of sand (16 cu. ft.) may contain from two to occasionally 15 carats. The average value is about 25s. a carat, and the cost of mining is very small. Since the discovery in June, 1908, to January, 1909, the total production was about 40,000 carats (220,000 stones) valued at £55,000.

The entire diamond region is covered by concessions, under several large companies. Thus far, no machinery is in use, the methods pursued being the simplest. The labor problem is serious, though natives can be had in moderate numbers at very low rates. The white population of the German protectorate is small, and immigration is important, but the region is unattractive. All food must be imported, either from the protectorate or from Cape Colony. The diamonds, however, are quite abundant, and quite recently it has been announced that a pipe or neck has been discovered, which may be looked upon as the source of the diamonds found in the alluvial sands.

Dr. Lotz,[1] who resided in the neighborhood of Luderitz bay from 1903

[1] *Centralblatt f. Mineralogie*, No. 8, April 15, 1909.

to 1906, and made a careful examination of the diamond fields in 1908, states that, so far, diamonds only occur in a narrow strip of territory running from the coast and stretching between Elizabeth bay, to the south, and Gallovidia bay to the north of Luderitz bay. No rocks have been found in the neighborhood of the diamond deposits from which the diamonds, which are certainly a secondary deposit, can be derived. The various diamond fields are not in immediate contact with each other and are also at different levels. The diamonds lie in loose material, 70 to 80 per cent. of which consists of reddish sand and 20 to 30 per cent. of fine gravel from the size of pin-heads to that of lentils. Small, finely-striped agate pebbles, jasper, and ferruginous quartz, as well as greenish stones of considerable weight are characteristic marks of these beds.

In the case of a rather rich deposit belonging to the Colmanskop Diamond Mines, Ltd., the proportion of diamond to the accompanying material was found to be as much as 5.3 carats, or 20 to 25 stones to a cubic meter of material, and a small field, 1250x150 ft., belonging to the same company, and which is being worked, contained, on a low estimate, 38,000 carats, worth 1,140,000 marks, at 30 marks per carat. Although but a small proportion of the fields assigned are now exploited, a daily output of at least 600 carats, or 180,000 carats per year, should be attainable, and as the territory reserved by the Government is not included in the above estimate, a much larger output can eventually be expected.

The average weight of the stones is surprisingly constant, and is from ¼ to ½ carat, but stones of a half-carat are frequent. The largest stone found, so far, weighs a little over two carats. The price of exported stones has risen considerably, because of an export duty of 10 marks per carat, so that 60 per cent. of the output brings from 50 to 60 marks per carat. The remainder is kept in the country until an *ad valorem* duty is substituted for the specific one.

Dr. Lotz expresses the opinion that the Luderitz bay diamond deposits represent older coast-formations, in part heaped up and deposited by the wind. The association of the diamond with the accompanying fine gravel, and with the agate pebbles of more recent shore-terraces, might lead us to suppose a connection with the region drained by the Orange river and hence with the Vaal river. This is rendered more probable by the fact that the diamonds of German Southwest Africa have this in common with the "river stones" of the Vaal, that they are more valuable than the diamonds from the blue ground.

To prevent injudicious exploitation, the Government has ordained that, south of 26 deg. south latitude, the fields can only be worked under special conditions.

From an examination of 1762 crystals, mainly from the Colmanskop Diamond Mines, Ltd., Prof. Keiser finds[1] that the predominant form is the rhombododecahedral, although there are many octahedral crystals and twin crystals of the spinel type; perfect octahedral crystals are rarer.

Brazil.—Among the modern methods now being introduced into diamond-mining in Brazil, an important one is that of dredging, which has been applied to portions of the river-bed near Diamantina. An American engineer, G. R. Young, has built and operated very successfully a powerful dredge, which can dig to a depth of 50 ft. The overburden of barren gravel is removed at night, and the "cascalho" layer, 3 to 5 ft. thick, by day, the buckets excavating also several feet down into the soft sandstone bed, to reach the rich cascalho in crevices and hollows. Screening, sizing, and sorting follow; and considerable gold is also recovered.

United States. (By John T. Fuller.)—It is now nearly two years since the first rumors of the discovery of diamonds in their original matrix in Pike county, Arkansas, began to reach the East. This discovery has since been verified by competent engineers and geologists, and active peparations are being made to develop the field on an extensive scale. The exact location of this Arkansas discovery is Pike county, Sections 21 and 28, Township 8 South, Range 25 West. The diamond-bearing area lies on the east bank of the Little Missouri river near the junction of that stream with Prairie creek. The nearest town is Murfreesboro, the county seat, which is about 2½ miles to the northwest. Direct rail communication to Murfreesboro will be established in the near future by a branch line which will connect with the main line of the Kansas City Southern at Ashdown, Arkansas. At present the easiest way to reach the property is by rail from Little Rock to Cooley's Crossing, via Prescott, which is about 6½ miles from Murfreesboro, the latter distance being covered by wagon.

The history of this Arkansas discovery presents several interesting features. First mention of this locality was made as early as 1842, but no report of any accuracy was made until 1890, when the region, including the volcanic area, was mapped and reported on by John C. Branner, who at that time was State geologist.[2] In this report Branner mentions that the rock resembled closely the diamond-bearing rocks of South Africa. No more notice was taken of this area until 1906, when John M. Huddleston, who had in the meantime purchased the property on which the greater part of the diamond area lies, found the first

[1] *Centralblatt f. Mineralogie*, No. 8, April 15, 1909, pp. 251–254.
[2] Geological Survey of Arkansas, Annual Report for 1890.

diamond on Aug. 1, 1906.[1] This was a white stone of 4½ carats. On the afternoon of the same day a second diamond was found by Mr. Huddleston, who, suspecting that he had found something valuable, although he did not know just what, sent the stones to C. S. Stifft, a jeweler of Little Rock, who pronounced them to be true diamonds. Mr. Stifft, with several friends, secured options on the Huddleston tract and on considerable other land adjoining, thus gaining control of by far the greater part of the igneous area. The company thus provisionally formed, secured competent professional advice, and on the strength of the reports that were made, the Arkansas Diamond Company was incorporated.

The matrix of the diamond is a rock which has been described by the late Prof. Henry Carvill Lewis as "a porphyritic volcanic peridotite of basaltic structure," which is named kimberlite.[2] This peridotite, or kimberlite, is more popularly known in South Africa as "blue ground." The use of the term "blue ground" in connection with this rock is misleading. Many people are deceived by this term and take it for granted that the diamonds occur in a soft clay soil, or ground which presents little difficulty or expense to wash and concentrate. This is not the case. The origin of the term "blue ground" is interesting. Like most peridotites, the greater number of the "diamond-bearing peridotites" show marked changes on exposure to the atmosphere and "weather in place" to considerable depth. The fresh peridotite is as a rule of a bluish green color, which changes on weathering to a dull grayish green or yellowish green. During the process of weathering, the rock disintegrates to a great degree and can easily be pulverized between the fingers. It is then, properly speaking, when in this condition, a "ground" and not a rock. This yellow disintegrated peridotite being naturally the top portion of the mass, was the rock in which the diamonds were first found and became known as "yellow ground." In time the "yellow ground" was mined out exposing the hard undecomposed blue peridotite beneath, which was called in contra-distinction by the early miners "blue ground."

The diamond-bearing peridotites of South Africa occur in what is there locally known as a "pipe," which is the neck, vent, or stock, of an old volcano, or dike, filled up solid with the diamond-bearing rock and extending to unknown depth. The Arkansas peridotite has been demonstrated to be in every essential similar to the diamond-bearing peridotites of South Africa, and the most essential characteristic, the actual finding of diamonds within its mass, is fulfilled.

I have heard it said, and seen it written, that this Arkansas peridotite

[1] *Am. Journ. of Science*, G. F. Kunz and H. S. Washington, Vol. XXIV, Sept. 1907, pp. 275-276. *Trans.* A. I. M. E. Feb meeting, 1908, Kunz and Washington, pages 187-194.
[2] Eighteenth Annual Report, De Beers Consolidated Mines.

shows marked differences from the peridotite of Africa. When it is realized that from the same pipe in Africa a dozen or more different hand specimens of peridotite can be taken, no two of which resemble each other either in inclusions or physical appearance, the folly of attempting to find differences in this Arkansas pipe from hand specimens brought from the African pipes, is evident. The only way they can be compared is to compare the masses as a whole, and by this method of comparison I can see no radical difference either in formation, occurrence or in the character of this peridotite from those of South Africa. That the Arkansas peridotite occurs in a pipe similar to its occurrences in South Africa has also been sufficiently demonstrated, by means of bore holes, to be accepted as a fact. The peridotite of the Arkansas pipe, as is also true of the South African pipes, varies considerably in hardness over different parts of the pipe. In most cases the rock will disintegrate readily after a few months' exposure to the atmosphere. . The softer varieties, which are comparable in hardness when fresh with ordinary sandstones, "weather in place" to depths varying from 20 to 60 ft. and form the famous "yellow ground" of the South African mines. The harder varieties of the peridotite, which may and do occur in the same pipe with the softer variety and which are known as "hardibank," are of an entirely different nature. No amount of exposure to the atmosphere will have any practical effect on this rock. In hardness it is equal to the hardest quartzites, and to be concentrated at all, it must first be crushed by powerful crushers. This variety does not weather in place to any extent, and forms the hard outcrop which first drew the attention of geologists to the Arkansas pipe.

A roughly elliptical area represents approximately the peridotite pipe, as known at present. The major axis which lies in the northeast-southwest direction, is about 2400 ft., while the minor axis is about 1800 ft. The area covered is approximately 60 acres. On a line which lies practically through three hills, known as the West, Middle and East hills, are the outcroppings of the hard peridotite referred to as hardibank. To the south of these hills the ground falls away gradually to the Little Missouri river. This south slope on the volcanic area is composed of a much softer peridotite than that of the hills. It has here weathered in place to a depth which may reach 60 ft., and is known to be weathered for at least 20 ft. This portion of the area which is also the largest portion, being at least 35 acres in extent, can be excavated easily without or with a minimum use of explosives to a depth of at least 20 ft. I estimate that there is at least 1,500,000 cu.yd. of this material that can be easily and cheaply mined. Overlying the greater part of the whole area to a depth which does not exceed 1 ft., is a black vegetable soil

locally known as "gumbo." This soil contains some peridotite and a mixture of other rocks and soils brought in by the surface waters. A number of diamonds have been found in this soil which were undoubtedly derived from the underlying peridotite.

The operations to date have consisted chiefly of haphazard experimental tests of the peridotite from various places on the pipe. These tests were roughly made with inadequate washing machines, and the chances are that as many diamonds were lost in the tailings as were recovered. Considerable work has been done in the way of clearing up the area of stumps and gumbo and plowing loose the weathered peridotite. During these operations a large number of diamonds were picked up by chance by the workmen. The number of diamonds found in this way is remarkable and is an indication of considerable richness. Three bore holes have been put down with a diamond drill on the pipe area to depths of 79, 184 and 205 ft. These holes are in the peridotite for the entire distance sunk.

Toward the end of 1908 all work on the property was suspended until such time as a complete mining, washing and concentrating plant can be designed and erected, when mining on a large scale will be undertaken. The total number of diamonds recovered from this area up to the end of 1908 by chance finds or concentration is 540. Of these 540 stones 505 weighed 217 carats, or an average of nearly ½ carat per stone. The largest stone so far found weighs 6½ carats. Most of the stones found are white, while a few are yellow or brown. A large number of these stones are of good water and of remarkable purity, many being of finer quality than African stones. Three of those found have been cut and have yielded beautiful gems which have been valued at from $60 to $175 per carat, with an average value for the three of $104 per carat. Taking a parcel of rough, unsorted stones from this pipe it will easily average $10 or more per carat in value at the present prices of diamonds. This average equals that of several of the large producing mines of Africa.

The engineer who is called upon to report on an undeveloped diamond mine has the serious obstacle to confront that he can make no definite statements or figures as to what average may be expected from the working. It is impossible to make assays of a diamond mine. Bore holes give only indication of the nature of the peridotite and tell nothing of its richness. The only way that a definite statement of values can be made is actually to mine, wash and concentrate at least 50,000 cu.yd. of the rock. The greater the amount the more accurate will be the result. Test runs from certain parts of the Arkansas pipe indicate at least 0.21 carat per load of 16 cu.ft., which may be considered an excellent

average if it can be maintained. In many ways the Arkansas pipe has decided advantages over the South African mines. There is an abundant and nearby supply of water and timber. Fuel and mining supplies are comparatively cheap. Mining should be done here at a cost which would be unattainable in Africa.

WEIGHT IN CARATS AND COLOR OF THE FIRST 75 DIAMONDS FOUND IN THE ARKANSAS FIELD

No.	Weight.	Color.	No.	Weight.	Color.
1	2½	White	39	¼	Brown
2	2¼	Light brown	40	¼	Brown
3	2⅛	White	41	¼	Green yellow
4	1¼	White	42	¼	Brown, found in screen wash
5	1⅛	Crystal white			
6	¾	White	43	1/32	White, found in dry wash
7	¾	White			
8	¾	White	44	Diamond in rock	
9	¾	White	45	4¼	Perfect blue
10	¾	Light brown	46	2¼	Crystal
11	¾	Yellow	47	2⅛	Brown
12	¾	Yellow	48	⅞	White
13	¾	Cloudy brown	49	⅞	White
14	¾	Cloudy white	50	⅞	White
15	¾	Dark brown	51	¾	White
16	¾	Bort	52	¾	White
17	⅝	Light yellow	53	⅝	White
18	⅝	White	54	⅝	White
19	⅝	Crystal	55	⅝	White
20	⅝	Light brown	56	⅝	White
21	⅝	Light brown	57	⅝	White
22	About ½	Yellow	58	½	Yellow
23	2	White	59	½	Yellow
24	2	Silver cape	60	½	Yellow
25	1⅞	Crystal	61	½	Brown
26	1⅞	Crystal	62	½	Brown
27	1⅝	Yellow	63	½	Brown
28	1⅝	Light yellow	64	½	Brown
29	1⅝	Yellow	65	½	Brown
30	1⅝	White	66	½	Brown
31	1⅝	White			
32	1⅝	White	67	3	are diamonds found in two pieces.
33	1⅝	White	68	1	Brown
34	1⅝	White			
35	1⅝	White	69	¼	Brown
36	1⅝	White	70 to 75	6 pieces bort. Total weight 2½ carats.	
37	1⅝	White			
38	1⅝	White			

RUBIES.

The reports of the Burma Ruby Mines for the year ending Feb. 29, 1908, show a total output of 2,033,666 truckloads of ruby earth,—considerably more than in any previous year—at a cost of 7.6d. per load; the net profits were $18,906,—both somewhat less than in 1907, though more than in 1906. The company pays the government a rent of 200,000 rupees (£13,400), plus 30 per cent. on all further profits, both from regular working and from royalties in excess of 200,000 rupees paid by native miners.

Owing to the depression in the gem-market, and with a view to restrict the output, the company had entirely stopped night work, from the first of February, 1908, and the wages of the coolie laborers, as well as all

other expenses, were reduced as much as possible. As precautionary measures, the payment of the semi-annual rent due the government on Feb. 29, 1908, was with official consent, deferred for the present, and the usual dividend was postponed until trade conditions should improve, although the output for the past year had increased, and the cost somewhat diminished. An important work in connection with the mining operations, the Mogok Valley drainage tunnel, is reported as nearing completion.

Other Stones.

Turquoise.—Edward R. Zalinski has published in *Eng. and Min. Journ.*, Oct. 31, 1908, a description of the turquoise occurrences in Grant county, N. M., and particularly of the Azure mine, thus far the most important. The rock of the immediate region is biotite-granite, intruded by masses of andesite-porphyry, and both traversed by dikes of mica-andesite. The granitic uplift, which formed the Burro mountains, was post-Cretaceous, but earlier than the great volcanic activity of the Tertiary, to which period may belong the later andesite dikes. The granite is much altered and kaolinized near the andesite masses, and it is in this altered portion that the turquoise mainly occurs. The Azure mine itself is in a well-defined vein or zone of alteration, about 50 ft. wide, with distinct walls; and the turquoise appears in two forms, as nodules or concretions in kaolin, and as veins filling cracks in the altered granite. Its quality varies much, and only the best is shipped away; but some of this is equal to the best Persian material, and the total output has reached a greater amount than that from any other group of mines.

Burmese Jade.—A full and very valuable account of the occurrence of jadeite in Upper Burma has been given by A. G. W. Bleeck, presenting an interesting theory of the origin of this mineral.[1] He visited three localities where it occurs, in the Myitkyina district, but dwells chiefly on that at Tammaw, about 50 miles northwest from the Burma railway, where the jadeite is quarried in place, and not obtained (as elsewhere) from boulders. The rocks are described in detail, and their relations strikingly presented. The jadeite occurs in a great dike, plainly igneous, traversing a ridge of serpentine, flanked by "crystalline schists," altered from igneous rocks. The dike gives evidence of some remarkable modes of differentiation in the magma from which it was formed, and of subsequent alteration. It now consists mainly of jadeite and albite, the latter flanking the former on both sides, and passing into it by mixture. Actinolite and some other minerals are also present, and their relations are carefully studied; but the mineral of prime importance is the albite,

[1] "Jadeite in the Kachin Hills." *Records*, Geol. Survey of India, Vol. XXXVI, part 4, 1908, pp. 254–285.

which is also known elsewhere in association with jadeite, and is the only feldspar so associated. Some occurrence of nephelite is also noted.

The genesis of the jadeite is then taken up, forming the most interesting part of this important paper. Mr. Bleeck reviews previous theories as to the jadeite-albite rock, and gives his own conclusions very clearly and forcibly. With Noething, the only other geologist who had studied the actual occurrence, he considers it clearly an igneous intrusion, and refers to the penetration of small veins from the main dike into the country-rock. After much discussion of the magma, whence the dike and, in his judgment, some of the adjacent rocks were derived, Mr. Bleeck presents the view that the dike originally developed as a mixture of albite and nephelite, and that a subsequent intrusion of granite, which altered the basic rocks into the "crystalline schists," caused a change also of the nephelite-albite to a jadeite-albite. From comparison of a series of analyses, the striking fact is shown, that "one molecule of nephelite plus one of albite are equal to two of jadeite ($Na\ Al\ Si\ O_4 + Na\ Al\ Si_3\ O_8 = 2\ Na\ Al\ Si_2\ O_6$), and that such an alteration, affecting the mineralogical, but not the chemical, constitution of the dike, is perfectly possible. The microscopic structure of the rock shows that it has undergone intense pressure since its original development; and jadeite is a mineral of high density (3.2) in the pyroxene group. The general conclusion, based upon a great array of observations and data, is expressed as follows: "That the jadeite-albite dike . . . originally developed under normal conditions as a nephelite-albite rock. As neither albite nor nephelite possesses heteromorphic equivalents which could crystallize under anomalous conditions, metamorphism cannot alter either of these two minerals . . . independently of the other." But the combination of one part albite and one part nephelite, yields jadeite, which could thus be produced by metamorphism under great pressure.

The other Burman occurrences of jadeite are as boulders, some in the beds of streams, and some in a conglomerate of Tertiary age, as shown by associated beds containing Miocene plant-remains. Whether these boulders came from the dike at Tammaw or from other unknown outcrops, has not been fully determined, although the former is probable.

Opal.—Australia, that land of surprises in the domain of natural history, as well as in that of mineralogy, has given us many forms of opal during recent years. First, the wonderful specimens that are found in the brecciated ironstone which was filled with large, wonderful white, blue, and blue-green opals. Then again, there are the opals found in the Wilcannia White Cliffs, New South Wales district, where the most beautiful white noble opals frequently reproduce the vertebrae and other bones of animals, markings in wood, marine shells, etc.; here a

mineral resembling gypsum or gaylussite, is also found. The pine-apple opal, as it is termed, sometimes occurs in nodules the size of a man's fist.

Until within the past few years, black opal was exceedingly rare. It was supposed to bring "good luck" to its wearer and for this reason, as well as for its beauty, it has always been highly prized. During 1906-08, however, a quantity of black opal, of a velvet-black in general effect, but permeated with a rich display of red, green, blue and purple, was discovered at the Lightning Ridge mine in New South Wales. Gems weighing 100 carats were taken from there, and a quantity of material has been obtained and formed into specimens which have found their way to the gem markets of the world. The opal field is situated in close proximity to the Queensland border and is said to have an extent of 25 miles.

Formerly the name "black opal" was often erroneously bestowed upon matrix opal, a material consisting mainly of veins of opal in the matrix rock; but the true black opal, such as is found at Lightning Ridge, is composed entirely of opal. The material occurs as nodules in the rock and varies much in quality. It frequently happens that fine specimens cannot be utilized for gem purposes because of extensive flaws, or "sand spots" as they are called, which traverse the stone in different directions.

In 1906, opals to the value of £8000 were secured from the Lightning Ridge fields and the worth of the total output there is nearly £50,000. Cut stones have sold for from $5 to 50 and more per carat, the higher prices being for very choice pieces owing to their novelty and present rarity.

QUICKSILVER.

By H. W. Turner.

The larger part of the domestic production of quicksilver is consumed within the United States and is mostly sold through the Eureka Company of San Francisco. Shipments to Mexico are small, and those to the Orient have practically ceased, since to compete with the Spanish product would reduce the price to a low figure. The import duty of 7c. per lb. protects the domestic product.

STATISTICS OF QUICKSILVER IN THE UNITED STATES.

Year.	Production.					Exports.			Imports.	
	Calif. (a)	Texas.	Others.	Total.	Value. (f)	Flasks	Metric Tons.	Value	Pounds.	Value.
	Flasks.	Flasks.	Flasks.	Metric Tons						
1896...	30,765	.....	.....	1,061	$1,075,449	19,944	692	$618 437	.....	$2,037
1897....	26,648	.....	.....	919	993,445	13,173	475	394.54	45 539	20,147
1898....	31,092	(b)	153	1,077	1,194,746	12,880	445	440,587	81	51
1899....	29,454	261	.....	1,025	1,416,790	16,518	573	609,586	131	83
1900....	26,317	1,700	233	974	1,279,436	10,702	353	425,812	2,616	1,051
1901....	26,720	2,932	75	1,031	1,382,205	11,219	389	475,609	1,441	789
1902....	29,552	5,252	.....	1,208	1,515,714	13,247	459	575,099	*Nil.*	.....
1903....	32,094	5,029	.....	1,288	1,564,734	17,575	610	719,119	*Nil.*	.....
1904....	28,876	5,336	700	(c)1,204	1,348,185	21,064	731	841,108	212	160
1905....	24,655	5,000	1,050	1,045	1,217,052	13,460	458	497,470	2,690	1,710
1906....	19,516	4,517	1 276	861	1,035,138	6,455	220	244,299	84	50
1907....	(d)17,532	3,000	400	712	868,678	5,132	175	192,094	16,566	6,719
1908...	(d)16,969	(b)	(e)1000	661	805,690	2,995	110	124,960	15,113	8,215

(a) Reported by the California State Mining Bureau, except 1907-08. (b) Included in "Other States." (c) Estimated; the weight of the flask was changed from 76.5 lb. to 75 lb. within this year. (d) Figures collected by *The Mineral Industry*. (e) Grouped to avoid disclosure of identity of individual producers. (f) Computed at average price at New York.

Of the domestic consumption, most of the quicksilver is used in the manufacture of vermilion, of scientific instruments, and mirrors. The consumption for amalgamation of gold and silver ore is not large. It is estimated from careful records that not more than 0.2 oz. of mercury is used up in amalgamating the gold and silver of one ton of ore, and in the Alaska-Treadwell and some Mother Lode mines, the amount is less.

California, the largest producer in the United States of free-gold ore, mills yearly about 2,500,000 tons of silicious ore, using the amalgamation process for the recovery of the free gold, and consuming for this purpose not more than 400 flasks of mercury yearly. The amount now consumed

in placer mining in the United States is not great, but in Alaska it must be considerable. The Lagrange hydraulic mine in Trinity county, Cal., consumed about 3½ flasks of quicksilver in collecting the fine gold from 4,000,000 cu.yd. of gravel in eight months, or about 0.001 oz. per cubic yard.

QUICKSILVER PRODUCTION OF THE WORLD.
(Metric tons.)

Year.	Austria.	Hungary.	Italy.	Mexico.	Russia.	Spain.	United States.	Total.
1896	564	1	186	218	491	1,524	1,036	4,020
1897	532	1	192	294	616	1,728	965	4,328
1898	491	7	173	353	362	1,691	1,058	4,135
1899	536	27	205	324	360	1,257	993	3,902
1900	510	32	260	124	304	1,095	963	3,308
1901	525	33	278	128	368	754	1,031	3,117
1902	511	45	259	191	416	1,425	1,208	4,055
1903	523	44	314	188	262	966	1,288	3,587
1904	536	45	357	(e) 190	393	1,120	1,192	3,843
1905	519	36	370	(e) 190	318	853	1,045	3,331
1906	526	50	418	(e) 200	210	1,568	963	3,935
1907	527	40	423	(e) 200	130	1,212	712	3,244

(e) Estimated.

QUOTATIONS FOR QUICKSILVER IN LARGE LOTS.

Month.	1905			1906			1907			1908		
	New York.	San Francisco.		New York.	San Francisco.		New York.	San Francisco.		New York.	San Francisco.	
		Domestic.	Export.		Domestic.	Export.		Domestic.	Export.		Domestic.	Export.
Jan...	$40.00	$40.42	$39.17	$40.25	$39.13	$37.63	$41.25	$39.50	$37.50	$45.00	$45.00	$43.50
Feb...	40.00	38.88	37.63	41.00	39.50	38.00	41.25	39.00	37.37	45.00	45.00	43.50
Mar..	38.95	38.15	36.90	41.00	39.50	38.00	41.00	38.50	37.25	45.00	45.00	43.50
Apr...	38.25	38.00	36.70	41.00	39.50	38.00	41.00	38.50	37.25	45.00	45.00	43.50
May..	38.38	38.25	37.00	41.00	39.50	38.00	41.00	38.50	37.25	45.00	44.50	43.50
June..	38.50	37.85	36.50	41.00	39.50	38.00	41.00	38.50	25.37	44.25	44.00	42.50
July..	41.25	39.00	37.75	41.00	39.50	38.00	41.00	38.50	25.37	44.00	43.50	42.00
Aug...	40.50	39.00	37.75	41.00	39.50	38.00	40.00	38.00	36.75	43.30	42.70	41.30
Sep...	40.00	39.00	37.75	41.00	39.50	38.00	40.00	38.05	36.70	42.87	42.25	40.50
Oct...	40.00	39.00	37.75	41.00	39.50	38.00	40.50	38.19	36.50	43.25	43.50	41.62
Nov ..	40.00	39.00	37.75	40.75	39.50	37.50	45.00	45.00	43.50	46.60	44.50	42.50
Dec...	40.00	39.00	37.50	40.75	39.50	37.50	45.00	45.00	43.50	45.75	45.12	43.12
Year..	$39.65	$38.80	$37.52	$40.90	$39.47	$37.89	$41.50	$39.60	$38.17	$44.84	$44.17	$42.54

Arizona.—The Colonial Mining Company at Ehrenberg is opening a deposit by means of a shaft said to be 300 ft. deep. The ore contains sulphides of copper and mercury, as well as some gold and silver. The intention is to recover the mercury by a preliminary roast, and subsequently to treat the residue for copper, gold and silver. The deposit is not a new one, for it was known to J. W. C. Maxwell 30 or more years ago. A 30-ton Scott furnace has been erected.

California.—As in recent previous years, the chief producer in California in 1908 was the New Idria. B. M. Newcomb states that three furnaces are now in commission, one 40-ton fine-ore modified Scott furnace, and two 80-ton Newcomb coarse-ore furnaces. The average daily con-

sumption of the three furnaces is about 200 tons of ore, but their actual capacity is greater. For two years, oil fuel was used in the furnaces. This gave great satisfaction, on account of a saving in labor, and of the formation of less soot in the condensers than with wood fuel. Wood fuel is now cheaper, and is used exclusively, one cord of wood being equal in fuel value to about 2½ bbl. of oil. The sulphur in the pyrite of the ore serves also as fuel, reducing the amount of wood required. In 1907 the amount of ore treated was 59,285 tons, 0.486 per cent. mercury being recovered, or 7675 flasks; the cost per flask was $25.40; the value per flask, $37.10; and profits $89,650 for the year.

The statement quoted in THE MINERAL INDUSTRY, Vol. XIV, 1906, that the Florence Mack mine in San Benito county had extracted 35,000 tons of 2½ per cent. ore is evidently an error, but development work is going on in that property.

At the Karl mine in San Luis Obispo county, N. S. Kelsey is opening new ore, and making some changes in the furnace for a future run. Some of the new condensers will be built entirely of concrete, and iron condensers cooled by means of overflowing water pans on top, will also be used. At the Oceanic mine, there has been a small production.

The Napa Consolidated mine in Napa county, in 1907 reduced 18,853 tons of ore, recovering 0.5 per cent. mercury, or 2440 flasks. The cost per flask was $34.38, and the value per flask, $37.05. In 1908 this mine produced about 200 flasks per month. One of the furnaces, a modified Scott fine-ore furnace, was run for four years without closing down for repairs or for cleaning condensers. At the St. John's mine in Solano county a body of 0.7 per cent. ore was recently struck in a shaft, the point being 410 ft. from the surface. A fine-ore Scott furnace may be erected. In Lake county the Great Western mine using a Litchfield furnace is still producing, as is also the Helen mine, which has considerable good ore and a Scott furnace. In Sonoma county, at the Culver-Baer (formerly the Oakland) a good orebody has been opened up, and preparations are now being made to erect a Scott fine-ore furnace. The Great Eastern produced during most of 1908, but is now nearly exhausted. In Santa Clara county, both the New Almaden and the Guadalupe mines are producing. In Siskiyou county a quicksilver deposit on the west fork of Beaver creek is being developed. The lode is said to be a dike with a net-work of seams carrying cinnabar in a micaceous schist country rock. Cinnabar has been known from this vicinity for many years, but there is apparently no production on record.

Nevada.—There are no producing mines in Nevada. A cinnabar-bearing lode has been found on the eastern slope of the Humboldt range,[1]

[1] *Min. and Sci. Press*, XCVII, 156.

about 26 miles south of Mill City. The ore occurs in a vein in limestone at a contact with intrusive diabase. Coarse gold also is found in this vein. Cinnabar occurs in a gangue of calcite and quartz and is exposed in cuts for 1400 ft. along the lode. Development work is in progress. Cinnabar is also found in the west side of the Humboldt range, six miles south of the Humboldt House. The ore occurs here in irregular bodies in limestones and serpentine. Cinnabar has been found in the Red Cloud gold mine about four miles northwest of Good Spring, in Lincoln county, on a spur of Potosi mountain in the Yellow Pine district, and there is a vein in a granitic rock about 10 miles from Belmont in Nye county, from which rich ore has been extracted. In the sulphur deposit now being developed at Cuprite, 14 miles south of Goldfield, there is some cinnabar, and considerable gold is also reported.

(By W. H. Shockley.)—In former years a few flasks of mercury were distilled from the ores of Mill City, Humboldt county. In 1908 Davis & Workman produced 50 flasks of quicksilver from cinnabar at Ione, Nye county.

Oregon.—The Blackbutte is the best developed quicksilver mine in Oregon. The main deposits occur along a strong fissure which is more than 3000 ft. in length, dipping north about 57 deg. and striking south 67 deg. east. The country rock is volcanic ash (andesite) with massive lavas, all of Tertiary age, and these materials are strongly bleached by the action of the waters that formed the cinnabar deposits. There is but little native mercury found in this mine, and only traces of stibnite and bitumen. The property is finely situated for cheap working, as wood for fuel is plentiful and cheap, water power is available, and the mine is opened by adit levels so that the ore can all be handled by gravity. The total length of adits, drifts and raises is more than one mile. A hydroelectric plant for power and lighting has been installed, and the old Scott furnace remodeled according to plans of W. B. Dennis, producer gas being used for fuel. This furnace is said to have been fired up in the early part of 1908 and some quicksilver produced.

Cinnabar is said to occur in a lode in the southeast corner of Oregon near the Idaho-Nevada line, and about three miles from the R.A.D. gold mine (which is in Nevada).

Texas. (By Walter E. Koch.)—Quicksilver ore is mined at Terlingua, in Brewster county, about 12 miles from the Rio Grande river and about 100 miles from Marfa station. The ore is found in veins or pockets in Cretaceous limestone and is mined chiefly by open cut. It is smelted in wood-fired furnaces. There are several companies owning mines in this district, but only one was operated in 1908.

Quicksilver Mining in Foreign Countries.

Asia Minor.—The rediscovered mines of Koniah have been reported on by F. F. Sharpless.[1] The country rock is limestone. The occurrence of the mineral is not uniform throughout the limestone; it appears to be present only where, or near where, the limestone is much silicified or entirely replaced by quartz, suggesting that the quartz and mercury had a common origin, and were deposited about the same time, the greater part of the workable deposits consisting of the silicious limestone seamed with little stringers of cinnabar, the whole assaying 1 to 2½ per cent. mercury. One considerable body was found associated with stibnite; this carried 8 per cent. of mercury.

This property appears to have been operated, perhaps as a paint mine, possibly 3000 years ago. Some of the ore was found by a goatherd, and was sent to Whittall Bros. of Constantinople. One of the Whittall brothers, who is a mining engineer, visited the place and found a cave in limestone, the walls of which showed stringers of cinnabar. This cave may have been an old mining excavation. At any rate, a winze sunk in the floor broke into another cave in which there were more than 50 human skeletons. There were quantities of stone hammers, several pottery lamps, etc., and the presumption is that the skeletons were those of miners entombed by a cave.

According to F. P. Monaci,[2] about 13,000 tons of 1 per cent. ore has been blocked out. The smelting plant consists of a Spirek shaft furnace, capacity 15 tons per day, and a Czermak-Spirek furnace having a capacity of eight tons per day. Monaci further states that at the Kara Bournu mine about 18 miles from Smyrna, there is a quicksilver deposit consisting of a silicious rock impregnated with cinnabar. The deposit is of sufficient size to be quarried, and ore containing only 0.25 per cent. mercury is profitable. The production for the year 1906-7 was about 3000 flasks from ore averaging 0.75 per cent. mercury. There are two Spirek shaft furnaces for coarse ore and one Czermak-Spirek furnace for finer ore. The Karabouan Mercury Syndicate (presumably owning the Kara Bournu mine) reports for the year ending Sept. 30, 1908, sales of quicksilver amounting to £23,076, and profits of £15,850.

Austria.—Quicksilver ore is reported[3] from two places, Peroc and Donja Glavica, near Spizza in Dalmatia. The ore is cinnabar in a gangue of calcite and barite, and according to Bukowski, is accompanied by a little stibnite. The deposits do not appear to be important.

Brazil.—According to the *Bulletin* of the Internat. Bureau of American Republics for January, 1909, work has been begun on the mer-

[1] *Eng and Min. Journ.*, LXXXVI, 601.
[2] *Rassegna Mineraria*, April 11, 1908.
[3] *Zeit f. prak. Geol.*, Feb., 1908

cury mine recently discovered in Minas Geraes. This mine is situated near Tripuhy station on the Ouro Preto branch of the Brazilian Central Railroad.

British Columbia.—The following notes are on the authority of Wm. Fleet Robertson, Provincial Mineralogist: No quicksilver mines in British Columbia are now being worked. In 1902, in the Yale mining district near Savona on the Canadian Pacific Railway, the Hardie Mountain Cinnabar Mining Company ran more than 2000 ft. of tunnels to open up its mercury deposits; the Copper Creek Cinnabar Mining Company did assessment work; and the Toonkwa cinnabar claim south of Savona is reported to have found a body of good ore. A small quantity of cinnabar and metallic mercury was found on an island in Barclay sound on the west coast of Vancouver Island; and in the fall of 1908, samples brought in from the north end of Vancouver Island, contained an average of 0.5 per cent. mercury.

According to G. F. Monckton,[1] cinnabar is found near Kamloops lake in the Nicola formation, which consists of volcanic rocks and sandstones interbedded and in the overlying Oligocene rocks and volcanic and dolomite beds. The dolomites only have produced commercial ore. The British Columbia Cinnabar Mines Company, with mines at the mouth of Copper Creek, produced 114 flasks from 150 tons of ore (0.27 per cent. mercury) some years ago, but is not now producing.

China.—Brelich[2] reports that the Wan Shan Chang mines are the principal and most extensively worked deposits of quicksilver in Kweichow. They are situated in the prefecture of Toon Yen, about 1400 miles by road from Shanghai. The ore occurs in nearly horizontal beds of magnesian limestone: (1) impregnating well-defined beds; (2) along joints, cracks and planes of stratification; (3) in isolated bunches, vugs, etc.; (4) irregularly disseminated through beds which in most cases have undergone local disturbances. There are two varieties of the ore; one a bright transparent red; and the other a dark opaque red, with which there is usually associated antimony in small amounts.

In extracting the ore, the miner follows a stringer or mineralized band wherever it leads, resulting in irregular workings. The miners use iron drills pointed with steel, and the holes are usually about 15 in. deep. Gunpowder is used for blasting. This powder is made at the mines. The pay of the miners is about 25c. per day, and they are furnished with two meals. A shift is five or six hours; overtime is paid for. The ore is sorted in the mine and brought to the surface in bamboo baskets of about 35 lb. capacity. It is then further cleaned, and crushed to pass through

[1] *Trans.* I. M. M., London, 1904.
[2] *Trans.* I. M. M., 1905, XIV, 483–495.

a ¼-in. bamboo sieve. If the cinnabar is the red variety, the crushed ore is panned, and the clean cinnabar removed, the tailings being retorted. The dark ore is treated directly without panning. The quicksilver brings from $30 to $35 per 75 lb. at the mines. It is used mostly in the manufacture of vermilion.

Brelich says nothing about the Anglo-French Quicksilver Mining Concession, which in 1902[1] was said to have had two 12-ton granzita furnaces in operation.

(By T. T. Read.)—Quicksilver is extensively employed in China to make vermilion, a pigment that is in very common use. A large part of the demand is at times met by the native product, at others largely by importation. The present seems to be one of the periods of depression, as less than 30 tons of the native product passed through the customs in 1907. All of this comes from the province of Kueichou, where there are three productive districts. These were described by Brelich in *Trans.* I. M. M. for 1905. Quicksilver also occurs in several other provinces, but does not seem to be worked.

Hungary.—It is reported that Baron Ottokar Jacobs has discovered a deposit of mercury ore at Alsokomonocs (Zempelen district), and intends to work it at an early date.

Italy.—The main quicksilver district is the Monte Amiata, which was described[2] by Vincenzo Spirek. Monte Amiata itself is composed of trachyte, but the cinnabar occurs mostly in limestones of various ages from Jurassic to Tertiary. Besides mercury, there are found small amounts of copper, arsenic and antimony. The ore was deposited from solutions reacting on the limestones. Gypsum, calcite and pyrite are prominent as vein materials. From 1894 to 1904 inclusive, the Monte Amiata mines produced 341,749 tons of ore, from which mercury equivalent to 71,306 flasks of 75 lb. was obtained, showing a recovery of 0.78 per cent. The following costs are deduced from the returns for 1903-04: Tons of 2000 lb. smelted, 127,790; flasks of 75 lb. produced, 19,518; percentage recovery, 0.57; costs per ton of ore, $4.82;[3] cost per flask, $31.52; value per flask, $37.54.

The types of furnaces mostly in use are the Czermak-Spirek continuous automatic reverberatory for fine ores, described in THE MINERAL INDUSTRY, Vol. VI; the Spirek shaft furnace for coarse ore, described in Vol. X; and the improved Spirek reverberatory, described in Vol. XI. The shaft furnace has also been used for smelting zinc and lead ores containing mercury at the Taghit mine in Algeria, the quicksilver being recovered

[1] *The Mineral Industry*, Vol. XI.
[2] *Mining Magazine*, April, 1906.
[3] Spirek gives the extremes of costs of mining in different mines as 17 *lire* and 29 *lire*. I have taken 21 *lire* as an average. The cost of smelting is 77c. per ton of ore.

in a preliminary roast. Spirek claims a loss of only 4 per cent. of mercury in his improved furnace. The advantages of his furnace consist in: (1) Slow circulation of gases in the condensers, less than 2 ft. per min., giving time for solid materials to deposit; (2) temperature of gases in condensers is kept low, between 68 and 86 deg. F., water being used for cooling; (3) all gas pipes communicating with collecting troughs are kept under water seal; (4) in the upper zone of the furnace, the moisture is removed without any escape of furnace gases, thus obviating the necessity of drying the ores outside.

In the ascending movement of the gases in the furnace, their temperature is communicated to the ores, so that the temperature of the gases is lowered to about 392 deg. F. at the point of exit into the first condenser. The air required for combustion and roasting is heated by means of the cinders in the residue compartment of the furnace. The cinders are discharged at a temperature below 212 deg. F., so that three-fourths of their heat is recovered. The gases are cooled down from the furnace temperature of about 1300 deg. to about 390 deg. F., where they enter the first condenser; and in the first pipe to the first condenser down to about 194 deg. F. There is a plate of sheet iron below the furnace to prevent penetration of mercury into the foundations. The roasted ore (cinder) is flushed away by water that has been used for cooling the condensers, and by this means labor is saved. The normal temperature of the gases at their exit from the condensers is 86 deg. F.

A cinnabar deposit in Pliocene sandstone has been found near Pereta and opened up to some extent by shafts.[1] The Upper Pliocene sands are impregnated with cinnabar, and in the underlying Eocene limestone there are veinlets of calcite containing cinnabar and pyrite.

Mexico.—The Huitzuco mercury mine in Guerrero is remarkable for the large amount of antimony found in the ore. According to Paglucci,[2] the ore is a sulphantimonide of mercury and occurs in an earthy form in old pipes of extinct geysers and in depth in stringers, bunches and layers in Cretaceous limestone. Gypsum and native mercury are found in the ore in depth. One shaft is 700 ft. deep. The deposit occurs in low rounded hillocks rising 300 ft. above the town and 3500 ft. above sea-level. The ore is treated in a continuous furnace built after plans furnished by J. W. C. Maxwell of San Francisco. The deposits are said to have been found by an Indian in 1874. One of the antimony-mercury minerals is known as livingstonite.

The Guadalupana mine in the state of San Luis Potosi is described in THE MINERAL INDUSTRY, Vol. XVI. The cinnabar occurs as an impreg-

[1] *Rassegna Mineraria*, July, 1908.
[2] *Eng. and Min. Journ.*, March 2, 1905.

nation in the country lime, as isolated masses, and as stringers between the strata. The total production up to February, 1903, was 3520.8 short tons of ore, from which mercury equal to 6787 flasks of 75 lb. or 7.22 per cent. was extracted.

Russia.—In 1907, according to Rogovin,[1] the only mines worth mentioning that produced were those in the Bachsmut district, province of Ekaterinoslav, in southern Russia, controlled by Auerbach & Co., the product being 8055.4 poods, equal to 3879 flasks of 75 pounds.

(By I. I. Rogovin.)—There was no production of quicksilver in Russia in 1908. The only mines in the Empire are those of Auerbach & Co., which were idle because of financial difficulties.

Progress in the Metallurgy of Quicksilver.

During 1908 there were no improvements of note in quicksilver furnaces. The most popular furnace in the United States is the fine-ore tile furnace built by Robert Scott, although some modifications of this have been introduced at New Idria and Oathill. The term fine-ore is used here to include both *tierra* (finer) and *granzita* (medium) grades of ore, as both are usually worked together in the same furnace.

Coarse-ore furnaces, differing in detail, are in use at several of the mines in California, as the Newcomb furnace at New Idria, and the Maxwell furnace at the Great Eastern.

The Dennis fine-ore furnace at Blackbutte, Ore., is not yet in steady operation, and it is doubtful if it is adapted to large operations. In this furnace, there is a series of hearths, the ore being dumped from one to the next lower. Producer gas is used for fuel, which is a desirable innovation as no soot is produced.

The losses in the Czermak-Spirek furnaces in use at Monte Amiata, Italy, are given as 4 per cent. It is probable that the losses in the best American furnaces are much more than that. Now that the grade of ore treated is becoming lower, year by year, it would appear that the subject of furnace losses should receive the attention of a competent metallurgist.

Oil fuel is used at New Almaden, but at New Idria its use has been discontinued, as at the present price of oil wood fuel from the neighboring forest reservation is cheaper. Coke is used in the Neate coarse-ore furnaces at the St. John's. These are shaft furnaces, the coke and ore being charged in alternate layers or mixed with the ore.

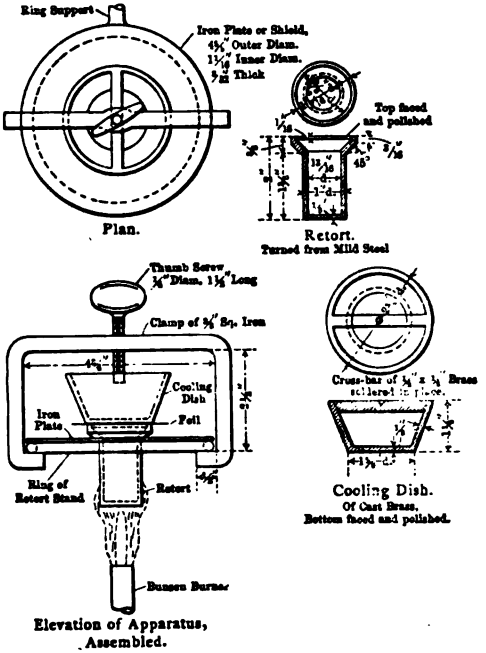
In the reduction plant at the Texas-Almaden mine in Texas[2] the first four condensers are described as being lined with dry-pressed machine-

[1] *Eng. and Min. Journ.*, LXXXVI, 119
[2] *Eng. and Min. Journ.*, Dec. 1, 1906.

made brick of a fine quality with close mortar joints, thus presenting a smooth surface so that a minimum amount of quicksilver penetrates the joints, and the soot does not readily cling to the walls.

Chas. J. Burgess has patented a furnace for roasting quicksilver ores (1907, U. S. Pat. No. 845,690) consisting of an inclined shaft, provided with soapstone shelves. The ore is presumably dumped from one shelf to the next lower. The ore is charged through a hopper over the top of the shaft. This hopper opens and closes automatically.

Assaying Mercury Ores.—A new device for assaying the ores of mercury has been designed by W. W. Whitton.[1] This has been thoroughly tested and found to be accurate. Briefly it consists of a steel retort, with



WHITTON APPARATUS FOR ASSAYING MERCURY ORES.

a cover of sheet silver, and above this a flat-bottomed cooling dish of brass, these three essential pieces being clamped tightly together as shown in the accompanying illustration. Thus the distillation is performed in a closed retort, which prevents the escape of mercury vapor, and renders careful regulation of the heat unnecessary. The apparatus is made by the Braun-Knecht-Heinmann Company of San Francisco, and Los Angeles.

[1] *California Journal of Technology*, Sept., 1904.

Jas. W. Howson, who has used this method extensively, describes the process as follows: "Six or seven grams of iron filings, previously washed with alcohol and heated to free them from any traces of grease, are introduced into the retort. From two grams of low-grade to 0.05 gram of 60-per cent. ore, finely ground, is now intimately mixed with the iron filings in the retort, and a cover of iron filings is added. The weighted silver foil is carefully adjusted and the cup, filled with water, is screwed on. The apparatus is now so placed that the bottom of the retort is about 1¼ in. above the top of the Bunsen burner. The retort is heated from 12 to 15 min. by a flame which envelops its lower third. At the end of this time the mercury will have been entirely volatilized from the ore and amalgamated with the silver foil. The flame is now removed and after cooling 5 or 10 min. the silver foil is taken from the apparatus and weighed. The gain in weight, divided by the amount of ore used, represents the percentage of mercury in the ore.

Owing to the great accuracy of the method and to the small amount of ore frequently taken for analysis, it is necessary to adjust the scales carefully, and to conduct the weighing of the silver foil, both before and after amalgamation, in the same manner as nearly as possible. The silver foil used is 1½ in. wide and of such thickness that a square of thin dimension weighs about 1.4 gram. It should be kept scrupulously clean and be handled as little as possible. If the ore contain more than very small amounts of water it should be dried. The presence of a dark stain on the foil along with the amalgamated mercury is usually without significance. Blanks should be run on every new lot of iron filings. In grinding ores containing free mercury especial care must be taken that none of the mercury amalgamates with the grinding or sieving apparatus. If the mercury is not completely amalgamated on the foil and shows a tendency to flow, a new assay should be made using a proportionately smaller amount of ore. The iron filings should be passed through a 40- or 60-mesh sieve.

SALT.

The production of salt in the United States in 1908 was 28,745,349 bbl., or 4,024,345 tons of 2000 lb., valued at $7,486,894. The average value per barrel was about 26c. As compared with 1907, the production shows a decrease of 974,174 bbl., or 136,384 tons, with an increase in value of $47,343. The bulk of the output came from Michigan, New York, Ohio and Kansas, named in the order of their importance as producers.

PRODUCTION OF SALT IN THE UNITED STATES. (a)

(In barrels of 280 lb.)

Year.	California.	Illinois.	Kansas.	Louisiana.	Michigan (c)	Nevada.	New York (c)	Ohio, W. Virginia and Pa. (b)	Utah.	Other States.	Total Barrels.
1900..	621,857	(d)	2,233,878	(d)	7,210,621	(d)	7,897,071	1,669,156	249,128	987,631	20,869,342
1901..	601,659	99,700	2,087,791	451,430	7,729,641	13,781	7,286,320	1,385,257	334,484	569,092	20,566,661
1902..	682,680	90,009	2,158,486	399,163	8,131,781	14,829	8,523,389	2,318,579	417,501	1,112,824	23,849,221
1903..	629,701	(d)	1,555,934	568,936	4,297,542	(d)	8,170,648	3,043,135	212,995	489,238	18,968,089
1904..	821,557	(d)	2,161,819	1,095,850	5,425,904	(d)	8,600,656	3,030,829	253,829	639,558	22,030,002
1905..	664,099	(d)	2,098,585	1 055,186	9,492,173	(d)	8,359,121	2,728,709	177,342	1,390,907	25,966,122
1906..	806,788	(d)	2,198,837	1,179 528	9,936,802	11,249	8,978,630	3,436,840	262,212	1,361,494	28,207,743
1907..	626,693	(d)	2,667,459	1,157,621	10,786,630	6,459	9,657,543	4,007,390	345,557	464,143	20,719,493
1908..	899,028	(d)	2,588,814	947,129	10,194,279	9,714	9,000,000	3,572,635	242,678	1,291,042	28,745,319

(a) Statistics of the U. S. Geological Survey except for New York during 1906, 1907 and 1908, which were taken from reports of the State Geologist. (b) The production of Pennsylvania in 1906, 1907 and 1908 is included in "Other States." (c) Includes brine used in manufacture of alkali. (d) Included in "Other States."

CONSUMPTION OF SALT IN THE UNITED STATES.

(In tons of 2000 lb.)

Year.	Production.		Imports.		Exports.		Consumption.	
	Amount.	Value.	Amount.	Value.	Amount.	Value.	Amount.	Value.
1897.....	2,236,248	$4,920,020	209,025	$565,038	5,797	$52,320	2,439,476	$5,432,738
1898.....	2,465,789	6,212,554	185,530	588,653	8,640	63,624	2,642,659	6,737,583
1899.....	2,759,206	6,867,467	189,051	579,682	12,600	86,465	2,935,657	7,360,684
1900.....	2,921,786	6,944,603	199,900	634,307	7,511	65,410	3,114,106	7,513,500
1901.....	2,879,332	6,617,449	201,733	676,324	9,433	86,414	3,071,632	7,207,359
1902.....	3,338,892	8,068,636	184,764	647,554	5,094	55,432	3,518,562	8,260,758
1903.....	2,655,532	5,286,988	165,981	495,948	12,750	95,570	2,808,763	5,687,366
1904.....	3,084,200	6,021,222	166,140	467,754	13,964	113,625	3,236,376	6,375,351
1905.....	3,635,257	6,095,922	161,159	492,189	34,238	239,223	3,762,178	6,348,888
1906.....	3,944,133	6,658,350	170,505	502,583	33,988	274,627	4,080,650	6,886,306
1907.....	4,160,729	7,439,551	153,435	452,227	30,802	232,995	4,283,362	7,658,883
1908.....	4,024,345	7,486,894	156,609	440,484	26,627	202,338	4,154,327	7,725,040

PRODUCTION OF SALT IN FOREIGN COUNTRIES.
(In metric tons.)

	1899	1900	1901	1902	1903	1904	1905	1906	1907	1908
Algeria	17,378	18,325	18,518	27,263	26,329	18,563	27,000	22,615	20,400	(c)
Austria	342,059	330,277	333,238	310,807	359,014	369,877	343,375	378,912	356,445	(c)
Canada	53,847	56,296	53,927	57,203	56,671	62,411	41,170	69,291	73,858	81,253
France	1,193,532	1,088,634	910,000	863,927	967,531	1,153,754	1,130,000	1,335,410	1,226,000	(c)
Germany	1,432,181	1,514,027	1,563,811	1,583,458	1,693,935	1,701,654	1,777,557	1,870,212	1,950,689	1,992,866
Greece	22,411	22,411	23,079	25,200	26,000	27,000	25,201	25,167	26,966	(c)
Hungary	182,593	189,363	(a)211,321	174,882	183,327	187,620	195,410	201,369	395,000	(c)
India (d)	977,240	1,021,426	1,120,187	1,056,899	908,911	1,188,900	1,212,000	1,176,324	1,212,078	(c)
Italy	28,842	367,255	435,187	458,497	488,506	464,326	437,699	496,872	505,000	(c)
Japan	390,433	669,694	659,118	620,820	657,489	701,965	483,506	484,000	(c)	(c)
Russia	1,681,362	1,768,005	1,705,922	1,847,019	1,658,938	1,908,275	1,844,678	1,730,934	(c)	(c)
Spain	598,108	450,041	345,063	426,434	427,394	543,674	493,451	541,978	605,895	(c)
U. Kingdom	1,945,531	1,873,601	1,812,180	1,924,273	1,917,184	1,921,899	1,920,149	1,996,593	2,038,072	(c)
U. States	2,522,610	2,651,278	2,612,204	2,409,174	2,408,646	2,797,461	3,297,285	3,578,061	3,773,781	4,088,733

(a) Sales by the royal monopoly, including imports entered for consumption. (c) Statistics not yet published. (d) Does not include the untaxed output of certain native States.

SALT MINING IN THE UNITED STATES.

California.—At Salton, in southern California, 265 ft. below sea level, is a salt farm of about 1000 acres. The deposit is from 6 to 16 in. in depth. Work is carried on the year round, and during the 20 years that the deposit has been operated, more than 40,000 tons of salt has been gathered from 10 acres. The salt is first plowed up in furrows, and then thrown into conical piles by men with barrows, after which it is taken to the works nearby and put into marketable condition.

Louisiana. (By G. D. Harris.)—The geological occurrence of rock salt in Louisiana and east Texas was described in a paper in *Economic Geology*, 1909, Vol. IV, page 12. This paper treats of the highly interesting geological conditions of the region, but does not go into any of the industrial phases of the deposits.

New York. (By D. H. Newland.)—The production of rock and brine salt during 1908 fell behind the record output of 1907 and was about on a par with the total for 1906, or roundly 9,000,000 bbl. Most of the decrease was in the salt obtained from wells, a considerable portion of which was consumed in the manufacture of soda products by the Solvay Process Company. The salt industry is distributed among seven counties, including Onondaga, Tompkins, Schuyler, Wyoming, Genesee and Livingston. Practically all of the different grades of brine salt known to the trade are made in the State; and most of the rock salt for the eastern markets is supplied from mines situated in Livingston county. There are at present two active mines, the one owned by the Retsof Mining Company, of Retsof, and the other owned by the Sterling Salt Company, of Cuylerville. The latter company first produced in the fall of 1906. The mines formerly worked at Livonia and Greigs-

ville have not been operated in recent years. The largest producer of brine salt for the market is the International Salt Company, with three active works situated at Ithaca and Myers, Cayuga county, and at Watkins, Schuyler county. For a long time the State of New York has held an interest in salt manufacture through ownership of lands in Onondaga county, from which the natural brines used by the solar evaporating plants around Syracuse were derived. A sale of these lands was effected in 1908, the title passing to the Onondaga Pipe Line Company, and the Mutual Pipe Line Company of Syracuse, for the nominal sum of $15,000. With this transfer, the entire control of the salt business, together with its supervision, was definitely relinquished by the State.

Ohio. (By J. A. Bownocker.)—In territorial and even later days salt was an expensive necessity in Ohio. The supply was transported across the Appalachian mountains and sold for $8 per bushel, perhaps more. Later, salt springs were discovered in what is now Jackson county, and the making of salt began. Soon wells were dug and a stronger and more plentiful brine secured, and salt making became an important industry in the State. The industry thrived until the great development in New York and Michigan which with their natural advantages, particularly their strong brine, so lowered the price of salt that the Ohio producers in general, could not meet the competition. Furnace after furnace closed until at present only four remain in this part of the State. These are on the bank of the Ohio river in the vicinity of Pomeroy. While this region has the advantage of coal in the hills right by the side of the furnaces, and the river and rail for shipping, these are more than offset by the weakness of the brine. Formerly much help was derived from the by-products, calcium chloride and bromine, but the recent low price for the latter has materially reduced the profits. It is not too much to say that it is a constant struggle to keep these furnaces in operation.

At first the brines were evaporated in iron kettles. These soon gave way to large rectangular boxes known as grainers. A few years ago many of these were equipped with self-rakes, thus materially reducing the labor. In 1906 a new furnace was completed and a novelty or two introduced. The brines are run into "evaporators," which are large iron tanks with steam pipes on the inside. Steam is supplied from boilers and the brine enters the evaporators at 9 deg. B. and leaves them at about 15 deg. B. The brine is then run into the filters, which are large tanks with crushed stone covered with sand on the bottoms. The liquid filters through these and then is run into settlers and finally the

grainers. The steam produced in the evaporators is used for heat in the grainers and settlers.

When the production of salt in southeastern Ohio had suffered a notable decline, and it looked as if the State might cease to be important in this industry, rock salt was discovered when drilling for oil and gas in the northeastern part of the State. After some years furnaces were erected. The industry has grown and at present there are two furnaces at Cleveland, one between Akron and Barberton, one at Wadsworth and one at Rittman. All of these furnaces use a saturated brine. The salt is made by the grainer and vacuum-pan process. It is the production from this field that makes the State important in the industry. The rock salt is known to underlie several counties and the supply is adequate to meet demands for an indefinite period.

Utah.—The plant of the Inland Crystal Salt Company, where salt obtained from the water of Great Salt Lake was purified, was destroyed by fire Nov. 5, 1908. Plans for rebuilding on a large scale are now under consideration.

West Virginia.[1] (By G. P. Grimsley.)—Salt making was one of the earliest industries in the West Virginia area. The first well was bored for salt in 1803, on the Great Kanawha river, six miles above Charleston; a depth of 80 ft. was obtained and weak brine, of which 400 gal. were required to make one bu. of salt, was found. Later, wells were drilled a depth of 350 ft., yielding a stronger brine, of which 75 gal. were required to make one bu. of salt. The annual production of salt in this valley for a number of years preceding 1833 was a million bushels.

According to Hildreth, the common method of evaporating the brines was in kettles which held 60 to 90 gal. each. A block of 30 or 40 kettles required five to six cords of wood. The first use of coal as fuel was in 1817, and in 1835 steam was first used in the evaporation of the brines. Down to the time of this introduction of steam in the pan method of evaporation, the progress in the work was not large. Dr. Hale, one of the pioneer salt operators, describes the industry through these years previous to 1835 as follows: "Wells were bored deeper and larger, tubing was improved, the pumps and rigs made simpler. The furnaces were constructed larger and were more effectively operated, the quality of salt improved and the quantity increased, but still the plants were kettle furnaces of the original type. The usual size of the kettle furnaces produced 25,000 to 50,000 bu. annually, but the Campbell's Creek furnace produced 100,000 bu. In 1835 there were 40 furnaces producing annually about 2,000,000 bu. of salt in the Kanawha valley."

The Mason City and Hartford districts were opened on the Ohio river

[1] Published with permission of the State Geologist.

in 1849. In 1875 there were 13 furnaces in this field, which produced that year 2,500,000 bu. of salt. While other salt developments were made in the State before 1870, only two fields have ever made any quantity of salt, and these two fields, Kanawha valley, Mason City and Hartford, are today producing salt.

In contrast with the past great development of the salt industry in West Virginia, only four plants are in operation at the present time, one situated in Kanawha valley and the others on the Ohio river, in Mason county. The numerous salt works which made the Kanawha valley famous 30 years ago have with one exception disappeared, and only here and there remains an old stack or pile of rubbish to mark their sites. The low price of salt and the discovery of richer brines in the more northern districts have forced the companies one after another to discontinue operations. The one plant which is still in operation is situated at Malden, six miles above Charleston, on the Kanawha & Michigan Railroad, and on the north bank of the Kanawha river. It was built in 1865 and is owned and operated by J. Q. Dickinson & Co.

The salt wells of this company are 800 to 900 ft. in depth, reaching the sandstone of the Pottsville series of the Carboniferous. Six wells are now used and the brine is pumped into a large wooden storage tank holding 45,000 gal. From this tank the brine flows down a wooden pipe to the furnace heated by natural gas and coal, the latter from the company's mines. Above the furnace are the three brine pans, 3 ft. deep, 30 and 45 ft. wide, and 10 ft. long, in which the brine is concentrated from a specific gravity of 1.048 to 1.063, with a salt content of 7 to 9 per cent. as received from the wells, and brought up in the furnace pans to a gravity of 1.085 and salt percentage of 12. Over the furnace-pans is constructed a tight wooden box forming the steam chest; the steam retained in this box passes through wood pipes under four pounds pressure to the settlers and grainers. The pipes from this steam chest lead to a horizontal log pipe at one end of the vats and connected by gooseneck pipes with the five-inch copper pipes running lengthwise of the vats.

The brine from the furnace pans is colored by the suspended iron oxide, mud and sand, and is drawn off into the upper side of the first mud settler. The two mud settlers are long vats constructed of heavy plank, 165 ft. long, 8 to 10 ft. wide, 18 in. deep, and divided longitudinally by a plank partition. The brine entering the upper side, flows to the opposite end and there passes over a low place in the partition into the lower side and back to the head of the vat again. It there passes through a pipe into the second mud settler, constructed on a similar plan, and follows a like course. By the time it reaches the head of the lower side of the second settler, it is perfectly clear and free from iron, mud and

sand. The brine is now ready for use and it has been kept at moderately high temperature by the steam in the copper pipes in the vats.

The brine leaves the second mud settler with a gravity of about 1.125 and flows into one of the two settlers which are of same construction as the mud settlers, except there are two partitions, thus giving a threefold division of the vat. It then passes into the fifth vat, or draw settler, without partitions, 165 ft. long, 14 ft. wide, and 45 in. deep. Here the brine is concentrated to a gravity of 1.179 and salt crystals begin to form. It is then conveyed to the four grainers, which are plank vats lined with tile plates, 150 ft. long, 10 ft. wide, 18 in. deep, and containing three copper steam pipes the full length of the vats. In these the salt is deposited and removed by rakes to the runway and hauled to the store house.

After most of the salt is precipitated and the brine has a gravity of 30 deg. B., the liquor is drawn into the tenth or bittern vat, heated by two copper steam pipes, where it is further concentrated and the rest of the salt precipitated. This salt is more or less impure and somewhat colored, and is usually sold for agricultural uses, although at this plant it is sold to the Kelly Axe factory, at Charleston, and used in their tempering mixture.

When the mother liquor has reached a gravity of 35 deg. B. it is removed to the bittern water tank, to be subjected to further treatment for extraction of bromine and calcium chloride. The daily capacity of this plant is 125 to 150 bbl. of salt, 22 bottles of bromine, or 150 lb., and 4 tons of calcium chloride.

The bittern water from the tank is carried to the bromine plant, where it is heated in a furnace tank and concentrated to 41 to 43 deg. B. The tank holds 250 to 300 gal. of bittern, and the concentrated liquor is drawn from it into stone stills, where it is mixed with sulphuric acid and potassium chlorate poured in from the top. About 25 lb. of acid and 8 lb. of potassium chlorate are used to 400 gal. of bittern; this yields about 25 lb. of bromine.

The still is made of two blocks of solid sandstone about 5 ft. square and of a total hight of 6 ft., hollowed out at the center. Steam is blown into the mixture, heating it to a temperature of 160 to 180 deg. F., and making a thorough mixture. The bromine vapor passes down through two lead pipes through a condenser, which is a rectangular wooden box, 5 ft. long and 1 ft. square, in which cold water is constantly running around the pipes. The lower ends of the lead pipes empty into two bottles with connections sealed with clay to prevent escape of fumes. In the condenser the vapor is condensed to liquid bromine, and as the bottles are filled they are sealed and new bottles added.

The residual liquor freed from bromine in the still is drawn from below into a cistern, where it is treated with lime to neutralize the acid; the liquor is pumped into the calcium kettles, two in number at this plant, but only one is now in use. These iron kettles are enclosed in steam jackets and have a steam coil in the kettle furnishing the necessary heat. Each holds three tons of calcium chloride and the liquor is heated and condensed to a thick syrup, which is then run into sheet iron drums holding 600 to 700 lb. In a few days the liquid solidifies and is ready for shipment. In removing the material for use, the drums are cut or pounded off, leaving the core of solid calcium chloride.

The West Virginia brines are different from the brines of nearly every other region in containing the bromine, and also in the absence of calcium sulphate, or gypsum. The bromine in the brine would attack iron pipes, especially as it becomes more and more concentrated, so the conducting pipes are made of logs hollowed out; in the vats copper pipes are used, thus adding to the cost of construction of a plant. The wood pipes have low radiation and heat conduction so there is a loss of heat with their use. The thin copper pipes permit the heat of the low pressure steam to be rapidly given up to the brine; the vats are readily made on the ground and have a low cost of wear and tear. The mechanical conditions are all favorable to a low cost of manufacture, but the weak brines would probably increase the cost beyond any adequate profit on the investment if the valuable by-products of bromine and calcium were absent or lost in process of manufacture. It requires skilful and careful management to operate such a plant at a profit, but this company, first organized in 1832, has been in successful operation ever since and today is looked upon as one of the prosperous industrial companies of this great valley, so rich in natural resources. The salt is shipped by rail and water to distant points. The only other salt-producing area in West Virginia is on the Ohio river in Mason county, opposite Pomeroy, Ohio, at Mason City and Hartford. These two towns are three miles apart, on the Ohio river division of the Baltimore & Ohio Railroad, and 62 miles below Parkersburg. Of the 13 plants in operation in this area in the seventies, only three are now making salt.

The only salt operation at Mason City is the Dixie salt works; this plant was built 35 years ago by the Hope Salt Company. It was closed for four years, but in the spring of 1907 was overhauled and partially rebuilt. The wells are 1250 ft. deep and the brine is found at 1100 to 1150 ft.; it is pumped by sucker rods from a depth of 600 to 800 ft., and has a gravity of 8 deg. B., or 1.056. The plan of the works and equipment is similar to the Malden plant described. There are two mud settlers, two draw settlers, five salt grainers, and a sixth bittern vat. The

bromine plant is of similar construction, but the calcium chloride is made across the river in the Ohio plant of the same company. The daily capacity is 200 bbl. of salt, and 200 lb. of bromine. The fuel used is coal from the company's mines just back of the town; this is brought down a long incline track to a coal storage shed above the level of the railroad. From this shed the lump coal is loaded into the railroad cars for shipment, while the slack and poorer coal is used at the furnace. The coal mined is the Pittsburg seam, 4 to 5 ft. thick. One bushel of coal is said to make one bushel of salt.

The works of the Hartford City Salt Company is located at the upper end of the town of Hartford City, between the railroad and the river. The brine is pumped from five wells drilled to the Big Salt Sand of the Pottsville series (Carboniferous), into the storage tanks located on the hill above the plant and flows down by gravity. There are two mud settlers, two draw settlers, 145 ft. long and 12 ft. wide, and six grainers, 126 ft. long and 12 ft. wide, equipped with automatic salt rakers which push the salt forward at the rate of 4½ ft. per min., discharging it on a conveyer belt which carries the salt to the storage house. In other respects the equipment is similar to that of the other plants. The daily capacity of this plant is 250 bbl. of salt, 125 lb. of bromine, and 5 to 6 tons of calcium chloride. The Pittsburg coal from the mines near the works is used as fuel.

The works of the Liverpool Salt and Coal Company is located at the lower edge of the town of Hartford and near the railroad. The wells are 1100 to 1200 ft. deep, with 600 ft. of brine in them; the horizon is the same as at the other plants. A different method of concentrating the brine is used here. Ordinary tubular boilers, 72 ft. long and 16 ft. in diameter, are used, with 10 tubes in which the brine is concentrated and carried to the grainers. The daily capacity is 350 to 400 bbl. of salt, 350 lb. of bromine, and 3 tons of calcium chloride; the Pittsburg coal is used as fuel.

As has been stated, one of the peculiar properties of the West Virginia brines is the absence of calcium sulphate. They also contain barium, which explains the absence of the gypsum, for the latter could not remain in solution with an excess of barium chloride. The following analyses, made by C. D. Howard, of the State Agricultural Station, show the composition of the brines at Malden and at Hartford; the figures show parts per 1000 by weight, the first being for Malden brine and those in parenthesis being for Hartford brine; Sodium chloride, 60.1762 (74.191); sodium bromide, 0.2238; sodium iodide, 0.00098; sodium nitrate, none; sodium arsenite, 0.0000097; sodium borate, trace; potassium chloride, 0.57 (0.415); lithium chloride, 0.101 (0.0093); ammonium chloride,

0.0787 (0.13); strontium chloride, 0.2013; calcium chloride, 15.0103 (12.554[1]); magnesium chloride, 4.9692 (5.438); calcium bicarbonate, 0.2661 (0.171); ferrous bicarbonate, 0.0822; manganous bicarbonate, 0.0076; oxides of iron and alumina, (0.048); calcium phosphate, 0.0005; alumina, trace; silica, 0.004; total, 83.3456 (93.2973); specific gravity at 15.5 deg. C., 1.628 (1.0732). An analysis of the salt from Pomeroy, Ohio, opposite Mason City, is given by Prof. John Bownocker as follows: Moisture, 7.42 parts per 1000 by weight; sodium chloride, 91.31; sodium sulphate, 0.00; calcium chloride, 0.95; magnesium chloride, 0.32; silica, iron and alumina, none.

The uses of West Virginia and other salt at the present time may be grouped as follows: *Domestic*—Table, cooking, preserving, pickling, dairy, ice-cream; *Industrial*—In refrigerator cars, meat packing, thawing of ice on railroads and car tracks, salting hides, fertilizer; *Mining*—Chlorination of gold ores; *Chemical*—Manufacture of soda ash, salt cake, caustic soda, chlorine and carborundum.

The production of salt in West Virginia amounts to about 200,000 bbl. a year with a value of $50,000 to $60,000.

Technology.

Mining of Salt.—An invention of H. Frasch, of Cleveland, Ohio, (U. S. Pat. 874,906, Dec. 24, 1907) is described in the *Journ.* of the Society of Chemical Industry, March 31, 1908. The invention consists essentially in introducing into the salt deposit, unsaturated brine heated to a temperature as high above that of the atmosphere as is economically possible, pumping out the brine when saturated, recovering a portion of the dissolved salt from it by partial evaporation and cooling, heating and diluting the residual brine, and then introducing it again into the salt deposit. The partial evaporation of the charged brine is effected at the expense of its own heat, whereby it is also cooled, and preferably the aqueous vapors given off during the evaporation are removed by exhaustion. The residual brine, before being again introduced into the deposit, is heated to the desired temperature, at least partially, by condensing steam in it, thus diluting it at the same time. The heating may be effected either at atmospheric or at increased pressure. It is recommended to work with exhaust steam at atmospheric pressure. If the deposit contain gypsum, the temperature of the unsaturated brine introduced into the deposit should be well above 32 to 41 deg. C., the temperature of maximum solubility of calcium sulphate, say 100 to 104 deg. C., and the charged brine should be cooled during the partial evaporation to a temperature near 32 to 41 deg. C., say 27 deg. C. In this way the separa-

[1] Strontium chloride and calcium chloride combined.

tion of calcium sulphate during evaporation is almost entirely prevented. The invention, which can be applied to other similar processes besides the recovery of salt, is claimed to possess the following advantages: (1) No heat-conducting walls (steam pipes, etc.) are used for the transmission of heat in the evaporation of the charged brine, and thus objectionable incrustations are avoided. (2) The column of unsaturated brine entering the deposit more nearly balances the ascending column of charged brine, than would a column of water, and thus entails less strain on the pumping machinery. (3) More salt is recovered than by the usual process for a given consumption of fuel and water.

In the process described above, the separation of salt from the saturated brine, without the application of external heat, is small, and the underground deposit is subjected, therefore, to the solvent action of nearly saturated brine, which is so slow that, if the deposit is newly opened, or the cavity small, the usefulness of the process is impaired. It is proposed to supplement it as follows: (U. S. Pat. 874,907, Dec. 24, 1907.) The temperature of a stream of brine, circulating above ground only, is raised at one point by the surface condensation of steam in chambers jacketed or bathed by the brine. This promotes evaporation of the brine and gives a supply of hot condensation water, which is introduced into the deposit below ground, while the cavity is small, instead of the unsaturated brine previously recommended, which latter may, however, be employed with success as the size of the cavity increases. Evaporation of the hot brine from the deposit is brought about partly by utilizing its own heat, as by forcing it through sprinklers in a well-ventilated chamber, and partly by heat from steam or from the previously heated residual brine. Any calcium sulphate deposited on the heat-conducting walls is removed by occasionally reversing the relative positions of the steam and brine with respect to them.

The Pumping of Brine, and the Manufacture of Common Salt.— Under the above title, Philip M. Pritchard presented a paper before the 1907-08 meeting of the Institution of Civil Engineers, London. Mr. Pritchard describes the more usual methods of obtaining brine and of the manufacture of white salt. He states that the method adopted for mining salt, or more generally for pumping the brine, will be determined by: (1) Depth of the salt below the surface. (2) Presence or absence of brine runs on the rock head. (3) Presence or absence of water in the strata above the rock salt. The various methods of pumping the brine or mining salt may be classified under the following heads: (1) Mining of the rock salt. (2) Direct pumping of brine from natural brine runs. (3) Pumping brine which has been made by allowing water to come into contact with the rock salt. This water may occur

in strata above the rock salt, or may have to be obtained from a distance and run down the wells. (4) "Forcing wells." In this case the water is pumped down a well with a closed top into the rock with sufficient pressure to deliver the brine made in the well to wherever the brine is wanted. (5) Raising the brine (either natural or artificial) by means of compressed air by what is known as the air-lift system. The boring and tubing of wells, and the causes of well failures, are discussed fully. Methods of brine evaporation in general use are discussed under the following heads: (1) Coal-fired open pans. (2) Steam-heated pans with either bottom heat or steam pipes immersed in the brine. (3) Vacuum evaporators. The paper is accompanied by six sheets of drawings.

Salt Manufacture.—At the mid-October, 1908, meeting of the American Society of Mechanical Engineers, George E. Willcox presented a paper on "Mechanical Methods and Engineering Features of Large Salt Plants." This paper noted, from a mechanical engineer's point of view, a few of the more recent developments in the mechanical methods and appliances of some of the large salt plants. Discussion was limited solely to plants operated by what is known as the steam grainer system, as distinguished from the vacuum-pan system, and the solar or open air system. The paper discusses the general arrangement of a grainer plant, salt grainers of reinforced concrete, details of concrete grainer construction, rakers, belt and reciprocating conveyers, and apparatus for loading salt barrels into cars.

The Fusion Process.—Until quite recently the manufacture of white salt has been performed entirely by the evaporation of brine process. Experiments have lately been made (*Chem. Trade Journ.*, May 23, 1908) for the direct production of white salt from rock salt by what is known as the fusion process, in which rock salt is fused or melted and the impurities separated while the salt is in a molten state. From a brief consideration of two points, viz., melting and volatilization temperature, and the degree of heat conductivity of salt, two great difficulties are found: (1) The temperature must be so regulated as to get rapid melting without excessive volatilization. (2) The molten salt bath must be very shallow, but must not be exposed to the very high temperature of a melting furnace. It is, moreover, of the greatest importance to select properly the materials employed in the construction of the plant, for the presence of, for example, iron in the brick-work of the melting furnace may cause the salt to be of a dark color after the solid and organic impurities have been separated. Proposals as to the form of a melting furnace have been varied, and include an ordinary regenerative furnace as employed in steel manufacture, and a furnace into which

the salt, after being finely ground, is injected. If an ordinary furnace is employed, the molten mass is subjected to the same temperature as the rock salt being melted, and as such temperature may be as high as 2000 deg. F., it will readily be appreciated that excessive volatilization and consequent loss of salt will ensue. In addition to the above points, there are other difficulties to be met with. Proposals have been made and patents secured on different methods to perfect the process, but success has not yet been attained. The fusion process is yet in its infancy, but it is asserted that considerable progress will be made in the near future, and that it is likely to prove a formidable competitor of the brine process, especially in connection with the production of the chemical quality of salt.

Salt Refining at Salt Air.—The process of refining salt as practiced near Salt Air, Utah, is a continuous one, the salt being fed to a rotating cylinder about 30 ft. long by 6 ft. in diameter; in the inner part of this cylinder is fixed a second iron cylinder, about 12 to 16 in. in diameter, which is charged with superheated steam. The water vapor is carried off by an exhaust fan, as is also the fine dust of magnesium chloride and magnesium sulphate. In this process the magnesia salts are not entirely removed. For table use, the material has to be treated by a second and third grinding, sifting and blowing.

SILICA.

Among the many materials which may properly be classed under silica, may be mentioned the following: Silica sand, quartz, flint, infusorial earth, diatomaceous earth, tripoli, ganister, and pumice. The chief use for pure silica sand is in the manufacture of glass; the poorer grades are used largely in masonry. The bulk of the flint consumed in the United States is imported from England and France. This is used for pottery work and as flint pebbles for grinding in ball mills. Domestic flint, as amorphous and crystalline quartz, is found in Maryland, Pennsylvania, Alabama and Connecticut. Ganister is a term applied to a silicious fire-clay that is used largely for lining furnaces and converters.

Infusorial and diatomaceous earths include various porous earths composed chiefly of organic skeletons. Material of this class is to be found in the coastal plain area of Virginia, Georgia and Alabama, in Missouri and Illinois, and in California. Of the foreign countries, Germany has the largest deposits; the material is mined and shipped to all parts of the world. Heretofore, infusorial earth has been largely used for abrasive purposes, in the form of polishing powders, scouring soaps, etc., and in the manufacture of dynamite, for which purpose it acts as an absorbent of nitroglycerine. This same quality renders it a non-conductor of heat, and its use has extended to packing materials in safes, steam pipes and boilers, and as a fire-proof material in general. Some of the California product is quarried for building-stone.

The Missouri tripoli finds its most important use in the manufacture of filter stone; however, much of it is ground and utilized as an abrasive. The Illinois product, which is very similar to the Missouri product, is employed in the paint industry, as a wood-filler, for enameling purposes, etc.

Flint Ballast in Nebraska. (By E. H. Barbour.)—The flint beds bordering the Blue river in Gates county were neglected prior to 1895, the occurrence of two layers of flint rendering the Permo-Carboniferous limestone seemingly worthless. The upper layer of flint is about 3 ft. thick and is separated from the lower layer by 2 or 3 ft. of limestone. Below this is found from 8 to 9 ft. of cherty limestone in which flint nodules are almost continuous. Quarrying began in this district in

1895, but it was not until seven years later that the industry was stimulated by the introduction of crushing machinery. Since then three crushing plants have been built and an industry of local importance has been developed. The product finds a ready sale, and already about 500 miles of railroad have been ballasted with the material. Also large amounts have been used in the various cities of the State in concrete and asphalt work. The average output is about 200,000 tons of screened ballast per year, worth from 60 to 65c. per ton. This, however, does not represent the full capacity of the three plants and does not include the output of many tons of rough rock. The industry gives employment to 200 or 300 men, according to the demand for the material.

Sand-lime Brick.

The value of the sand-lime brick produced in the United States in 1908 was $961,226, and represented the output of 87 plants, according to Jefferson Middleton of the U. S. Geological Survey. In 1907 the output was valued at $1,225,769 and was furnished by 94 plants. As in previous years, by far the greater part of the total value in 1908 was represented by the common building brick. Fancy brick and block showed a slight increase in value. More than 30 States reported an output of sand-lime brick in 1908, Michigan and Florida leading the list in the order named. The production of Michigan was valued at $138,809; that of Florida at $117,040. Montana appeared in the list of producers, and Alabama dropped out. The value of the production from Pennsylvania showed an increase, while that of California, Indiana, Iowa, Michigan and New York showed decreases. The largest number of plants was reported by Michigan, in which State 10 were in operation. The average price per thousand received for common sand-lime brick in 1908 was $6.33, as compared with $6.61 in 1907. For front brick the average price was $12.76 in 1908, as compared with $10.96 in 1907.

New Jersey. (By Henry B. Kümmel.)—Sand-lime bricks were manufactured by four companies during 1908, the factories being situated at Passaic, South River, Penbryn and Atlantic City. Several additional plants were in process of construction, and may become producers in 1909. The total value of brick sold was $20,819, divided into 840,000 common brick, valued at $6270, and 1,192,000 front brick valued at $14,549. The average value of the common brick was $7.46 per thousand; that of front brick $12.20 per thousand.

Pumice.

The great bulk of the pumice in the United States lies in the States west of the Missouri river. Deposits are found along the Pacific Slope

and throughout the Rocky Mountain country. Pumice is used as an abrasive by manufacturers of furniture, wood, metal, glass, bone, and many other articles; as a filler for paint; and extensively, by compounding with other articles, for scouring, cleaning and polishing materials. The principal market for the material is east of the Mississippi river. Some pumice is imported, chiefly from Italy and Sicily, the material coming over as ballast in ships. The material enters this country in two forms, unmanufactured and manufactured. The unmanufactured is in rock form and is admitted at a duty of 50 per cent. of the appraised value at American ports. The manufactured product, which has been ground, graded and sacked, is admitted upon the payment of a duty of $6 per ton. The consumption of pumice in the United States is about 20,000 to 22,000 tons per year.

Nebraska. (By E. H. Barbour.)—The beds of volcanic dust, or native pumice, especially common in western Nebraska, attracted much attention during 1908. In places, especially in Harlan county, stripping was completed, and from 8 to 10 acres of a good quality of material exposed ready for loading on the cars. A large acreage of land was purchased by those interested in the industry. Exclusive of the small producers, one firm produced and sold 5,000,000 lb. in 1906, 15,000,000 lb. in 1907, and 22,000,000 lb. in 1908. The chief use of the material is for soap and other scouring preparations for kitchen-ware, glass, etc., but in addition it is used as commercial pumice for polishing purposes, also as the base for dustless sweeper, condition powders for stock, etc. Machinery is being installed for grading this natural pumice, and an attempt will be made to introduce it throughout the United States in competition with the pumice of Sicily. The material appears under a variety of trade names.

SILICON.

The Carborundum Company was the first to produce pure silicon commercially, the process for its manufacture having been developed by its works manager, F. J. Tone (U. S. patents 745,122, 833,427, 842,273 and 869,276). The production is now carried on by the company on a large scale.

The furnaces used are large arc furnaces having two depending vertical electrodes extending a considerable depth into the charge, which consists of coke and sand. The furnace structure is of fire brick, lined on the interior with carbon. Each furnace has a capacity of 1200 h.p. and the metal is tapped out at intervals of a few hours in pigs of from 600 to 800 lb.

Silicon is made in grades ranging from 90 to 97 per cent. pure. The following are typical analyses of some of these grades: I. Silicon, 90.60; iron, 6.80; manganese, 0.08; aluminum, 2.35; phosphorus, 0.02; carbon, 0.22. II. Silicon, 95.71; iron, 2.24; aluminum, 1.96; phosphorus, 0.01; and carbon, 0.08. The principal use of the 90 per cent. silicon is in the refining of steel where it replaces the higher grades of ferro-silicon. It accomplishes the deoxidation of the metal in the most efficient manner. About 500 tons of 90 per cent. silicon metal were sold for this purpose during 1908.

Another important use of silicon is as a reducing agent in the manufacture of low-carbon ferro-alloys, such as ferro-vanadium, ferro-chromium and similar alloys. Silicon has a greater deoxidizing capacity per unit of weight than aluminum and is, therefore, very efficient for this purpose. An alloy of calcium and silicon which is now being introduced into the steel trade for the refining of steel is produced by the reduction of calcium oxide by silicon. This has called for a small tonnage of the material and will doubtless increase.

With constantly decreasing cost, silicon is destined to become a metal of common use. When the problem of its manufacture was first undertaken it sold at $4 per lb. It is now marketed in carload lots at about $120 per 2240 lb. Its use will be greatly extended when certain problems are solved in connection with its industrial applications. One of the most important of these problems is the production of silicon castings. Such castings would be of great utility as acid containers and for

the manufacture of various kinds of apparatus which are subjected to the action of acid and are widely used in the chemical industries. There will undoubtedly be a large market for such silicon or silicide containers when the technical difficulties of casting silicon have once been overcome. Silicon is now produced in the form of small castings, such as rods for electric resistances, but it is difficult to cast perfect forms unless the section is thin. Castings of considerable thickness are porous and subject to shrinkage strains. Resistance rods ¼ in. in diameter are easily cast in lengths of 12 in., and several of these rods may be electrically welded together to form longer resistances.

G. O. Seward and F. von Kügelgen recommend (U. S. pat. 916,793, March 30, 1909) an arc furnace, without the use of any fluxes, for the production of silicon. A 15,000-amp. 30-volt arc is used with an electrode of 4 sq.ft. cross-section. The charge is a coarsely ground mixture of flint rock and carbon, preferably merely crushing the flint rock to about 1-in. size and grinding the coke so that it shall pass a 10-mesh screen. The mixture is kept piled about 18 in. above the lower end of the electrode, so that the arc is maintained in the midst of the mass of porous mixture. At starting a charge is made up of about 75 parts by weight of flint rock to 25 of coke, so as to introduce a surplus of silica, afterward changing this to 70 parts of flint rock to 30 of coke. There is no excess of carbon, or barely enough to maintain a reducing atmosphere around the incandescent zone of the arc. The temperature maintained under these conditions is above that at which silicon carbide remains stable, so that, even if this material is formed in part, it is immediately decomposed. The silicon is reduced in the form of vapor and expands into the interstices of the porous mixture, where it condenses. These interstices being full of a reducing gas, namely, CO, allow the silicon to condense without being oxidized. The reduced silicon percolates down through the porous mass and collects at the bottom of the crucible. It is tapped out into carbon molds from time to time. A reduction zone is thereby maintained at a temperature so high that the silicon carbide cannot form, or, if formed in part, is decomposed. Around this zone of reduction a cooler porous condensing zone is maintained, the interstices of which are kept filled with a reducing atmosphere which prevents the vapor of silicon from oxidizing before it can condense.

SILUNDUM.

This new product of the electric furnace is thus described in *L'Industria,* Dec. 27, 1908, and *Min. Journ.*, Jan. 16, 1909. In his work on the electric furnace, Moissan lamented the porousness of the carbons which are employed as electrodes, and it was proposed to cover them with silicon carbide or boron, but experiments were not successful. In 1904, F. Bolling resumed study of the problem, and succeeded in partially converting the carbon into carborundum, whether in form of bars, tubes, or crucibles. The product is specially suitable for resistance furnaces to a temperature of 1700 deg. C. Being free from secondary conductive materials, it is not subject to oxidation in the air.

To explain the genesis of silicon carbide, it was admitted that it must be formed when the silicon and carbon are heated to sublimation temperature. Bolling found that silicon fumes at 1800 to 1900 deg. C. penetrate carbon and thus convert the electrodes into silundum; it suffices to treat them in an electric furnace in presence of carborundum or with a mixture of silicious sand and carbon. The degree of siliconization of the carbon depends upon duration of heating, temperature, and volume of the objects treated. Their form is not changed, and the trade mark stamped upon them remains in evidence. As the transformation first occurs on the surface it is possible to make tubes of various thickness by treating sticks of carbon from which the central part has been removed. Thus the sides will be formed entirely of silundum, which is not combustible in ordinary circumstances. The process can also be applied to filaments for incandescent lamps, work being easier according to the greater degree of porosity of the carbon. However, as the globes of lamps with these filaments become coated with a brown deposit, they cannot be employed. In the case of compressed objects, work can be accelerated by mixing carborundum dust with the paste before molding and treatment in the furnace.

Silundum produced as described is very hard, does not melt at very high temperatures, and resists the action of acids and chlorine. It is an electric conductor, though perceptibly less so than carbon. Hardness is proportional to the amount of silicon it has fixed. Heated to 1750 to 1800 deg. C., it disengages fumes from the silicon which ignite in the air. The resistance it presents to passage of a current depends not only upon the

amount of carbon and fixed silicon, but also upon its porosity. The first application of silundum was for construction of electric cooking ranges and stoves, as a substitute for metallic resistances, which become oxidized when platinum is not employed. A stick 800 mm. long and 6 mm. in diameter can support a continuous current of 4 kw. and temporarily 8 kw. For practical purposes the silundum sticks are 5, 6, 7, 8, 10 mm. in diameter and arranged in form of grating with the ends cemented together with a refractory substance. As silundum is not attacked by chlorine, it is useful for electrodes for bleaching apparatus. Silundum objects can be nickel-plated and enameled, but as they are not proof against fused metals, especially iron, the interior of furnaces for treatment of these materials should be preferably formed of carbon. Even siliconized coke might find useful applications, owing to the ease of its production, the percentage of silicon in the coke attaining as much as 69 per cent.

SODIUM AND SODA SALTS.

It is now several years since sodium has been an important commercial metal. In THE MINERAL INDUSTRY, Vol. XVI, nine works engaged in its production were enumerated in Great Britain, Germany, France and the United States. The world's production of sodium in 1907 was estimated at 3500 tons, but other authorities put it as high as 5000 tons. The capacity for production at the end of 1907 was said to be approximately 10,000 tons. We are unaware that there was any further increase in 1908.

The production of sodium in the United States, where two companies are engaged in the business, was approximately 2000 tons in 1907, valued at 25c. per lb. The production in 1908 was practically the same; i.e., 2000 tons, valued again at 25c. per lb. The producers were the Electrochemical Company, of Niagara Falls, N. Y., and the Virginia Electrolytic Company, of Holcomb Rock, Va. The former uses the Castner process, which consists essentially of the electrolysis of molten caustic soda. The Virginia Electrolytic Company uses a process of its own, which is based upon the electrolysis of molten sodium chloride, chlorine gas and metallic sodium being obtained directly.

Castner Process.—In the Castner process molten caustic soda is brought into an iron cauldron, set in brickwork and provided with an extension adapted to receive the negative electrode. Suspended directly above the cathode is an iron vessel attached to a lid; to its lower edge is secured iron wire gauze, which when the receptacle is in position, completely surrounds the cathode. The positive electrode is connected with the lid of the vessel, which is provided with openings for the escape of the gases resulting from the electrolysis, and is suitably insulated.

As the electrolysis proceeds, the alkali metal, being much lighter than the molten electrolyte, rises from the negative electrode and passes into the receiver, the gases escaping round the edges of the cover. The molten metal collects on the surface of the bath, and is removed by means of a large perforated spoon. As the several vessels are thus skimmed in succession the fused sodium is collected into an iron vessel, whence it is poured into molds of the size and shape of an ordinary building brick. Fresh melted caustic soda is added to the vessel from time to time to replace the metal removed, and in this manner the process is made continuous. The ingots, after being trimmed to remove adherent oxide, are immersed in

paraffin oil, and are then packed into large iron drums, holding 6 or 7 cwt., capable of being closed air-tight, which are protected in transit by an outer casing of wood.

Uses.—The major part of the sodium made in England is sent to Glasgow, where it is converted into sodium cyanide by the Cassel Cyanide Company. This is also the chief consumption of the sodium made in the United States. Other important uses of sodium are in the manufacture of peroxide for bleaching purposes, of artificial indigo and of a number of other synthetic dye stuffs and of drugs like antipyrin. For further statistics and data reference should be made to THE MINERAL INDUSTRY, Vol. XVI.

According to the Roessler & Hasslacher Chemical Company, recent experience in connection with the drying of transformer oils has shown that by the use of metallic sodium, moisture may be completely removed from oils which are to be used for insulating purposes. This means, of course, that the insulating qualities of the oil will be raised in a very marked degree. The method of using sodium for drying hydrocarbon oils is one that is familiar to every chemist who has ever measured the di-electric constants of these oils where the sodium is usually added in the final operation to remove the very last traces of moisture. In the fear that the caustic formed by the reaction of the moisture on the sodium might be left as a liquid or a solid in the hydrocarbon, the chemist has usually thought it necessary to distill off from the sodium. In working on a very large scale this fear has been found to be groundless, and it is absolutely unnecessary to distill the oil.

Sodium has a density of about 0.97 at 15 deg. C. and melts at 97 deg. C. In transformer oil it will sink unless dragged to the surface by hydrogen gas. With water it reacts to form caustic soda and hydrogen. If very much water is present the caustic soda dissolves and in the presence of oil forms a second layer. If very little water is present the caustic soda is formed on the surface of the metallic sodium and may be removed when removing the sodium. When the sodium surface becomes covered with caustic, it is advisable to remelt under oil not letting the temperature rise above 120 deg. C. After cooling and getting into the shape desired, it is again ready for use for drying more oil.

"One method which we have used with our own transformer oils has been the following: The oil on the granulated sodium is poured off and a good transformer oil poured over the metal. To the oil which is to be treated, and which is put into an open tank or barrel, the sodium is added at first very carefully, about 1 oz. to the barrel. If much hydrogen is evolved, this will be conclusive proof that there is much water in the oil, and the remainder of the sodium should be added carefully and in

small amounts. The amount which is to be added depends upon the oil, but as a rule 1 lb. to the barrel is usually much more than is required. The oil is then stirred up three or four times a day for a minute at a time. After several days the oil may be removed and tested, but the longer it remains over the sodium the better the oil becomes. Another method used is to put the sodium in the form of sticks in a cylinder of iron wire of about 28 mesh and hang the cylinder in the oil. This method may be used directly with the static which is in use, the only precautions required being those familiar to all electricians in the avoidance of short circuits. We have repeatedly taken oils which broke down at 3000 volts and by letting them stand over the sodium for two days have brought their breaking points up to 20,000 volts or higher."

Sodium and Potassium.—According to G. F. Jaubert (*Ber.*, 1908, XLI, 4116-20) by the action of sodium on potassium hydroxide or of potassium on sodium hydroxide, liquid alloys of the two metals, containing up to nearly 80 per cent. of potassium are obtained. The reaction is effected in a crucible under a layer of paraffin wax, or in vacuo, and according to the temperature and to the proportions of the reacting substances, the products contain from 62.9 to 77.2 per cent. of potassium, corresponding to the formulas, NaK and NaK_2; the yields are almost theoretical. The alloy, NaK_2 is prepared on a commercial scale at Clavaux (Isère), France, by heating together sodium and anhydrous potassium hydroxide to 350 deg. C. in an iron autoclave.

Soda Salts.

By Albert H. Fay.

The year 1907 was a very difficult one for nitrate producers. Apart from the situation as affected by the financial crisis, the year 1908 was much better. With a cessation of the strike, which for so long a time paralyzed the industry, working costs were adjusted on a somewhat lower level. The nitrate output of Chile in 1908 was largely increased. The extra demand in Europe was offset by a reduction of shipments to the United States, these having fallen off 15 per cent. Stocks at the end of the year were estimated at 47,000 tons in the United Kingdom, 354,000 tons in Continental ports, and 20,000 tons in the United States and other ports; and afloat 510,000 tons for Europe.

The supply of raw material is very large and since the scarcity of labor has been overcome and wages, coal and bags have become cheaper, there is every inducement to keep up a large production. During the last 12 months the loading facilities at the various harbors have been greatly improved. As a result of the combination of nitrate producers,

the industry has enjoyed a term of prosperity and prices have gradually advanced. Negotiations for a renewal of the combination, which expires March 31, 1909, are going on actively. It is believed that the producers will come into line rather than have free production, the immediate effect of which would be a loss of confidence by the whole trade. Endeavors to form some scheme for controlling sales were instituted by a section of producers, but after careful examination were given up as impracticable.

Owing to the moderate price, abundance of supply, and a fair agricultural outlook, it is expected that the demand for fertilizer will continue to grow steadily. At present prices many producers are unable to work at a profit, while the more moderately capitalized companies are securing a fair margin at the price. Steps have been taken in one or two instances to effect an amalgamation of companies which are closely connected, thus meaning economies in working and saving of expense.

NITRATE OF SODA STATISTICS. (a)
(In tons of 2240 lb.)

Year.	Shipments from South America.	Consumed in Europe.	Consumed in United States.	Consumed in World.	Stocks in Europe.	Visible Supply at close of year.
1899	1,373,000	1,140,000	160,000	1,330,000	236,000	741,000
1900	1,429,000	1,126,000	175,000	1,324,000	221,000	794,000
1901	1,238,000	1,154,000	192,000	1,364,000	243,000	617,000
1902	1,360,000	1,028,000	214,000	1,259,000	263,000	660,000
1903	1,435,000	1,127,000	265,000	1,412,000	155,000	654,000
1904	1,476,000	1,131,000	275,000	1,447,000	162,000	672,000
1905	1,623,000	1,190,000	308,000	1,547,000	183,000	674,000
1906	1,700,000	1,243,000	355,000	1,636,000	190,000	733,000
1907	1,628,000	1,257,000	349,000	1,662,000	202,000	691,000
1908	1,993,000	1,376,000	309,000	1,730,000	401,000	911,000

(a) Statistics of W. Montgomery & Co., London.

IMPORTS OF SODIUM NITRATE INTO THE UNITED STATES.(a)
(In tons of 2240 lb.)

Year.	Quantity.	Value.	Value per ton.	Year.	Quantity.	Value.	Value per ton.
1899	146,492	$3,486,313	$23.80	1904	228,012	$9,333,613	$40.93
1900	182,108	4,935,520	27.10	1905	321,231	11,206,548	34.89
1901	208,679	5,999,098	28.75	1906	372,222	14,115,206	37.92
1902	205,245	5,996,205	29.21	1907	364,610	14,844,675	40.71
1903	272,947	8,700,806	31.88	1908	309,023	11,386,393	36.84

(a) As reported by the Bureau of Statistics, Department of Commerce and Labor. The figures of value appear to be doubtful, especially with respect to the earlier years.

Sodium Salts in the United States.

There is a large production of soda and other sodium salts in the United States by concerns which employ common salt (sodium chloride) as the initial product. Deposits of natural sodium sulphate and sodium

IMPORTS OF SODIUM SALTS. (a).
(In tons of 2000 lb.)

	1904		1905		1906		1907		1908	
	Tons.	Value.	Tons.	Value.	Tons.	Value.	Tons.	Value.	Tons.	Value.
Arseniate	5.7	$ 345	25.5	$ 1,745	79.9	$ 5,902	84.2	$ 9,306	159.1	$ 13,922
Ash	9,885	188,537	9,012	167,088	5,103	98,714	3,373	73,052	1,974	41,607
Bicarbonate	118.9	6,945	173.3	6,858	159.9	6,075	68	4,274	42.6	2,555
Bichromate and Chromate	47.9	4,381	56.7	5,449	6.6	584	3.5	425	54.0	4,304
Caustic	876	49,535	636	35,294	671	36,841	642	39,396	437	26,079
Carbonate (Crystal)	261	6,959	219	6,350	111	3,045	63	2,026	58	2,692
Chlorate	684	67,832	143	12,309	58	7,032	642	39,396	.....	.....
Chloride (Salt)	177,902	518,949	163,203	506,198	163,683	488,898	155,318	474,833	168,686	471,476
Hyposulphite	1,056	23,235	502	10,436	137	2,858	35	1,093	6	445
Nitrate	297,864	9,260,808	282,692	9,557,522	373,988	13,118,214	342,086	14,041,346	346,105	11,386,393
Nitrite	247	23,788	314	29,590	429	40,751	389	37,623	100	10,015
Phosphate	249	8,583	40	1,462	65	2,428	14	855	13	619
Prussiate	928	136,599	905	125,230	884	118,153	1,074	175,674	852	129,082
Sal Soda	1,850	20,918	1,484	18,470	498	7,381	380	4,771	249	3,419
Salt Cake	1,031	12,089	1,382	15,738	1,808	15,997	3,044	37,044	207	2,169
Silicate	378	9,188	552	13,434	681	13,504	618	11,461	744	13,376
Sulphide	107	2,656	359	10,339	1,024	29,835	723	20,988	300	9,539
Sulphite	45	1,947	29	1,168	78	2,851	12	803	58.4	3,259

(a) From the *Oil, Paint and Drug Reporter*, Feb. 22, 1909.

carbonate occur in Wyoming and California, but they are not utilized to any considerable extent, owing to their remoteness from the markets, which are principally in the East. So far as we are aware, there was no production of sodium sulphate or natural soda in the United States in 1908. The sodium sulphate in Soda Lake, San Luis Obispo county, Cal., was described by Ralph Arnold and H. R. Johnson, in "Contributions to Economic Geology," 1908, published by the U. S. Geological Survey. This deposit offers a very large supply, but its profitable exploitation is dependent almost entirely on transportation facilities, which at present are inadequate. The exploitation has been undertaken by the Carisa Chemical Company, which expects to haul the product by traction engines to Hazelton, near the terminus of the Sunset branch of the Southern Pacific Railroad, 32 miles distant. However, the operations of the company were temporarily suspended in 1908.

THE MARKET FOR SODA SALTS IN 1908.[1]

Nitrate.—The prices quoted in the New York market at the beginning of 1908 were $2.35@2.37½ for spot, and $2.40@2.45 for futures. The prices at various other ports were as follows: Boston, spot $2.47½; Philadelphia, Baltimore, Norfolk, Wilmington, Charleston, Savannah, $2.41½; New Orleans, $2.40. The quotations for 95 and 96 per cent. were from 2½@7½c. higher. By the middle of February there was an advance of about 2½c. per lb. Toward the end of February there were further

[1] From the *Oil, Paint and Drug Reporter*, Feb. 22, 1909

advances. The market through the latter part of May and early June was somewhat irregular and prices dropped to $2.25½@2.30. The market was weak during August and September showed still further decline, spot prices dropping to $2.12½@2.15 with futures at $2.12½. No improvement was noted through October so far as distributing demand was concerned, but a slightly stronger position was maintained abroad. By the end of 1908 the market had strengthened slightly, closing at $2.17½@2.20 for spot and futures. The average price for nitrate of soda ruled at about 10 per cent. below that of 1907.

Prussiate.—The market was rather dull all through 1908 and the lowest price was reached that has been noted in recent years. During March the price was 9¼c.; May, 8¾c.; August, 8½c.

Bichromates.—The aggregate business during 1908 was somewhat lighter than in 1907. As prices for this commodity are fixed each 12 months the market was deprived of any interesting features. The year opened with prices at 7⅜@7¾c. The sales were confined principally to immediate requirements. The figures to rule for delivery during 1909 are 6½@⅝c. per pound.

Chlorate.—The depression during the first six months of 1908 in the textile industry had a bad influence on the withdrawal of chlorate of soda, but as conditions improved during summer and fall the losses were largely offset. On Friday, Oct. 30, 1908, manufacturers announced delivery prices for 1909 as follows: Chlorate of soda remaining the same at 8½@9¾c. per lb., f.o.b. works according to quantity.

Caustic Soda.—The sales were restricted largely to regular contract delivery and the distribution is said to compare favorably with that of 1907. Sales were made at $1.75@1.85 per hundred on the basis of 60 per cent.

SULPHUR AND PYRITES.

By John Tyssowski.

Most of the sulphur produced in the United States is furnished by the Union Sulphur Company, of Louisiana. Sulphur is also produced in California, Colorado, Wyoming, Utah and Nevada, but in none of those States has the industry attained large proportions. This is to a large extent because of the remoteness of the deposits from the markets on the Atlantic seaboard, where brimstone is consumed chiefly by the manufacturers of sulphuric acid, of sulphite for paper making, and by the manufacturers of explosives.

CONSUMPTION OF SULPHUR IN THE UNITED STATES.
(In tons of 2240 lb.)

Source.	1902	1903	1904	1905	1906	1907	1908
Sulphur—Domestic production	7,443	35,098	193,492	215,000	294,000	307,806	312,670
Imports	176,951	190,931	130,421	84,579	64,646	22,523	21,135
Total	184,394	226,029	323,913	299,579	358,646	325,930	333,805
Exports	1,253	967	2,493	1,713	14,419	35,925	27,894
Consumption	183,141	225,062	321,420	297,866	344,227	290,005	305,911
(a) Sulphur contents	179,478	220,560	314,992	291,909	337,342	284,205	299,793
Pyrites—Domestic production	228,198	199,337	173,221	224,980	225,045	261,871	206,471
Imports	440,363	427,319	413,585	515,722	533,346	627,985	668,116
Total	668,561	626,706	586,806	740,702	758,391	889,856	874,587
Exports	3,060	1,330	.....	.....	.....	.....	.....
Consumption	665,501	625,376	586,806	740,702	758,391	889,856	874,587
(b) Sulphur in domestic	104,071	87,730	76,217	98,991	99,020	115,223	90,847
(c) Sulphur in foreign	205,532	200,215	194,385	242,399	250,673	295,153	314,015
Total sulphur content	309,603	287,945	270,602	341,390	349,693	410,376	404,862
Grand total sulphur consumption	489,081	508,505	585,594	633,289	687,035	(d)694,581	(d)704,655

(a) Includes crude and refined sulphur. Sulphur content of crude is computed at 98 per cent. (b) Computed at 44 per cent. (c) Computed at 47 per cent. (d) This figure is in excess of the true consumption as a large percentage of the domestic output of sulphur was stored.

IMPORTS OF SULPHUR INTO THE UNITED STATES.
(In tons of 2240 lb.)

Kind.	1905		1906		1907		1908	
	Amount.	Value.	Amount.	Value.	Amount.	Value.	Amount.	Value.
Crude	83,201	$1,522,005	72,603	$1,282,873	20,399	$355,941	19,620	$318,577
Flowers	572	16,037	1,099	29,565	1,458	41,216	793	22,562
Refined	79	19,960	709	17,928	606	14,589	692	17,227
Precipitated	27	3,352	28	3,224	60	8,426	30	4,012
Total	84,579	$1,561,354	74,439	$1,333,590	22,523	$420,175	21,135	$362,378

WORLD'S PRODUCTION OF SULPHUR. (a)
(In metric tons.)

Year.	Austria. (d)	Chile.	France. (c)	Hungary	Germany	Greece.	Italy. (b)	Japan.	Spain.	United States.	Total.
1895..	830	.....	4,213	102	2,061	1,480	370,766	15,557	2,231	1,676	398,916
1896..	643	940	9,720	138	2,263	1,540	426,353	12,540	1,800	3,861	459,798
1897..	530	664	10,723	112	2,317	358	496,658	12,013	(b)3,500	1,717	528,592
1898..	496	1,256	9,818	93	1,954	135	502,351	10,339	3,100	2,770	532,312
1899..	555	939	11,744	116	1,663	1,150	563,697	10,241	1,100	1,590	592,290
1900..	862	2,472	11,551	123	1,445	891	544,119	14,439	750	4,630	581,252
1901..	4,911	2,516	6,836	137	963	2,336	563,096	16,548	610	6,977	604,930
1902..	3,721	2,636	8,021	105	487	1,391	510,333	18,287	450	7,565	552,996
1903..	4,475	3,550	7,375	135	219	1,266	553,751	22,914	1,680	35,660	631,035
1904..	6,288	3,594	5,447	143	209	1,225	527,563	25,587	605	196,588	767,249
1905..	8,407	3,470	4,637	135	205	1,126	568,927	24,652	610	218,440	830,609
1906..	15,125	4,598	2,713	133	178	(f)1,000	499,814	27,589	700	298,704	845,956
1907..	(c)24,099	2,905	2,000	(f)135	176	(f)1,000	426,972	28,381	3,612	312,731	801,911
1908..	(e)	(e)	(e)	(f)130	.....	(e)	(e)	(e)	(e)	312,670	(f)800,000

(a) From the official reports of the respective governments. The sulphur recovered as a by-product by the Chance-Claus process in the United Kingdom, amounting to between 20,000 and 30,000 long tons annually, is not included. (b) Crude. (c) Crude mineral; limestone impregnated with sulphur. (d) Crude rock. (e) Not yet reported. (f) Estimated.

Market Conditions and Prices.—Although the trading in sulphur in 1908 amounted to not over 40 per cent. of that in 1907, prices held fairly constant throughout the year. The quotations on Louisiana, prime, opened at $22 per long ton and held at that figure until November, after which the price was a little higher. On account of the slack trade a large surplus of stock was accumulated.

Sulphur Mining in the United States.

Alaska.—Large deposits of sulphur have been rumored to exist near the volcano of Mt. Makushin on the northern part of the island of Unalaska, about 1730 miles northwest from Seattle, Wash. These deposits have recently been thoroughly investigated and proved to be of but small commercial importance. The sulphur occurs within the old outer rim of the crater at the summit of the mountain and the area covered by the deposit is claimed to be somewhere in the neighborhood of three or four acres, but the depth is only from 4 to 14 inches.

Louisiana.—This State produces more sulphur than all of the rest of the United States, and with the mines of Sicily furnishes the greater part of the world's sulphur supply. The deposits were discovered about 1868 and have been developed to a large extent. The sulphur is mined by the Frasch method, which has been described fully in previous volumes of The Mineral Industry. The sulphur as gathered in the tanks is about 98 per cent. pure. Most of the development of this field has been done by the Union Sulphur Company since 1895. At the present time the Union Sulphur Company keeps about seven of its wells in constant operation.

Nevada.—A few loads of sulphur were taken from a large deposit a short distance south of Goldfield, Esmeralda county. The deposit has not been extensively exploited.

Texas.—The Matagorda Oil and Mineral Company, Bay City, is reported to have leased the sulphur deposits at Big Hill, Limestone county, to a St. Louis firm for development.

Wyoming.—Sulphur deposits have long been known to exist in this State. In 1908 the output of crude sulphur amounted to about 300 tons, valued at $10,500, and as the mines were only in operation during the last few months of the year a much larger production is expected for 1909. The deposits which have recently come into prominence are situated near Thermopolis and have been investigated and made the subject of a bulletin by the U. S. Geological Survey. The deposits are about 3½ miles northwest of Thermopolis, in Fremont county. The section that has been prospected by bore holes and proved is on a gentle northeast slope of a small eroded anticline lying adjacent to the valley of Owl creek.

The deposits are undoubtedly the result of hot spring deposition and appear in commercial size at the contact of an underlying limestone with travertine. The travertine contains isolated crystals of sulphur but the minable deposits, so far, have all been found in the limestone, where in old hot spring sites the sulphur occurs as a replacement product. The deposits, from their nature, rarely exceed a horizontal extent of 100 ft. and it is not expected that the vertical persistence will be great.

As far as I have been able to determine, the only company producing sulphur in this region is the Wyoming Sulphur Company. This company is working the deposits in open quarries, the ground having first been proved by drilling. The rock is crushed or broken to convenient size, hand sorted and hauled a short distance by wagon to the reduction works. The smelting operation is simple and not highly efficient as only about two-thirds of the sulphur is extracted. It consists simply in heating the broken ore with steam. The cars of ore, three at a time, are run into a cylindrical retort into which steam is admitted at 60 lb. pressure for two hours, the sulphur being melted and collected in bins connected by a trap with the bottom of the retort.

Sulphur in Foreign Countries.

Chile.—Chilean sulphur is not as yet exported. A refinery is situated at Tacna where the two classes of sulphur obtained are: Azufre sublimado, or flour sulphur, 99.82 per cent. pure; and refined sulphur in lumps of 99½ per cent. The former brings $6.30 Chilean (at 18d. exchange, equivalent to $2.30 American gold) per quintal of 101.61 lb. and the refined $4.30 ($1.57 American gold) per quintal. These prices are for cash f.o.b. Tacna. It is estimated that this district can supply about 1000 quintals per month.

About six miles west of Santa Rosa de Pastos Grandes, at an elevation of more than 5000 m., sulphur deposits are found on the volcanic mountain known as Cerro Azufre. The principal deposit occurs on the side of the mountain facing Pastos Grandes, a little below the crest between the two peaks of the mountain. The sulphur occurs in numerous cavities, sometimes many decimeters wide, which permeate the formation, the purer sulphur occurring in the center. The sulphur deposits are mostly associated with crystalline aggregates of gypsum. There have also been deposits of sulphur-bearing rock noted on the southern slope of the northern peak, where these rocks occur intercalated with altered eruptives.

An important deposit of sulphur is said to exist on the summit of the extinct volcano Cerro Lacco. White sulphur here occurs mixed with the ordinary yellow variety. The volcanic mountains Estrella and Sin Nombre also contain sulphur, but from their situation on the extreme edge of the Atacama desert these deposits at present possess no economic importance.

Ecuador.—At Santa Elena, Jipijapa and other places in Ecuador sulphur is obtained from hot springs and used for bleaching straw for the manufacture of Toquilla straw hats. Rich deposits are also said to occur on one of the islands of the Galapagos group, and near Alausi on the Guayaquil & Quito railway, but these latter have never been exploited to any extent. The former source is said to supply the greater part of Panama's trade in sulphur.

Japan.—In 1907 the exports of sulphur amounted to only 20,696 tons, as compared with 28,030 tons in 1906. This is accounted for in great measure by the heavy stock held over from the previous year, and also to competition by the chief producer in the United States. The principal mines produced about the same amount of sulphur as in the previous year. The chief districts in which sulphur occurs in Japan are those of Volcano bay, central Hokkaido, and the Kuriles. In the Volcano Bay district, most of the sulphur comes from near Hakodate, but a considerable quantity is also obtained from near Kuchan in Shiribeshi. The chief mines of the central district are the Ishikaridake, Mount Akan, and Shari. Sulphur is found upon nearly all of the Kurile islands. Of Japan's total sulphur output 90 per cent. comes from the Volcano Bay district. Throughout the volcanic mountainous region of Nippon sulphur is won in a crude way from the hot springs. The sulphur is simply precipitated on straw or dried grass in troughs on the mountainside, into which the sulphurous spring water is run.

Mexico.—Modern methods for mining sulphur have been introduced into Mexico, and recent developments indicate that this country will

compete on a moderate scale with the American company. The mines at Conejos have been worked for a number of years and now a new refinery has been built, and the Frasch method of mining introduced. The sulphur as obtained is 98 per cent. pure. Until recently the product of these mines has been shipped in the crude state to the United States via Tampico. The Porvenir, one of the principal mines of this district, was recently sold to the Compania de Dinamita, and the product therefrom is used in the manufacture of explosives. A new discovery has recently been made in this district, but so far but little development work has been carried on. Conejos is situated on the Mexican Central railway in the State of Durango, about three hours' ride northwest of Torreon. The Los Cerritos sulphur mines near San Luis Potosi are said to be of importance and to resemble those of Conejos in the character of the formation. Recent reports indicate the deposit of Mt. Popocatapetl to be much overrated.

Persia.—During 1908 the mines at Bostana were again started. However, in the coast ports of Persia it was cheaper to buy foreign sulphur than the native mineral, so the product of the home mines had to be marketed in the interior of the country. East of Lingah at Kameer is a sulphur deposit similar to that at Bostana. The sulphur occurs finely disseminated throughout a white dusty decomposed rhyolite, associated with gypsum. The ore is not rich and hardly pays for the mining when carried on by the crude Persian method.

Sicily.—American competition has wrought havoc with the Sicilian sulphur industry, and the setback which was experienced during 1906-07 continued to be felt in 1908. The export trade remained about stationary during 1908, and meanwhile the stock increased until by the end of the year the accumulation in Sicily was 21,000 tons in excess of that at the end of 1907, amounting in all to about 600,000 tons. Figures supplied by Parsons & Petit, of New York, American agents for the Sicilian sulphur industry, indicate a counter move on the part of the Consorzio, for they show that during the first 10 months of 1908 the United States imported 11,182 tons of sulphur from Sicily, an increase of 7139 tons over the import during the corresponding period in 1907. Owing to the unsettled conditions following the earthquake of Dec. 28, it is, at the time of this writing, impossible to obtain complete statistics of the industry for 1908.

According to Emil Fog & Son, of Catania, Silicy, 80 of the smaller mines—those worked by the most primitive methods, the ore being carried on the shoulders of men—were closed during 1908, thus reducing the total number working to 500. Of big mines producing over 10,000 tons there were only six operating at the close of 1908. These were

responsible for about one-third of the total production for the year. The remainder were by no means in a flourishing condition, and a strict enforcement of the law prohibiting the employment of boys for the carrying of the ore would compel many of them to close down.

So far, the means suggested to improve conditions have proved ineffectual; in many cases, impracticable. It has been suggested to check the production of the mines, but it was feared that this would cause starvation among the miners; against placing the sulphur held in stock upon the European market there is the objection that this would likely bring forth such disastrous competition that the resulting low prices could with difficulty ever be raised again to a profitable figure. However, because of the lack of demand prices on both crude and refined sulphur were cut slightly by the Consorzio in October.

The Italian government recently dissolved the administration of the Consorzio and nominated a royal commissioner to administer its affairs until a new administration should be elected by its members. Much dissatisfaction was manifest between the different interests involved as

TOTAL EXPORTS OF SULPHUR FROM SICILY, 1900-1907. (a)
(In tons of 1030 kg.)

Country.	1900	1901	1902	1903	1904	1905	1906	1907
Austria	21,594	18,842	19,086	17,926	23,374	25,111	22,756	24,597
Belgium	9,721	7,471	12,223	15,233	13,627	14,442	13,940	8,853
France	103,647	74,394	67,249	74,372	103,040	96,170	67,536	59,725
Germany	28,702	23,448	25,906	32,553	31,613	28,319	34,967	37,100
Greece and Turkey	19,647	21,702	20,548	22,133	25,376	25,069	26,560	27,606
Holland	18,595	10,848	8,648	5,157	8,122	4,425	5,539	11,379
Italy	101,073	74,516	45,603	45,572	79,619	99,633	79,519	58,926
Portugal	10,937	11,385	10,614	14,064	8,373	13,196	12,302	12,778
Spain	6,187	2,979	2,249	4,099	4,064	2,478	3,120	
Scandinavia (c)	22,681	24,486	24,918	28,292	20,120	18,288	21,608	25,155
Russia	22,090	15,110	17,295	15,068	15,141	16,573	16,181	15,210
United Kingdom	23,973	22,468	25,477	19,210	18,108	18,847	20,883	16,561
United States	162,505	144,817	168,919	155,996	100,000	70,332	41,283	9,476
Other countries (b)	6,810	9,484	18,484	25,833	25,167	23,277	21,288	26,646
Totals	558,162	462,299	467,319	475,508	475,745	456,260	387,432	334,014
Stock in Sicily, Dec. 31.	221,204	302,410	330,113	361,220	396,541	462,457	525,115	576,377

(a) In 1900 and 1901 by A. S. Malcolmson, New York; for following years, by Emil Fog & Sons, Messina. (b) Mainly South Africa, Northern Africa, Australia and the East Indies. (c) Including Norway, Sweden and Denmark.

to the treatment each should receive. At Catania approximately one-third of Sicily's sulphur is refined, yet the selling price fixed for that city has been uniformly higher than the rate fixed by the Consorzio for either Girgente or Licata.

Up to the end of 1908 the Consorzio has been unsuccessful in its endeavor to cope with American competition, so the Italian government has appointed a commission to look into the possibilities of modifying the law which created the obligatory coalition of interests and to investigate the continued overproduction of sulphur in Sicily. The question of American competition became critical when the United States entered

SHIPMENTS OF SULPHUR FROM SICILY TO THE UNITED STATES.
(In tons of 1016 kg.)

Port.	1902		1903		1904		1905		1906		1907	
	Seconds.	Thirds.	Seconds.	Thirds.	Seconds.	Thirds.	Seconds.	Thirds.	Seconds.	Thirds.	Seconds.	Refined. (c)
New York...	76,283	26,842	70,800	21,201	41,429	10,547	26,782	16,270	18,106	13,009	3,098	254
Philadelphia..	3,500	10,399	4,910	8,500	1,325	5,325	800	1,848	554	200	.....	275
Baltimore....	9,065	2,400	10,900	2,000	3,370	1,400	.....	....	21	.....	.....	.....
Boston.....	2,204	2,300	5,508	2,450	11,397	1,749	5,477	1,009	950	1,750	.....	169
Portland, Me.	26,328	.....	23,855	.....	23,638	.....	17,882	.....	4,993	.....	.....	.....
Other ports(a)	8,498	1,000	5,872	.....	.....	.....	.....	.....	1,419	272	(b) 5,500	180
Totals.....	125,978	42,941	121,845	34,151	81,159	19,521	50,941	19,127	26,052	15,231	8,598	878

(a) Norfolk, Mobile, New Orleans, Savannah, San Francisco, Bangor, Portland, Ore., St. Louis and Canada. (b) All to San Francisco. (c) No thirds shipped.

into the market for the refined sulphur as well as that for the raw material. This came about when an American company erected a sulphur refinery capable of consuming 100,000 tons of mineral per annum at Marseilles, thus getting around the import duty imposed on refined sulphur by France. The action of Scandinavian firms in altering their wood pulp paper plants, so as to replace the use of sulphur by that of pyrites, also means a further limitation of the sulphur market and hence more hardships for the Sicilian industry.

Investigations conducted at the instigation of the Italian government by Dr. Giuseppe Oddo, of the University of Pavia, have shown the superior adaptability of Sicilian crude sulphur ore, as compared with pyrite, for the manufacture of sulphuric acid, and this may lead to a larger demand for the material and hence a betterment of conditions in Sicily. Experiments were first conducted with hand-worked Maletra furnaces at Regio Polverificio, and later with a mechanical furnace of the Herreshoff type at the Fabbrica di proditti azotati de Piano d'Orte, near Torre dei Passeri. The ore used contained 31.75 per cent. of free sulphur and there remained, when proper care was exercised, not more than 0.03 per cent. of free sulphur in the residue after calcination. It was found that a delicate regulation of temperature was necessary in this work, as too low a roasting heat resulted in the distillation and loss of the sulphur; excessive heat, on the other hand, oxidized the sulphur in the ore to a sulphide or sulphate in the cinder. To avoid volatilization the ore was introduced only on the third hearth in the Maletra furnace and on the third or even on the fourth hearth of the Herreshoff mechanical furnace.

PYRITES.

The pyrite business was probably affected less by the general business depression of 1908 than was any other branch of the mineral industry. Two factors contributed to this: In the first place, the stable nature of

the business, closely allied as it is with the agricultural interests of the country (the chief use for pyrite being in the manufacture of sulphuric acid for use in the fertilizer industry); and secondly, the fact that contracts for pyrite delivery usually cover a period of years so that the 1908 trade was upheld by contracts made during 1906 and early 1907. However, there was a decrease in the domestic production of pyrite, a number of the producing companies being unable to operate. On the other hand, the imports of pyrites in 1908 increased, although, as is shown in the accompanying table, this increase was not as great as that of the last few years. The fact that ocean rates were lowered during the time of financial depression explains this increase in the consumption of foreign pyrite in 1908.

PRODUCTION, IMPORTS AND CONSUMPTION OF PYRITES IN THE UNITED STATES. (a)
(In tons of 2240 lb.)

Year.	Production.		Imports. (b)		Consumption.	
1897	133,368	$404,699	259,546	$847,419	392,914	$1,252,118
1898	191,160	589,329	171,879	544,165	363,039	1,133,494
1899	178,408	583,323	310,008	1,074,855	488,416	1,658,178
1900	201,317	684,478	322,484	1,055,121	523,801	1,739,599
1901	234,825	1,024,449	403,706	1,415,149	638,531	2,439,598
1902	228,198	971,796	440,363	1,650,552	668,561	1,622,648
1903	199,387	787,579	425,989	1,628,400	625,376	2,416,179
1904	173,221	669,124	413,585	1,533,464	586,806	2,202,688
1905	224,980	752,936	515,722	1,780,800	740,702	2,533,736
1906	225,045	767,866	597,347	2,138,746	822,392	2,906,612
1907	261,871	851,346	656,477	2,637,485	918,348	3,488,831
1908	206,471	744,463	668,116	2,624,339	874,587	3,368,802

(a) These statistics do not include the auriferous pyrite used for the manufacture of sulphuric acid in Colorado. (b) Net imports, less re-exports of 3060 tons in 1902 and 1330 tons in 1903.

The entrance of Ducktown, Tenn., into the field as a producer of sulphuric acid in enormous quantities at first had a depressing effect upon the pyrite trade, particularly the import trade. It was thought that the Tennessee Copper Company, with an estimated output of 120,000 tons of acid per anuum, would flood the market and cause a serious curtailment in the demand for foreign pyrite. There also loomed upon the horizon the Ducktown Sulphur, Copper and Iron Company, with an annual output of 50,000 tons of sulphuric acid. However, the fertilizer business is one that is continually growing, and although these companies may gain control of the acid market in their vicinity, or even in the greater part of the South for a while, it is not anticipated that this will seriously affect the pyrite trade as a whole for any great length of time.

The product of the Ducktown companies will naturally tend to displace Spanish pyrite at Gulf and Atlantic points, and will also gain control of that part of the southern acid trade which has heretofore been supplied by the zinc smelters of the Middle West. The fact that it

is cheaper to transport pyrite than acid will not, however, permit the shipment of acid north of the Ohio river; this will tend to limit the zone controlled by these companies. The effect of the competition, which is bound to come, for the control of the acid market, should materially reduce the price for acid, especially in the South and Middle West and afford an immense benefit to all industries which employ chemicals. The erection of fertilizer plants near the sources of acid will certainly operate to the advantage of those sections.

The operations of the Tennessee Copper Company are fully described in the article on copper in this volume of THE MINERAL INDUSTRY. In Vol. XV is given a complete treatise on the commercial uses of sulphuric acid.

Prices.—After a slight drop in the early part of 1908 prices held fairly constant. The following are the average prices for the year quoted in cents per unit of sulphur in the ore: Domestic, non-arsenical, furnace, 11.25c.; domestic, non-arsenical, fine, 10.25c.; imported, non-arsenical, furnace, 12.70c.; imported, arsenical, furnace, 12.20c.; imported, arsenical, fine, 8.75c.; imported, non-arsenical, fine, 10.75c. These prices are slightly lower than those prevailing in 1907.

Pyrites in the United States.

A general improvement of methods was noted in 1908 at the mines in the United States. This is most striking in the substitution of mechanical breaking devices for the old hand methods. In some instances gyratory crushers were installed. The general unrest in the acid market closed down but few mines and pyrite miners are optimistic as to the outcome in 1909.

Alabama. (By Eugene A. Smith.)—As heretofore most of the pyrite produced in Alabama in 1908 came from the "Copper lead" in Clay county, and within three or four miles of Pyriton station. At these points the deposit shows a thickness of 7 to 9 ft., and in some parts carries a small percentage of copper. The actual production was confined to three companies. This industry in 1908, like so many others, suffered from the financial depression.

California.—The production of pyrite in California during 1908 was small.

Georgia.—Pyrite mines have recently been developed and are already producing.

Massachusetts.—The production of pyrite in this State increased in 1908. Massachusetts pyrite is very pure, containing almost no arsenic, and for this reason the demand for it is good. The Davis Sulphur Ore Company continues to be the chief producer.

New York.—Several of the large pyrite mines were forced to close down during 1908. The total production amounted to only 23,775 tons ($104,798) or not quite one-half of that for 1907. Nearly all of this was mined in St. Lawrence county by the St. Lawrence Pyrite Company. The American Pyrite Company, operating in the same field, was forced to close down and did not produce any ore during 1908.

New Hampshire.—There was an increase in production of pyrite in 1908. The total production was 13,830 long tons, against a production of 11,000 long tons in 1907. The Milan Mining and Milling Company was the largest producer.

Ohio.—The total production of Ohio amounted to 5700 tons. The East Goshen Coal Company was a large producer.

Virginia.—This State continues to lead as a producer of pyrite. Louisa county mines alone produced 76,229 long tons of pyrites in 1908, valued at $271,811. The total output of the State amounted to 117,440 long tons, valued at $389,808.

A great deal of interest attaches to the operations of the Pulaski Mining Company, whose mines are on the Cripple Creek extension of the Norfolk & Western railroad and whose roasting and sulphuric-acid plant is at Pulaski. At the latter pyrrhotite is being practically dead-roasted, sulphuric acid is being manufactured, and the resulting cinder employed as part of the charge in an iron blast furnace. The ore is mined from open cuts, and is delivered to the plant at Pulaski at a cost of about 80c. per ton. The ore is first crushed and then conveyed to Herreshoff roasters. The cinder from these burners is taken to a 100-ft. rotary cement kiln and clinkered. Powdered coal is blown into this kiln at the end opposite the feed. The cinder when fed to the kiln contains from 4 to 7 per cent. sulphur; after going through the clinkering process the sulphur content is reduced to 0.05 per cent.

The raw ore as it comes to the plant has about the following analysis: Sulphur, 27 to 32 per cent.; iron, 45 to 52; calcium and magnesium oxides, 3 per cent.; phosphorus, trace; manganese, trace; moisture, 3 per cent. The product, as it comes from the kiln, will analyze about as follows: Iron, 52 to 56 per cent.; sulphur, 0.05 per cent.; phosphorus and manganese, trace. About 75 tons of sulphuric acid and 100 tons of iron cinder are being produced daily. The cinder is sold to the Pulaski Iron Company, the furnace of which is only a short distance away, where it comprises $\frac{1}{16}$ to $\frac{3}{8}$ of the ore mixture. The price paid for the cinder is in the neighborhood of 7c. per unit of metallic iron.

A deposit of pyrite in Spottsylvania county attracted considerable attention in 1908. The deposit is about 10 miles northeast of the Louisa county mines and the iron cap or gossan is said to resemble that at the latter.

Pyrites in Foreign Countries.

The accompanying table shows the world's pyrite production for a series of years. Most of the pyrite is used directly in the manufacture of sulphuric acid for the production of superphosphates in the fertilizer trade.

WORLD'S PRODUCTION OF PYRITES.
(In metric tons.)

Year.	Belgium.	Bosnia	Canada.	England.	France.	Germany.	Hungary.	Italy. (a)
1896	2,560	.....	30,580	10,177	282,064	129,168	52,697	45,728
1897	1,828	.....	35,291	10,752	303,488	133,302	44,454	58,320
1898	147	3,670	29,223	12,302	310,972	136,849	58,079	67,191
1899	283	.....	25,112	12,426	318,832	144,623	79,519	76,538
1900	400	1,700	36,308	12,484	305,073	169,447	87,000	71,616
1901	560	4,570	31,982	10,405	307,447	157,433	93,907	89,376
1902	710	5,170	32,304	9,315	318,235	165,225	106,490	93,177
1903	720	6,589	30,822	9,794	322,118	170,867	96,619	101,455
1904	1,075	10,421	29,980	10,452	271,544	174,782	97,148	112,004
1905	976	19,045	29,713	12,381	267,114	185,368	106,848	117,667
1906	908	13,474	35,927	11,318	265,261	196,971	112,023	122,364
1907	397	3,671	46,982	10,357	283,000	196,320	99,503	127,000
1908	(b)	(b)	48,093	(b)	(b)	219,455	(b)	(b)

Year.	Japan.	Newfoundland	Norway. (c)	Portugal. (d)	Russia.	Spain.	Sweden.	United States.	Total.
1896	(b)	27,267	60,507	207,440	11,550	100,000	1,009	111,031	1,071,778
1897	7,626	32,790	94,484	276,738	19,380	100,000	517	133,502	1,252,472
1898	8,726	32,335	89,763	302,686	24,570	70,265	386	194,219	1,341,383
1899	8,376	26,154	95,636	347,234	23,250	107,386	150	181,263	1,446,782
1900	16,166	*Nil*	98,945	402,870	23,154	34,638	179	204,538	1,464,512
1901	17,589	7,532	101,894	443,397	30,732	33,953	*Nil*	238,582	1,568,999
1902	18,580	26,000	121,247	413,714	26,465	145,173	*Nil*	231,849	1,713,654
1903	16,149	42,674	129,939	376,177	22,780	155,739	7,793	202,577	1,692,812
1904	24,886	61,166	133,603	383,581	31,667	161,841	15,957	175,992	1,696,099
1905	25,569	51,534	162,012	352,479	30,689	179,079	20,762	228,580	1,789,816
1906	36,038	28,583	197,886	350,746	(e)30,000	189,243	21,827	228,646	1,841,815
1907	36,124	28,000	225,000	351,000	30,500	225,830	27,000	266,061	1,690,084
1908	(b)	(b)	(b)	(b)	(b)	(b)	(b)	209,774	(e)1,650,000

(a) Cupriferous in part. (b) Reports not yet available. (c) Both iron and copper pyrites. (d) Copper pyrite. (e) Estimated.

Algeria.—Investigation has proved the presence of pyrites in commercial quantities in Algeria. The ore is found mainly in ancient schists or in calcareous formations of Triassic age in conjunction with acid or basic eruptive dikes, and also in the eruptive rock itself. Algerian pyrite is generally quite pure, with the sulphur content averaging as high as 50 per cent. Twelve valuable deposits are known to exist in the department of Constantine, but only two concessions have been let, one at Ain-Sedma and the other at El-Azonar. The former was originally an iron concession but has developed into a pyrite mine pure and simple, with depth the orebody having been found to contain practically nothing but pyrite. The pyrites of Ain-Sedma occur in a trachyte.

The only mines in actual operation in Algeria are those of El-Azonar. These workings are situated about 40 miles southeast of Bougie, and the concession, dating from 1903, embraces 243 hectares. The deposit occurs in a calcareous formation of Triassic age and is capped with a thin gossan of poor iron ore. The orebody seems to be associated with ophitic diabase which outcrops in the vicinity of the pyrites orebody.

Spain.—Nearly all of the pyrites imported into the United States comes from Spain. The production of pyrites in Spain in 1907 was 225,830 metric tons, valued at $211,060. Spanish pyrite is of a good quality and commands a high price in American markets. The seeming discrepancy between these figures and those of pyrite exported from Huelva in 1907, and also those of the Rio Tinto company, is accounted for by the fact that the Spanish government considers as pyrite only that ore mined solely for its iron-sulphur content. The Rio Tinto ore contains copper which is removed by leaching, after which the residue is marketed as sulphur ore.

The Huelva pyrite exports in 1907, not including Spanish and Portuguese shipments via Guadiana river, as compiled by M. Yglesias & Son, were: Germany and Holland, 774,827 tons; Great Britain, 533,706; United States, 522,828; France, 263,928; Italy, 110,652; Belgium, 81,210; other countries, 107,419; a total of 2,394,570 tons.

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TALC AND SOAPSTONE.

By Richard H. Vail.

Talc though a mineral of almost universal occurrence is produced commercially in but a few countries, Italy, France and the United States supplying the bulk of the production. In the United States, on account of transportation and market conditions, the output has been limited to the States on the eastern slope of the Appalachian range. It is marketed under three general heads: (1) The ground fibrous talc used in paper making, this being obtained almost exclusively from the deposits in St. Lawrence county, New York, which State is the most important producer of talc in this country; (2) the massive variety known as steatite, or soapstone, the supply of which comes mainly from Virginia; (3) ordinary ground talc of various grades, produced in many of the Atlantic States, Vermont, Massachusetts and North Carolina now contributing most of the domestic talc of this class.

STATISTICS OF TALC AND SOAPSTONE IN THE UNITED STATES. (a)
(In tons of 2000 lb.)

Year.	Production.						Imports.		
	Fibrous Talc.			Talc and Soapstone. (b)			Sh. Tons.	Value.	Value Per Ton.
	Sh. Tons.	Value.	Per Ton.	Sh. Tons.	Value.	Per Ton.			
1896.....	51,816	$256,080	$4.94	21,448	$207,085	$9.66	1,950	$18,693	$ 9.60
1897.....	52,836	283,685	5.37	27,068	259,948	9.60	779	8,423	10.54
1898.....	54,807	285,759	5.21	27,974	237,280	8.48	445	5,526	10.70
1899.....	57,120	272,595	4.77	26,682	241,267	9.04	254	3,534	13.91
1900.....	45,000	236,250	5.25	26,726	249,777	9.35	79	1,070	13.50
1901.....	69,200	483,600	6.99	28,643	424,888	14.83	2,386	27,015	11.32
1902.....	71,100	615,350	8.65	26,854	525,157	19.36	2,859	35,336	12.36
1903.....	60,230	421,600	7.00	26,671	418,460	15.69	1,791	19,677	10.99
1904.....	65,000	455,000	7.00	27,184	433,331	15.94	3,268	36,370	11.13
1905.....	67,000	519,250	7.75	40,134	637,062	15.87	4,000	48,225	12.06
1906.....	64,200	541,600	8.43	58,972	874,356	14.82	5,643	67,818	12.02
1907.....	59,000	501,500	8.50	72,010	905,047	12.57	10,060	126,391	12.56
1908.....	70,739	697,390	9.86	.....	.....	.....	7,429	97,296	13.09

(a) Statistics for 1902 and subsequent years, are as reported by the United States Geological Survey, except that fibrous talc is as reported by the New York State Geological Survey. (b) The value of these products has not much significance owing to the diverse conditions of the material reported.

Talc production in the United States during 1908 declined in the varieties marketed as ground talc and soapstone, but there was a marked increase, both in the value and amount of fibrous talc produced in New York for the paper making industry. This was partly due to the short

output of 1907, the production in that year being much below normal. Conditions surrounding the production of fibrous talc in New York are exceptional and do not obtain in other States. The superior quality of this fibrous talc for book and writing paper gives the Gouverneur district a practical monopoly of the field. In the other States, most of which produce both soapstone and the ordinary foliated talc, the deposits as a rule are not extensive and their economic importance depends primarily upon their proximity to transportation and market. Deposits of talc are not rare, and if of only ordinary grade, the mineral is seldom profitably shipped as mined. Most of the talc mines are owned or controlled by manufacturers or users, and in consequence the mineral usually finds its way into the markets in the manufactured state under one of the three general heads above mentioned. Varieties of unusual purity and quality are sometimes shipped as mined, but this is the exception rather than the rule, and the prices usually quoted are for the manufactured or finished product. The ground talc is of many grades, as indicated by the wide range of price—$8 to $40—the various grades entering a much diversified market according as they meet the qualifications required by the different trades. There is a good demand for the best grade, as is indicated by the importation of high-grade talc from Italy and France, notwithstanding an import duty of 20 per cent. ad valorem on the ground talc. The crude is admitted free of duty and a considerable quantity is thus imported and prepared in the mills of this country by jobbers of these products. Some fibrous talc is exported from the United States, the amount shipped during 1908 being estimated at 5000 to 10,000 tons, most of it being consigned to Germany for use in paper manufacture.

Prices and Market Conditions.—Owing to stock carried over from the previous year and the generally depressed condition of many industries, the talc market was over-supplied during the early part of 1908 and lower prices prevailed for all grades of the ordinary foliated talc and soapstone. During the latter half of 1908 the market was steadier and better prices prevailed. The prices of ground talc for 1908 according to quality and quantity were from $8 to $23 per ton for American talc; $15 to $25 for French talc; and from $25 to $40 for Italian talc. The effect of the general business depression was most strongly felt by the producers of soapstone and articles of soapstone manufacture, prices for these articles being greatly reduced.

While lower prices prevailed generally for the ordinary powdered talc, the fibrous talc market in New York was fairly steady at the outset by reason of its strong statistical position, the 1907 output having been curtailed owing to one of the largest mills being out of commission.

The price per ton at Gouverneur ranged from $8 to $10, the latter price being maintained most of the year. The average for the year's production was $9.86 per ton as compared with $8.50 in 1907.

Talc and Soapstone in the United States.

Arkansas.—The deposits of the Arkansas Soapstone and Refractories Manufacturing Company, near Benton, Saline county, were not operated during 1908. The project to build a branch railway line to Pinnacle was abandoned, and fresh surveys were made, providing for a connection with the St. Louis, Iron Mountain & Southern railway at Bryant. The soapstone of these deposits is suitable for the manufacture of refractory products, and the company expects to start production in 1909.

Georgia.—Except that obtained from development work, but little ground talc or soapstone was produced during 1908. The most important deposits in Georgia are in Murray county where are the properties of the Cohutta Talc Company, and the Georgia Talc Company of Chatsworth. Both of these companies expect to operate upon a larger scale during 1909.

Maryland.—No production was reported from Maryland in 1908. The Steatite Corporation owns a soapstone deposit in Carroll county, near Marriottsville. The property is equipped with a mill having a capacity of 40 to 50 tons of powdered talc per day and machinery for producing 5000 sq.ft. of sawed slabs. The mill, however, was closed during 1908. Another Maryland deposit is that of the Cecil Mineral Company near Conowingo in Cecil county.

Massachusetts.—The only producer of importance in this State during 1908 was the Massachusetts Talc Company of North Adams, Mass. Its mines are situated in Franklin county, about four miles from Zoar station, on the Fitchburg division of the Boston & Maine railroad. The rocks in this vicinity are schists and gneisses of Silurian age. The deposit is from 18 to 28 ft. wide and is operated from a two compartment shaft 14x7 ft. in the clear, sunk to a depth of about 200 ft. The property is equipped with a modern mill and the entire output during 1908 was pulverized.

New York.—The production of fibrous talc in this State increased during 1908 both in value and amount, the output for the year being 70,739 tons, valued at $697,390. There was also an increase in the average price received, the selling prices at Gouverneur ranging around $10 per ton for most of the year, as compared with an average of $8.50 per ton in 1907. St. Lawrence county furnishes almost the entire quota of fibrous talc used in the paper industry, and most of this product

is controlled by the International Pulp Company, of New York. Its milling capacity during 1907 was curtailed through the burning of its Hailesboro mill in 1906. This mill was rebuilt and placed in operation during the early part of 1908 with a nominal capacity of 100 tons of ground talc per day. The company's mill at Dolgeville was increased at the time of the Hailesboro fire, so that the International Pulp Company's milling capacity is now the largest in its history.

The Ontario Talc Company, an independent producer with a mill at Fullerville, has continued the development of the Potter mine with favorable results, finding an excellent grade of fibrous talc. The company has had under consideration the construction of a new mill at Gouverneur, but has not as yet begun its erection.

The Uniform Fiber Talc Company was organized in September, 1908, and secured a talc property on Wintergreen hill, being a part of the N. H. Freeman farm, west of Talcville. The company contemplates the construction of a mill near the mine, and for this purpose has secured a water power on the Oswegatchie river, about one half mile above Dolgeville. The power will be transmitted to the mill by electricity and the company plans to produce from 30 to 50 tons of ground talc per day.

Some exports of fibrous talc were made in 1908, particularly to Germany. Besides the fibrous talc, which is the main product of the St. Lawrence county mines, the ordinary foliated variety is also found. This is prepared separately, requiring long-continued grinding to reduce the flakes to the proper fineness, and finds special use in the paper trade, where it is employed in the place of ground mica for giving a lustrous surface to certain papers.

North Carolina. (By Joseph Hyde Pratt.)—There are two distinct minerals mined in North Carolina that are being placed on the market as talc. One of these, pyrophillite, is chemically very distinct from talc but in its physical properties is similar, and for certain uses, it answers as well as talc. This mineral is a hydrous aluminum silicate, while talc is a hydrous magnesium silicate. But little has been found that is suitable for cutting into crayons and nearly all that is mined is sold after it has been ground. Both foliated talc and the massive steatite or soapstone are produced in North Carolina, although the soapstone variety is mined in but small quantity. The talc found in Swain, Graham and Cherokee counties is of exceptional quality and that from the Hewitt mine, Swain county, has an international reputation, the demand for certain grades of this talc being largely in excess of the supply. During 1908, talc was mined in Alleghany, Graham and Swain counties and pyrophillite in Moore county. The total value of the pro-

duction for 1908 was $51,442 and the quantity sold was 3584 short tons. This is a decrease of 501 tons in quantity and of $22,907 in value, as compared with 4085 tons, valued at $74,347, the production of 1907. This large falling off in the production was not confined to any one county or any one grade of talc but was felt throughout the whole industry and was due to the general business depression. The demand for North Carolina talc is increasing and it is expected that the production for 1909 will be largely in excess of that of 1908.

Pennsylvania and New Jersey.—Talc deposits occur near Easton, Pennsylvania, and on the other side of the Delaware river in the vicinity of Phillipsburg, Warren county, New Jersey. Most of these properties were idle during 1908 and in fact have only been operated in a desultory manner in recent years. The deposits of John O. Wagener & Co., near Easton, Penn., however, were operated in 1908.

Vermont. (By G. H. Perkins.)—Thus far, only a comparatively small number of the outcrops of talc in Vermont have been worked. Many of the deposits that are worked produce both soapstone and talc, but some of the talc mines do not furnish soapstone and conversely some of the soapstone deposits present little or no foliated talc. Talc is now taken out at Johnson by the American Mineral Company; at East Granville, by the Eastern Talc Company; at Windham, by the Vermont Talc Company; at Rochester, by the United States Talc Company. Only one of these deposits, viz., that of the Eastern Talc Company, is convenient to railroad transportation. The large deposit of the International Mineral Company at Moretown was not operated during 1908. In quality, the Vermont talc varies greatly. Some of it is exceedingly pure and white, while the product from other deposits is more or less injured by the presence of iron and silica. Some of the beds are worked by open cut, others by tunnels and shafts. Most of that mined is the ordinary massive talc, but in several localities there is foliated talc in small quantities.

Soapstone has been quarried and used in Vermont since the advent of the earliest settlers and in many parts of the State old and long-abandoned quarries are found. Few of these deposits, however, furnish sound slabs of such size as are needed for tubs and simliar objects. At present, soapstone is quarried at Chester by the Union Soapstone Company and the American Soapstone Finish Company and at Perkinsville, by the Vermont Soapstone Company. Each of these properties has a mill where the larger and more solid blocks are sawed into slabs, while the smaller blocks are trimmed and sold in that form. In most cases the soapstone occurs in lenses or seams inclosed in gneiss or gneissoid rock. The annual production of talc and soapstone in Vermont amounts to approximately $100,000.

Virginia.—Production in Virginia is almost entirely confined to the soapstone produced by the Virginia Soapstone Company and marketed by the Alberene Stone Company. The property is situated at Alberene, in Albemarle county. The product of these quarries is sawed into slabs and much of it is used in plumbing fixtures such as laundry tubs, laboratory tables, etc. The principal supply of soapstone in the United States now comes from Albemarle county and the Virginia Soapstone Company, with its affiliated marketing company, the Alberene Stone Company, is the dominant factor in this market. Another producer of importance in Virginia is the Old Dominion Soapstone Company, with quarries at Esmont.

TALC IN FOREIGN COUNTRIES.

Talc is produced by a number of countries in Europe but chiefly in France, Italy and Austria. The best grades come as a rule from Italy, though some of the French and Austrian properties produce talc of almost equally high grade. Foreign talc is imported into the United States in the form of rough rock, powdered talc, soapstone slabs for use in plumbing work, and blocks for making gas tips and other small articles of manufacture.

Austria.—The concession for all talc mines in Austria is held by M. Elbogen of Vienna, who has purchased the large deposit of talc found in the province of Styria in the commune of Floing. The production in Austria, however, is not equal to the domestic consumption and talc is imported from Italy. During 1908, however, some offers of talc from Vienna were received in the United States, though so far as can be learned, none was actually exported to the United States.

Brazil.—There are extensive deposits of talc in Brazil, but production is limited by the inaccessibility of many of these deposits. The present output comes from the State of Sao Paulo near the city of Sao Paulo and Loreno, particularly from deposits near the latter city. The talc is of good quality but is usually marketed in three qualities, graded according to color.

Canada.—The most important Canadian deposit now being worked is in the vicinity of Madoc in the province of Ontario. Official figures are not yet available, but the production during 1908 is estimated at 1500 to 1700 tons. Most of this tonnage was shipped crude and ground in the United States. In the latter part of 1908, however, a mill was erected on the property by local contractors, and some of the output was pulverized. This product is said to be of good color and grade, when sorted.

France.—The supply of talc and soapstone in France is abundant, occurring principally in the Pyrences, the most important deposits being near Luzech. These deposits are usually worked only about five months out of the year. Some of the talc is of excellent quality, nearly equalling that of the best Italian grades, though much inferior talc is also shipped. The principal ports of export are Bordeaux and Marseilles.

Italy.—Talc is mined in Italy in the northern part, especially in the vicinity of Pinerlo in the Italian Alps, and is usually shipped from Genoa. The premier grades of talc in the world are produced in this country, the Italian talcs bringing the highest prices and finding their main use in toilet and medicinal preparations on account of their superior color and absence of gritty or harsh material. Inferior talcs are of course produced, but the greater portion of Italian talc imported into the United States is of the higher grade.

Occurrence, Uses and Methods of Mining.

Occurrence.—Talc is of extensive occurrence, but many deposits are not so situated in relation to transportation and markets as to make their development profitable. There are several varieties, the most important being the foliated talc and the massive variety known as steatite or soapstone. It is a hydrous silicate of magnesium, $H_2O.3MgO.4SiO_2$, having a greenish, whitish or gray color. It is derived from the alteration of pyroxene amphibole, enstatite and other magnesium silicates, and is often associated with dolomite, serpentine or magnesite. Deposits of economic interest occur in the United States in most of the States lying along the eastern slope of the Appalachian range. Both the foliated talc and the amorphous steatite occur in nearly all of these States.

Uses.—Being highly resistant to all ordinary acids except hydrofluoric, talc in the form of soapstone slabs finds a wide use in chemical laboratories, hospitals, urinals, acid tanks, etc. It is also used extensively for making laundry tubs, sinks, griddles, hearthstones, gas tips, and for marking pencil such as tailor's chalk, pencils for marking on iron and glass, and the ordinary slate pencil of the schoolroom. Having great dielectric strength, requiring 30,000 to 40,000 volts to pierce a ½-in. slab, it is used for flooring of electric stations and other electrical uses. As the soapstone is soft and easily shaped, it is often used for making images, particularly in such countries as China and Japan.

Powdered talc is used in the manufacture of paper, toilet powders, foundry facings, sizing for cotton cloths, insulated coverings for wire,

facing for rubber molds, dressing skins and leather, and in inferior grades of soaps and paints. The "French chalk" of commerce is mainly powdered talc. Aside from its legitimate uses, it has been used as an adulterant in numerous trades and this fact is partly responsible for the attitude of secrecy prevailing among many producers and users, and also explains the reticence of jobbers of talc products in giving information or statistics concerning the uses of talc.

The value of powdered talc depends upon physical qualities and is judged by its color, and its feel or "slip." The whitest talcs or those with a bluish tinge command the highest price. The poorer grades have a yellowish hue, and a portion of the French output has this disadvantage. The Italian talc is the whitest and the best for such purposes as the talcum toilet powders which have come into such extensive use in recent years. Some of the better grades of domestic talc are also used in these powders. Talc of good "slip," or free from grit, is used in the sizing of cotton cloth. The color of the talc for this important trade is a secondary consideration but its freedom from grit, which dulls the cutting knives, is essential.

In the manufacture of book and writing paper, the fibrous talc is especially valuable as a filler as it increases the strength of the paper and reduces the brittleness characteristic of paper weighted with clay. Although apparently granular when ground to 100-mesh, its fibrous character is still maintained and is readily observed under a microscope. The fibrous talc is superior to the ordinary foliated talc in that a larger proportion is retained in the paper pulp. This is particularly true as compared with the china clay formerly used of which only about 30 to 35 per cent. was retained in the pulp as against 75 to 90 per cent. when the fibrous talc is used. In news paper, however, South Carolina clay is still used.

Mining Methods.—Occurring generally in flat beds or lenses without any well-developed cleavage, soapstone is usually worked by open cut, where possible, in order to obtain a large proportion of the output in the form of slabs, which are more valuable than the powdered form. Owing to its lack of cleavage it has to be cut on all sides to obtain the best results, and channeling machines have in consequence been introduced for this work at some of the Virginia quarries. The channeled blocks are taken to gang saws which produce slabs about 1½-in. thick. These may then be planed, grooved and finished on the rubbing beds for manufacture into various products, or shipped in slab form from the quarries.

When the talc is to be powdered and the deposit persists in depth, the usual mining methods are employed in breaking the mineral. In

the Gouverneur district in St. Lawrence county, New York, practically all the talc now comes from the underground workings which have attained a depth of over 200 ft. Blasting is practised and ventilation is easily accomplished as connections have been made in numerous instances with adjoining properties. The mines are comparatively dry and the mineral does not require artificial drying before going to the fine crushing department. In fact, on leaving the pebble mills the product has acquired sufficient heat to be readily perceptible when it is handled. Bolting of the pulverized product is not practised here as such fineness is not required in the talc used in the paper industry.

TANTALUM.[1]

By Pierre Breuil.

Tantalum is one of the rare metals, which, not long ago were interesting only to the scientist, but are now greatly sought after for use in one art or another. The principal use of tantalum is in the making of filaments for incandescent electric lights, competing with carbon and tungsten in this field. The principal literature on the subject is found in an article by Nicolardot, "L'Industrie des métaux secondaires et des terres rares" in the Encyclopédie Scientifique, published by Octave Doin, Paris, and in a *Bulletin* of the Imperial Institute, of London. According to Nicolardot, tantalum was discovered by Ekeberg, in a fossil from Ytterby, while the characteristics of the oxide of the metal were first studied by Klaproth and Rose.

Ores of Tantalum.

The most common ores of tantalum are tantalite and columbite. These minerals have a varying composition, but consist of tantalates and columbates (niobates) of iron and manganese. Tantalite contains around 84 per cent. of tantalic oxide, Ta_2O_5, while columbite contains only a little tantalum, and consists mainly of columbic oxide, Cb_2O_5, which has no commercial value at this time, and is generally troublesome. The weight of the minerals is a rough but serviceable guide as to their richness in tantalum, since it increases with the percentage of that metal; thus columbite has a specific gravity of about 5.3, while pure tantalite approaches 7.8 in specific gravity.

Tantalite.—This is a black mineral, the composition of which corresponds to $(FeMn)Ta_2O_6$, the iron frequently being replaced, wholly or in part, by manganese, and the tantalum by tin, zirconium and columbium. According to Nicolardot the general formula for the minerals of this order is $(FeOMnO)mCb_2O_5nTa_2O_5$, ranging from columbite, containing no tantalum, to skogböllite, containing no columbium. The hardness of tantalite varies from 6 to 6.5, the streak is black, and the specific gravity varies from 6.05 to 8.20. The crystallization is orthorhombic. It is infusible before the blow-pipe, and insoluble in acids, but if fused with potassium hydroxide and then boiled with a piece of tin, it gives a deep-blue solution.

[1] Translated from *Le Genie Civil*, Nov. 7 and 14, 1908, by Edward K. Judd.

Columbite.—This is the most common of the tantalum ores. It is generally found associated with tantalite, and is essentially a columbate of iron and manganese $(FeMn)Cb_2O_6$, combined with the tantalite molecule. The streak varies from dark red to black. Its hardness ranges from 5 to 6, and its specific gravity from 5.3 to 6.5. The crystals are usually orthorhombic prisms.

There are several other minerals containing tantalum, but as they are rare and contain only a small proportion of the metal, they are not of present importance, since tantalite now furnishes the necessary supply. As the utility of tantalum may expand, however, it may not be out of place to describe here a few of these minerals.

Fergusonite.—This is a metacolumbate of tantalum and yttrium, containing usually from 2 to 10 per cent. of tantalic oxide. The mineral occurs in massive form with a distinctive brownish black color; the streak is often colorless. Its specific gravity is 5.8. It generally occurs in granites and pegmatites and has been found in Ceylon, in Greenland and at Ytterby, Sweden; also at Rockport, Mass., Amelia, Va., and in the mica mines of Mitchell county, N. C.

COMPOSITION OF TANTALUM ORES. NICOLARDOT.

	Tantalite			Tapiolite Rammelsberg	Yttrotantalite Rammelsberg	Hielmite Karafvet	Fergusonite		Samarskite		Polycrase
	Harkösari Tammela	Sweden	Brodbo				Greenland	Ytterby	Miask	N. Carolina	
Ta_2O_5	76.34	65.60	42.15	13.91	49.26	54.52	6.40	28.50	1.26	18.60	4.87
Cb_2O_5	7.54	10.88	40.21	11.22	13.15	16.35	45.13	29.66	47.47	37.20	21.20
TiO_2											27.70
SnO_2	0.70	6.10	0.18	0.48	1.19	4.60	0.18		0.50	0.08	
WO_3			traces		2.52	0.28	0.45		1.36		
Fe_2O_3	13.90	8.95	16.00	14.47	4.06	2.41	0.74	0.76	11.02	10.90	0.47
MnO	1.42	6.61	1.07	0.81		5.68			0.96	0.75	
CaO					6.12	4.05	0.61	4.40	0.73	0.55	
MgO						0.15			0.14		
UO_2					1.72	4.57	2.62	2.24	11.60	12.46	5.80
Yttria					18.39	1.81	35.21	33.47	12.61	14.45	32.14
Ceria					2.37	0.48	7.76		3.31	4.25	3.05
ThO_2									6.05		
ZrO_2									4.35		

Samarskite.—This is a fairly common mineral, consisting of columbates and tantalates of iron, calcium, yttrium and cerium, mixed with oxide of uranium. It occurs massive, or in flattened grains or well developed rhombohedral crystals in pegmatite. It has a velvety, black color, with a reddish streak; it has a vitreous luster and a specific gravity of 5.7. It has been found, in masses weighing up to 9 kg. in the Wiseman mica mine, in Mitchell county, N. C.

Yttro-tantalite.—This is a tantalate and columbate of iron and calcium, containing oxides of yttrium, cerium and uranium; it contains about 46

per cent. of tantalic oxide. Its specific gravity is 5.8, its color varies from yellowish brown to black, and its hardness from 4.5 to 5. It is found, in small quantities, near Ytterby, Sweden.

Tapiolite.—This mineral, containing about 74 per cent. of tantalic oxide, has been found only at Sukala, Finland.

Stibio-tantalite.—This mineral has a pale yellowish red color and an almost white streak. Its specific gravity is around 7.4 and its hardness is 5 to 5.5. It is found in the tin mines of Greenbushes, South Australia.

Distribution of the Ores.

United States.—Most of the tantalum minerals are found in Connecticut and South Dakota; columbite is more common than tantalite. The compositions of columbite and tantalite from different points in the United States are shown in the accompanying table.[1]

TANTALUM MINERALS IN THE UNITED STATES.

Locality	Density	Cb_2O_5	Ta_2O_5	FeO	MnO
Branchville, Ct.	5.73	60.70	19.20	12.91	7.03
Branchville, Ct.	6.59	20.16	52.29	0.43	15.58
Etta Mine, S. D.	5.89	54.09	18.20	11.21	7.07
Etta Mine, S. D.	6.37	40.37	41.14	8.28	9.09
Etta Mine, S. D.	6.75	29.78	53.28	6.11	10.40
Yancey Co., N. C.	6.88	23.63	59.92	12.86	3.06
Grizzly Bear Gulch, S. D.	7.77	6.23	78.20	14.00	0.84

At Baltimore, a mineral has been discovered, which, when analyzed by Hilldebrand, gave the following results: Ta_2O_5, 38.19; Cb_2O_5, 13.21; SiO_2, 12.98; FeO, 21.42; MnO, 10.48 per cent.

Australia.—Tantalite is found in South Australia in connection with tin ores. Two samples from the Firniss river had the compositions shown in the accompanying table.

TANTALITES FROM SOUTH AUSTRALIA.

	Ta_2O_5	Cb_2O_5	MnO	SnO_2	FeO	Not det.
No. I	41.70	19.00	14.83	21.00	2.14	1.33
No. II	55.52	24.92	11.16	4.40	2.72	1.27

In eastern Australia, tantalum was first discovered, in 1894, at Greenbushes, Bunbury, in the form of stibio-tantalite. Tantalite was discovered in 1900 in the alluvial workings in the same district. In 1904, the mines at Wodgina produced a manganese-tantalite, and in the following year a manganese-columbite and a calcium-tantalite were discovered at Wodgina and Mt. York (Chingamoug).[2]

[1] See also *The Mineral Industry*, Vol. XVI.

[2] The occurrences of tantalum in Australia, which appear to be numerous, were described by Edward S. Simpson, in *Min. Journ.*, Nov. 28, 1908. *Editor.*

TANTALUM MINERALS FROM EASTERN AUSTRALIA.

Ore.	Locality.	Ta_2O_5	Cb_2O_5	Ore.	Locality.	Ta_2O_5	Cb_2O_5
Tantalite.....	Greenbushes ..	80.61	2.50	Mn-tantalite	Wodgina.....	69.95	14.47
Tantalite.....	Greenbushes ..	68.50	5.46	Mn-tantalite	Wodgina.....	72.46	6.80
Sb-tantalite.....	Greenbushes ..	51.13	7.56	Ca-tantalite.....	Wodgina.....	73.82	6.44
Sb-tantalite.....	Greenbushes ..	50.57	12.58	Mn-tantalite.....	Greens Wall	57.46	27.24
Sb-tantalite.....	Greenbushes ..	51.95	4.49	Tantalite.....	Lalla Rookh	70.34	4.92

STIBIO-TANTALITES FROM WESTERN AUSTRALIA.

	Ta_2O_5	Cb_2O_5	Sb_2O_3	Bi_2O_3	NiO	Fe_2O_3	CuO	SiO_2	H_2O	Sp. Gr.
No. I.....	51.13	7.56	40.23	0.82	0.08	tr	.....	.....	0.08	7.37
No. II.....	51.95	4.49	38.04	0.79	tr	0.39	0.30	3.14	0.61	6.60

Other Sources.—The Sempang Tin Mining Company, operating in Pahang, Malay States, recovers a monazite sand the components of which are: Cassiterite, 65; ilmenite, 16; monazite, 13; columbite, 3 per cent. Nicolardot states that tantalum and columbium are almost always present in tungsten ores, and thinks that these and tin ores might furnish a considerable yield of tantalum. The Pennsylvania Salt Manufacturing Company, of Natrona, Penn., has for several years found an important amount of tantalum in the residue from the Greenland cryolite that it consumes.

Value of Tantalum Ores.—The value of ores containing tantalum is subject to wide fluctuations. The *Bulletin* of the Imperial Institute states that marketable ores should contain not less than 60 per cent. of tantalic oxide, not more than 3 per cent. columbic oxide, and should be free from chromium. The value of such ore is about $1.10 per kg. According to Nicolardot, the price, at New York, is $20 per unit of tantalic oxide for ore carrying 22 per cent.[1]

Metallurgy of Tantalum.

Ekeberg, Klaproth, Gahn, Berzelius and Eggertz tried in vain to reduce metallic tantalum from its oxide by means of carbon; they obtained only a sub-oxide mixed with carbide. Berzelius obtained the metal by reducing the fluotantalate of potassium with potassium, which is the process in use today. The ore is first converted into alkaline double fluorates, which are then reduced by potassium or sodium, while only a little tantalic oxide is produced. Moissan has tried to reduce the oxide with carbon in an electric furnace, but has never succeeded in liberating the carbon, which seems to form a carbide with tantalum.

Method of von Bolton.—The process employed by von Bolton is to reduce basic oxides in a vacuum. This depends on the fact that metallic tantalum does not absorb oxygen at white heat, provided that the pressure does not

[1] The present demand for the ores of tantalum is very small and very irregular. *Editor.*

exceed 20 mm., and also on the fact that the pentoxide of tantalum is less volatile than the metal. In practice, the metallic sponge obtained from the treatment of fluotantalate, as just mentioned, is pressed into a magnesia or thorium crucible, and melted in a closed electric furnace. The crucible forms the anode while the cathode is made of a bar of pure tantalum or silver, which is constantly rotated. The deposited metal is compact and seems to contain no oxides. The amount of dust made by the current is very small. In 1905, von Bolton gave the Bunsen Society an ingot of tantalum weighing 64 grams, the production of which required 75 electric horsepower.

British Thomson-Houston Process.—This company purifies tantalum by melting the fluoride or some other salt of the metal, and then electrolyzing it between an anode of impure tantalum and a cathode of the pure metal, all being contained in a refractory vessel made of magnesia or of oxide of tantalum. (British patent No. 24,234, October, 1906.) The same company also has an improvement on the vacuum reduction process, employing mercury baths at both anode and cathode; the metal to be purified is plunged into the mercury. (British patent No. 21,667.)

Bouhard's Process.—Bouhard received a French patent in 1907 on a process for the making of tantalum and preparing it for use in incandescent lamps. He dissolves 100 grams of potassium tantalate in three liters of water; adds sulphuric acid until precipitation is complete, allows to settle, and then filters and washes the precipitate. This is then mixed with one liter of warm water (70 to 80 deg. C.) and dissolved by the addition of some hot, 20-per cent. solution of oxalic acid. The excess of oxalic acid is then neutralized with ammonia, and, after cooling, enough hydrochloric or sulphuric acid is added to form 3 per cent. of the liquid. The solution is then electrolyzed by a current of two volts and 0.1 to 0.3 amperes. The anode is of carbon or platinum, in rectangular shape, and the cathode consists of a series of horizontal platinum wires, or carbon rods. The metal is removed from the bath when it has reached a thickness of 3 to 4 mm., and is then made into a wire of 4 to 5 mm. diameter.

Method of von Pirani.—The refining process described by von Pirani takes advantage of the fact that tantalum has a great affinity for hydrogen; it will absorb 740 times its volume of the gas, of which 550 volumes are simply occluded. The absorption of hydrogen affects greatly the properties of the metal; it reduces its strength, increases its brittleness, decreases its electric conductivity by one-half, and lowers its coefficient of expansion by 0.1 to 0.3 per cent. The metal can be saturated with hydrogen at relatively low temperatures by the following means: Hydrogen and vapors of tantalum chloride are mixed and heated, yielding a very brittle, metallic mass. This is then heated to whiteness (considerably below its melting

point) in a vacuum, arranged so as to allow the liberated hydrogen to be constantly withdrawn. This operation can be done electrically, since the saturated metal is a good conductor. The way Von Pirani does this is to place some of the hydrogen-bearing metal at the positive pole of an arc surrounded by a vacuum, the negative pole being composed of barytes. The latter is first heated to redness and the current is then allowed to pass, whereby the metal is melted and refined.

Properties of Tantalum.

Physical Properties.—Pure tantalum melts at about 2200 deg. C., its hardness varies from 9 to 10, its coefficient of linear expansion is 0.0000079 and its specific heat is 0.0365. Its specific gravity varies from 16.5 to 16.6; a wire heated by an electric current contracts by 7.8 per cent. of its length, at the end of 12 hours, and becomes correspondingly more dense. It has good tensile strength; a wire of 0.03 mm. diameter breaks with a load equivalent to 93 kg. per sq. mm., as compared with 200 kg. per sq. mm. for fine steel. Its ductility is high. It has the unusual characteristic of becoming softer under hammering. Carbon, up to 1 per cent., softens tantalum without increasing its ductility; above that percentage, it makes the metal brittle. The presence of oxides seems to soften it.

Chemical Properties.—As an element, tantalum is similar to gold and platinum. In bars, it combines very slowly with oxygen of the air. By heating, it will combine with sulphur, and liberates heat by so doing; its reactions with selenium and tellurium are similar. Boiling hydrochloric acid, nitric and sulphuric acids and aqua regia have no effect on the metal, nor have alkaline solutions. Alkalies in fusion, and hydrofluoric acid attack it. The latter reaction is very slow if a piece of tantalum is put into the acid alone, but if in contact with a piece of platinum, the solution is rapid. Tantalum forms nitrides, and even nitro-carbides. The carbide has the formula, Ta_2C_2, and contains 6.18 per cent. of carbon; it is a yellow material insoluble in acids.

Alloys of Tantalum.

Berzelius alloyed tantalum with iron and tungsten, and Marignac alloyed it with aluminum; the preparation of such alloys is difficult, with no present utility to make it worth while. Iron alloyed with 5 to 10 per cent. tantalum is very hard, yet ductile. Guillet examined four soft steels containing different proportions of tantalum, and found them of no practical use. It is possible that in a steel higher in carbon the effect of tantalum might be more pronounced. Tantalum has the peculiar property of transforming alternating into direct currents. With tensions of less than 120 volts, two electrodes of tantalum dipping into a bath of dilute sulphuric

acid will stop an alternating current completely; if one electrode be replaced by one of platinum, the current will pass, in one direction only.

Tantalum Lamps.

Tantalum has found its greatest utility in the manufacture of incandescent electric lamps. The advantages of the tantalum lamp are stated to be: The consumption of fewer watts per candle power; the whiteness of the light; and the ability to carry a momentary increase of current without burning out. The filament for a 25-c.p. lamp is about 650 mm. long, 0.05 mm. in diameter, and weighs 22 miligrams, so that a kilogram of the metal would provide material for 45,000 lamps. It is mounted between two star-shaped poles made of steel or nickel, passing zigzag from one to the other; if one strand breaks, it merely falls out of the way, and the other strands continue in service. The glass bulb sometimes becomes black, probably from the presence of columbium.

H. F. Haworth, T. H. Mathewmann and D. H. Ogley reported to the Society of Electrical Engineers of London, in February, 1907, their results on a long series of tests on the efficiency and durability of carbon, Nernst, and tantalum lamps. Some of the lamps were run at normal voltage, others at 5 per cent. excess voltage, and others with a current which fluctuated, at intervals of two minutes, between normal and 5 per cent. excess voltage. The average number of watts per candle-power was 4.86 for the carbon lamps, 4.14 for the Nernst lamps, and 1.97 for the tantalum lamps. The Nernst lamp was therefore 15 per cent. more economical of power than the carbon, but lasted only 560 hours; the tantalum lamp used 60 per cent. less power than the carbon, but lasted only 330 hours. The tantalum filament has the advantage of increasing in resistance as the temperature rises.

Hermann Zipp gives his results as follows (*Chemiker Zeitung*, Feb. 22, 1908): The tantalum lamp lasts 400 to 600 hours, and requires 1.7 to 2.1 watts per candle-power; the Nernst and the osmium lamp last respectively 300 and 1000 hours and require 1.4 to 1.7 watts per candle-power; the theosram, tungsten, zirconium, and other similar lamps last 500 to 800 hours and require 1.1 to 1.4 watts per candle-power; the ordinary arc lamp uses 1 to 1.5 watts per candle-power; and the Cooper Hewitt mercury vapor lamp lasts for 1000 hours and requires 0.3 to 0.5 watt per candle-power.

The Siemens & Halske company states (*Chemiker Zeitung*, No. 26, 1908) that its tantalum lamp is no more subject to breakage than the carbon lamp, and that its types A and B last 800 hours, requiring 1.5 to 1.7 watts per candle-power, and that its type C lasts 1000 hours with a consumption of 2.1 watts per candle-power.

TIN.

This metal is not produced in the United States to any significant extent, but we use 40 to 50 per cent. of the world's production, which is made chiefly by Malaya, the Dutch East Indies, and Bolivia. In 1907 the importation into the United States was 82,549,000 lb; in 1907 the amount was 82,503,190 lb. The price suffered heavily. In June, 1907, the average was 42⅛c., New York. The average for the year was 38.16c. The average for 1908 was 29.465c. January, 1908, showed the lowest monthly average, viz.: 27⅜c. The failure of tin to make more of a recovery appears to be due to continued large production. At the beginning of the year the prognostications in Malaya were for an extensive suspension of mining. Many of the corporations did indeed suspend, but apparently the miners went back to old diggings and made a new production by primitive methods.

IMPORTS OF TIN INTO THE UNITED STATES.

Year.	Pounds.	Value.	Year.	Pounds.	Value.	Year.	Pounds.	Value.
1900...	69,989,502	$19,458,586	1903...	83,133,847	$22,265,367	1906...	101,027,188	$37,446,508
1901...	74,560,487	19,024,761	1904...	83,168,657	22,356,896	1907...	82,548,338	32,075,091
1902...	85,043,353	21,263,337	1905...	89,227,698	26,316,023	1908...	82,503,190	23,932,560

The production of concentrated tin ore in the United States in 1908 was 50 tons, valued at $12,500, against 63 tons valued at $15,200 in 1907. Practically all of this production was made in South Carolina,

THE PRINCIPAL TIN SUPPLIES OF THE WORLD. (a)
(In tons of 2240 lb.)

	1900	1901	1902	1903	1904	1905	1906	1907	1908
English production	4,268	4,566	4,392	4,282	4,132	4,468	4,522	4,700	4,650
Chinese exports						4,482	4,052	(b) 4,000	(b) 4,000
Straits to Europe and America	46,058	50,339	51,831	52,212	57,419	56,840	57,143	52,520	60,463
Straits to India and China	1,785	2,655	1,882	3,123	3,261	1,484	1,292	3,140	2,190
Australia to Europe and America	3,235	3,345	3,199	4,934	4,846	5,028	6,482	6,612	5,921
Banka sales in Holland	12,631	14,978	14,978	15,070	11,363	9,960	9,286	11,264	13,900
Billiton sales in Java and Holland	5,882	4,387	3,897	3,650	3,215	2,715	1,968	2,229	
Bolivian arrivals on Continent	1,900	9,670	10,150	9,630	12,978	14,245	16,394	15,500	17,000
Bolivian arrivals in England	5,065								
Totals in long tons	80,824	89,940	90,329	92,901	97,214	99,252	101,139	99,965	108,124
Totals in metric tons	82,117	91,379	91,774	94,387	98,769	100,840	102,757	101,564	109,854

(a) Compiled from commercial reports. There is also a small production in Germany. The apparently large increase in the total for 1905 is due to the inclusion of the Chinese exports for the first time. (b) Estimated.

but very little being obtained from Alaska. Beside the production of tin ore in the United States, there is a fairly large production of metallic tin, recovered from tin plate scrap. In 1907 the amount of the metal thus produced was 1662 tons; in 1908 it was about 1200 tons.

The principal producer of metallic tin in the United States is the Vulcan Detinning Company. A new concern of probable importance was organized in 1908 under the name of the Goldschmidt Detinning Company, which has made a contract with the American Can Company for all of its tin scrap for a long period of years. The process of detinning with chlorine will be employed. This process has been in successful use in Germany, at the works of Theodor Goldschmidt, Essen-Ruhr, for the last four years, and at the plant of Gillet Fils, Lyons, France. The company has already acquired 10 acres of water front property on Staten Island sound, New York bay, near Carteret, N. J., where it is arranging to erect a detinning plant with a capacity of approximately 40,000 tons of tin scrap per year. Later, it is contemplating to erect another detinning plant in the West with a like capacity. It has been stated that the new company will control over 50 per cent. of the tin scrap produced in America.

According to K. Goldschmidt, the amount of tin plate scrap treated annually in Germany is about 75,000 tons; in other European countries about 25,000 tons; in the United States, about 60,000 tons; a total of 160,000 tons. The amount of tin recovered is from 3000 to 3500 tons. About 50,000 tons are treated annually in Germany at the Goldschmidt works, the remaining 25,000 tons being handled at eight to 10 other works. The detinning industry has grown extensively in the lower Rhine district of Germany, owing to the admission of the raw material free of duty, the bulk of the supply being obtained from abroad. There is a ready market in the vicinity for the detinned iron. Of late, however, better prices have been obtained from English steel makers, who have been willing to pay about 50s. per ton against 40 to 42s. offered in Germany.

Tin Mining in the United States.

Alaska.—According to Adolph Knopf, of the U. S. Geological Survey, most of the developments in the tin field of Alaska are still in the prospecting stage; many of the open cuts have not yet uncovered solid bed rock. No tonnage of tin bearing ore has yet been developed except at one place on Lost river. Small holes in the ground, which give no clue to dip, strike or persistence of the ore are held at enormous figures. The great need of the country is less desultory prospecting.

South Dakota.—The Gertie Mining and Milling Company of Hill City, South Dakota, was in litigation during 1908 and consequently did no mining. The company has now recovered possession of its property and expected to begin operations in the spring of 1909. It is interesting to note that the litigation between the American and British stockholders of the Harney Peak Tin Mining, Milling and Manufacturing Company, which has been before the courts for 14 years, has been ended (April, 1908) and a reorganization is in process of being effected. During all these years Dr. A. D. Ledoux has been receiver for the company, but has been prohibited by the courts from spending any money in development of the mines. Under the terms of the reorganization the first thing to be done is to make a careful test of the most promising properties to determine what is the grade of any ore found in quantity. It is gratifying to know that through this settlement there is at last a possibility of establishing a tin industry in South Dakota.

Texas. (By Walter E. Koch.)—The tin prospects on the eastern slope of Mt. Franklin, about 10 miles north of El Paso, were developed further in 1908 with gratifying results. A company has been organized to exploit the property, and the ore is now being tested to determine the best method of concentration. It is probable that a concentrating mill and a smeltery will soon be erected. The granite masses in places have been found to be impregnated with cassiterite, which varies in color from very pale to very dark crystals. This fact doubtless caused it to be overlooked or mistaken for other minerals. Careful and constant assaying is required. The use of a microscope with a 100-diameter object glass is necessary to detect the various forms in which the cassiterite occurs.

Tin Mining in Foreign Countries.

Australia. (By F. S. Mance.)—The Mount Bischoff company, of Tasmania, opened several new bodies of high-grade ore in 1908. The mines in the Derby district, including the Briseis and New Brothers Home, experienced a very satisfactory year, producing about 1500 tons of black tin; future prospects, however, indicate a reduction in the output. In New South Wales the alluvial deposits in the Tingha and Emmaville districts were worked systematically, operation of the dredges, although somewhat hampered by an insufficient rainfall, being fairly satisfactory. Several small but rich lodes were also worked. In Queensland the output was on a much reduced scale, the production for 1908 having been only 4792 tons. The Walsh & Tinaroo field continued to be the chief center of operations, but results fell short of those of 1907, because two of the large producing mines, viz., the Smith

Creek and Stannary Hills suspended operation, pending reconstruction of the companies. The setback was somewhat ameliorated by the opening of rich ore in the Vulcan mine, where the lode has now been proved to a depth of 1400 ft. The Great Northern Freehold mine at Herberton also contributed improved returns, while the dredging industry at Stanthorpe made good headway. Of the tin field in the Northern territory but little was heard during 1908. Operations at the Pilbarra and Greenbushes fields in Western Australia were unfavorably influenced by the adverse market conditions.

The Mount Bischoff company in the first half of 1908 produced 450 tons of tin, and 480 tons in the second half. The yield per ton of ore in the first half was 12.9 lb.; in the second half, 11.83 lb. Ore reserves have been increased from 2,518,000 tons to 3,000,000 tons. The cost of mining, milling and smelting has been substantially reduced. The average grade of the concentrate smelted in the second half of 1908 was 67.37 per cent. against 69.94 per cent. in the corresponding period of 1907.

Austria.—An application for 16 claims, for working tin ore in Sauersack and Hirchenstand, in Bohemia, has been registered at Teplitz by the Erzgebirgische Schurf-Gesellschaft für Zinnbergbau in Frühbuss, which intends to commence work in 1909.

Banka and Billiton.—According to the official report for the year ended Feb. 1, 1908, the average number of coolies employed was 19,106, as against 14,231 in the previous year. In spite of this increase, there was but little gain in production, the deliveries amounting to 11,516 tons against 11,402 tons in the previous year. The cost of the tin delivered at Batavia in 1907-08 was £3 12s. 0.6d., against £2 19s. 11.7d. in the previous year, the increased cost being due chiefly to a decrease in the individual production, owing to the large number of fresh laborers employed and the heavy rainfall in December, 1907, which inundated several mines in the Belingoe district. The shipments of tin from Batavia in the first 11 months of the calendar year 1908 were 10,169 tons.

During 1908 the sales of Billiton tin amounted to 66,600 slabs, which fetched an average of £7 14s. 4d. per pikul. In the previous four years the quantities and prices were as follows: 1907, 66,700 slabs, £10 6s. 1.6d.; 1906, 58,000 slabs, £10 13s. 5.4d.; 1905, 81,300 slabs, £8 3s. 2.8d.; 1904, 95,900 slabs, £7 7s. 4.6d.

Bolivia.—Tin mining in this country in 1908 suffered from the low price for the metal, and consequently the majority of the enterprises were not over-prosperous, although the fall in exchange (the boliviano being worth only 16d. toward the end of the year) was an ameliorating

factor. The high price for tin in 1906 imparted great activity to tin mining in Bolivia, many foreign companies, especially Chilean, entering the field and purchasing properties at inflated value. In 1908, there was the natural reaction. Nevertheless, a good deal of new work was inaugurated. Chorolque, the largest producer of the Republic, enlarged its plant so as to produce at least 450 tons of concentrate per month. Some plants were also installed in Quimsa Cruz, and one at Colcha, in the Cochabamba district, which will increase the production. The unfavorable labor conditions are still troublesome. The program of railway construction in Bolivia is being carried on in a half-hearted way, and has not yet benefited the mines to any great extent, most of them being still very far distant from rail.

Bolivia now has a new plant coming into operation, and doubtless its production will continue to increase. The import of Bolivian ore into Great Britain was 20,470 tons in 1908 against 16,460 tons in 1907.

Gilmour G. Brown, in London *Min. Journ.*, March 27, 1909, gave some valuable data respecting the Bolivian tin industry. Miners can be readily obtained from 2s. 8d. to 6s. per day of 10 hours, according to the altitude of the mine, the lower figure being the average at an altitude of 13,000 ft. Indian unskilled laborers are paid 1s. 6d. to 3s. per day. As an example of working cost, a good mine not far from the railway in the Oruro district has regular veins of full stoping width,

EXPORTATION OF TIN FROM BOLIVIA.
(In metric tons.)

Year.	Barrilla. Tons.	Metallic Tin. (a) Tons.	Year.	Barrilla. Tons.	Metallic Tin. (a) Tons.
1898....	4,327	2,596	1904....	20,369	12,221
1899....	9,134	5,480	1905....	27,690	16,614
1900....	15,088	9,053	1906....	29,370	17,622
1901....	21,573	12,943	1907....	27,678	16,607
1902....	17,340	10,404	1908....	31,223	18,734
1903....	21,785	13,071			

(a) Tin content of the barrilla (black tin concentrate), computing the latter at 60 per cent. metallic tin.

LIVERPOOL RECEIPTS OF BOLIVIAN TIN.
(In tons of 2240 lb.)

Year.	Bars.	Ore.		Total Tin.
		Crude Weight.	Metallic Content.	
1900.....	1,507	5,431	3,530	5,037
1901.....	1,730	9,086	5,905	7,635
1902.....	1,685	10,961	6,576	8,261
1903.....	1,614	10,401	6,240	7,854
1904.....	1,573	13,824	8,294	9,867
1905.....	1,386	17,504	10,504	11,888
1906.....	1,569	20,489	12,293	13,862
1907.....	1,143	18,532	11,119	12,262

and a good stamp mill. Its working cost is £4 per ton. Freight and smelting charges, commissions, etc., are high, varying from £28 to £35 per ton of concentrate, the higher figure being for concentrate containing 50 per cent. tin, below which grade it is scarcely profitable to ship. Transportation is done very cheaply by llamas at a cost of less than 0.2d. per mile per 100 lb., a llama carrying from 40 to 100 lb. as a load.

China. (By T. T. Read.)—China ranks as one of the more important producers of tin; practically all the output is derived from the province of Yunnan, in the extreme southwestern part of the Empire. The best known mines are at Ko-chi'u-ch'ang, not far from Mengtze, the provincial capital. No details regarding the deposits and methods of working seem to be available. The base metal in pigs is shipped out through Indo-China and back to Hongkong to be refined. A certain amount of tin also comes in native boats down the Hsiang and Yuen rivers, and as it is impossible that it has been transported so far overland as to reach these rivers, it must originate at some point in Kuei-chou or Hu-nan. Kuangtung also produces a not inconsiderable amount, which is exported through the port of Wu-chow. There are also deposits in Fukien province, and a small amount is shipped from Foochow. I have been unable to obtain the figures for 1908, but the amount which passed through the customs in 1907 was 3100 tons. The actual production was undoubtedly larger than this.

Great Britain.—The output of black tin in Cornwall in 1908 was 6520 tons, realizing £505,730, against 5720 tons worth £593,704 in 1907.

BLACK TIN SALES OF SOME OF THE PRINCIPAL CORNISH MINES.
(In tons of 2240 lb.)

Year.	Dolcoath.	Carn Brea.	Basset.	Grenville.	East Pool.	West Kitty.
1900..	2,004	436	533	560	682	436
1901..	2,035	650	568	712	647	534
1902..	1,838	600	793	719	626	316
1903..	1,739	547	728	772	805	389
1904..	1,705	628	613	617	607	340
1905..	1,696	783	768	787	559	395
1906..	1,813	781	696	680	678	357
1907..	1,708	736	733	604	557	326
1908..	1,766	1,001	715	636	711	288

OUTPUT OF CORNISH SMELTERS IN 1908.
(In tons of 2240 lb. £1=$5.)

Smelters.	Tons.	Value.
Consolidated Tin Smelting Co. (Chyandour)....	1,501	$583,941
Williams, Harvey & Co.....	1,670	670,875
Redruth Tin Smelting Company	869	323,589
Penpoll Tin Smelting Company.....	431	152,455
Cornish Tin Smelting Company.....	2,047	801,373
Total.....	6,518	$2,532,233

AVERAGE PRICES OF METAL AND ORE AT CORNISH TICKETINGS. (a)

1908	Computed Tons.	Average Price per Ton of Black Tin.	Average Price of Metallic Tin per Ton on Day of Ticketing.	Average Price of Black Tin per Ton on Corresponding Day of Ticketing in 1907.	Average Price of Metallic Tin per Ton on Corresponding Day of Ticketing in 1907.
January 13	228	£76 17s 2d	£124 0s 0d	£110 12s 11d	£188 10s 0d
January 27	225	77 3 8	124 0 0	114 2 4	191 5 0
February 10	250	78 18 10	129 5 0	115 0 1	191 0 0
February 24	233	80 4 7	131 15 0	117 8 3	193 15 0
March 9	272	81 6 7	134 5 0	117 8 8	192 10 0
March 23	237	84 18 2	140 15 0	112 16 2	180 0 0
April 6	278	86 6 4	145 10 0	114 18 7	184 15 0
April 22	241	86 10 6	144 0 0	118 2 2	188 10 0
May 4	281	84 14 9	143 5 0	120 3 11	192 5 0
May 18	241	80 5 4	136 15 0	118 17 2	189 15 0
June 1	261	78 12 6	126 15 0	118 5 11	189 0 0
June 15	232½	75 1 9	129 0 0	117 16 0	188 0 0
June 29	286	72 11 7	125 10 0	118 5 3	192 10 0
July 13	195	74 17 2	129 10 0	114 10 11	186 0 0
July 27	256	77 17 9	137 0 0	110 11 6	181 15 0
August 10	222	77 12 7	137 0 0	104 16 9	171 15 0
August 24	262	78 19 9	139 17 6	101 0 10	167 2 6
September 7	236	72 6 9	131 10 0	102 8 6	167 15 0
September 21	260	69 15 11	130 5 0	102 13 6	170 0 0
October 5	235	74 0 11	133 15 0	91 14 2	155 10 0
October 19	281	72 3 0	132 0 0	82 15 0	139 10 0
November 2	243	78 10 7	137 10 0	86 3 6	144 5 0
November 16	274	77 2 1	135 7 6	81 3 9	135 10 0
November 30	244	78 10 3	135 10 0	80 9 8	133 5 0
December 14	596	75 3 10	130 15 0	72 8 6	119 10 0
December 30	247	76 10 0	131 2 6	75 3 10	122 0 0

IMPORTS OF TIN ORE INTO THE UNITED KINGDOM. (a).

Country.	1906		1907		1908	
	Tons.	£	Tons.	£	Tons.	£
Africa—						
Cape of Good Hope	133	14,111	119	14,525	64	5,498
Natal	11	705	254	15,672	2	30
Niger Protectorate	.....	.....	75	7,208	464	40,307
Portuguese E. Africa	317	27,109	1,214	64,217	2,076	110,649
Madagascar	*Nil*	*Nil*	*Nil*	*Nil*	1	50
America—						
Bolivia { Peru	267	18,319	420	28,325	1,457	94,541
Bolivia { Chile	17,285	1,307,155	15,786	1,350,771	18,437	127,052
Bolivia { Argentine	74	4,684	254	14,819	576	34,877
Canada (Atlantic Ports)	6	346	17	1,290	20	1,195
U. S. Atlantic Ports	9	651	56	3,113	45	2,529
U. S. Pacific Ports		100			.....	28
Asia—						
Bengal	2	20	.....	.....	.....	.....
Burmah	1	.....	.....	.....	.....	.....
Straits Settlements	86	7,785	13	1,643	.....	.....
Australasia—						
New South Wales	599	59,049	46	4,290	37	2,793
Queensland	.....	.....	11	1,215	1	45
South Australia	.....	.....	1	95	.....	.....
Tasmania	.....	.....	.....	.....	19	1,190
Victoria	20	1,218	28	2,420	52	4,645
Western Australia	2	82	.....	.....	19	1,500
New Zealand	.....	.....	.....	105	.....	.....
Europe—						
Austria-Hungary	.....	.....	.....	.....	.....	24
Belgium	106	2,780	67	1,785	107	2,594
France	622	40,717	696	29,938	2,346	26,176
Germany	593	16,478	1,182	54,942	193	7,496
Italy	35	660	129	1,814	89	4,362
Netherlands	251	8,650	397	16,061	224	14,267
Norway	.....	.....	.....	.....	.....	28
Portugal	17	1,561	5	485	9	458
Russia, Northern Ports	22	525	72	3,051	12	404
Russia, Southern Ports	14	270			10	547
Spain	195	12,611	256	18,878	211	11,221
Sweden	3	366	.....	.....	.....	.....
Total	20,672	£1,525,926	20,781	£1,635,481	25,015	£1,640,656

(a) From the *Min. Journ.*

One of the principal of the Cornish tin smelters has decided to establish a smelting works at Bootle, Liverpool. The company at present has to carry all its foreign ores from Liverpool to Cornwall, and return the refined tin to Liverpool to be disposed of on the northern markets, or shipped abroad. One of the Cornish tin smelters opened works at Bootle some years ago, which have been very successful, owing largely to their having had the advantage over the smelters in Cornwall, for the reason above stated.

Malaya.—In spite of the low price for tin and the prognostications of disaster to the industry in the Federated Malay States, their production in 1908 showed a material increase. This was contrary to expectations, many authorities agreeing that they cannot see how tin can be produced at anything but a loss at the average price for the year, namely, about $66.50. According to the London *Min. Journ.*, Jan. 23, 1909, careful estimates in the case of a representative Chinese enterprise, on the basis of the wages ruling last Autumn indicate $70 as a figure below which working involves loss. In many cases, however, the Chinese mine owners hardly know where their exact limit comes. They make their profits largely from truck, and as they work from hand to mouth so far as development is concerned, they may find richer ground which will pay to work where the average of the mines would not. The worst disaster to a mine worked on Chinese methods is to be closed, inasmuch as then the profits made from the coolies are lost, wherefore there is every incentive to keep working.

PRODUCTION OF TIN IN THE FEDERATED MALAY STATES.

(In pikuls of 133⅓ lb.)

	1900	1901	1902	1903	1904	1905	1906	1907	1908
Perak	355,590	385,060	405,870	436,296	442,507	446,781	435,909	431,386	467,784
Selangor	269,490	302,570	278,360	284,592	300,413	289,867	268,624	273,900	282,540
Negri Sembilan	82,320	75,230	73,520	85,461	84,849	85,133	77,766	75,155	64,221
Pahang	15,700	26,310	23,120	25,317	27,469	34,879	34,488	33,195	39,520
Total	723,100	789,170	780,870	831,666	856,238	856,660	816,787	813,636	854,065
Metric tons.	43,123	47,713	47,211	50,254	51,790	51,793	49,359	48,411	51,654

Attention in Malaya is now being turned to lode mining. The first lode mine in the States which has commenced to make monthly returns is the Pahang Consolidated, in the State of Pahang. In October 3950 tons of ore were crushed, producing 119 tons of concentrate. It is the intention of the management to double the stamping power, and in a short time to produce 200 tons of tin ore per month. In the State of Perak also, attention is being turned to lode mining.

In the granite hills around the Kinta valley some valuable lodes have been discovered in the Chendai mine, and a local syndicate has been formed to test them. This syndicate has erected a 5-stamp mill, and is crushing monthly about 300 tons of ore, which yields about 15 tons of concentrate. This works out at about 5 per cent. concentrate, or 112 lb. of tin ore to the ton. The average produce of Dolcoath mine in Cornwall is about 38 lb. of tin ore to the ton. Another syndicate, known as the Menglembu Lode Syndicate, whose property joins the Chendai mine, has erected a 10-stamp mill and is thought to have a better mine than the Chendai.

Mexico.—A number of tin properties in the State of Aguascalientes are reported to have been purchased by Americans and are being developed. Deposits of tin ore have been discovered in Coahuila, near the Rio Grande river opposite Boquillo, Texas.

Nigeria.—According to Sir William Wallace, the resident general of northern Nigeria, the Bautchi highlands are now open to British trade. This district is believed to contain some very rich tin deposits. Progress is being made in the construction of the railway from Lagos to Jebba on the Niger river. A scheme is being executed for deepening the Niger to a uniform depth of five feet.

The Niger Company has continued work on its licensed areas in the Bautchi province, and is now exporting about 500 tons of black tin per annum. The main difficulty in the development of this promising industry is its situation. With the construction of the railway through Zaria it should be possible to place the mines in close connection with it by means of a road, which should also serve the Bautchi province.

Portugal.—Tin mines exist in the districts of Castello-Branco, Vizen, Aveiro, Oporto, and Braganza. The most important are in the provinces of Traz-os-Montes and Beira-Alta.

Singkep.—The Singkep Tin Company in its fiscal year ending June 30, 1908, produced 6619 Dutch Indian pikuls of tin. In the tin production is included 177 pikuls from the Kedah Estate, which was closed and abandoned in December, 1907, on account of exhaustion of the ground. The tin ore of Singkep and Kedah, which is delivered to the smelting works of the Straits Trading Company at Singapore and at Penang, and sold as Straits tin through the agency of the Dutch Trading Company at Singapore, averaged $66.99 per Straits pikul, equal to F. 95.78 (£7 19s. 7 3-5d.) per Dutch Indian pikul. The figures for the previous year were, respectively, $92.87—F. 134.13 (£11 3s. 6 3-5d.). The profit and loss account points to a considerable economy in the expenditure on the estate, as a result of the introduction of the hydraulic method of working and the working by tender. The management is

of the opinion that as it has been proved that favorable results are obtained, the extension of mechanical labor must be pursued with all energy. By such extension the number of coolies required will be decreased, and, consequently, the cost of labor will become less. There were operated 10 pits, two hill mines with ordinary washing, one hill mine with the hydraulic method of work, and in four places ore was obtained by means of tunnel borings. On the "Sultan" foreshore estate work was continued without any difficulties being experienced. The sea wall was kept in good repair, and proved to be able to resist the seas, which during the year were very heavy. The pit was easily kept dry. A production of 1,254.65 pikuls was obtained. This working will be capable of being continued for one year more in the same manner, after which the ground within the sea walls will be exhausted. The working will then be discontinued as the ore in the direction of the sea becomes too poor to pay for the cost of further walling in. From the new working in the Putie valley a production of 336 pikuls was obtained. The hydraulic installation for the exploration of the Tumang hill by means of the sluicing with water under pressure gave satisfactory results. In places where the surface ground has been sluiced away, and where the rock has been laid open on the hill, pockets have been noticed containing tin ore and wolframite. Full details of the production in Dutch Indian pikuls are as follows: From the mines, 3954; from the hill exploration, 618; from the tunneling, 360; from washing and various small estates, 311; from the hydraulic exploration, Tumang, 1196; from the estate Kedah, 177; total, 6619. The production in recent years in pikuls has been as follows: 1897-98, 11,280; 1898-99, 11,143; 1899-1900, 9533; 1900-01, 12,998; 1901-02, 9977; 1902-03, 7254; 1903-04, 4618; 1904-05, 6007; 1905-06, 7487; 1906-07, 6715; 1907-08, 6619.

Sumatra.—According to the *Indische Mercuur*, of the tin formerly—at the time of the existence of the East India Company—obtained from the Kota Kampar territory, only a small portion is accounted for at present, so far as its source is concerned. It seems likely, however, that the old workings are being found again. Near one of the branches of the Sitingkei river, called the Soengei Kasik Pantei, tin ore has been found.

Swaziland.—This colony, which exported 409 tons of tin ore in 1907, increased in 1908 to 535 tons. The mines of the district continue to maintain a good reputation. The McCreedy Tin Company, Ltd., is developing its property and expects to be able to produce 600 tons of concentrate per annum.

Transvaal.—Much interest was exhibited in tin mining in this colony in 1908, particularly in the Waterberg district, and a considerable production is already being made. This district is situated about 22 miles northwest of Potgietersrust in the mountainous belt of country which separates the Magalaquene valley from the Sterk river. The occurrence of tin in this district was described by H. Kynaston in the report of the Transvaal Geological Survey for 1907. In the year ending August 31, 1908, the Groenfontein mine, with a 10-stamp mill, crushed 3371 tons of ore, yielding 215 long tons of concentrate, assaying 72.5 per cent. tin. The Zaaiplaatz Tin Company installed a 10-stamp mill which is expected to crush 1000 tons of ore per month, and produce 90 to 100 tons of concentrates. The occurrence of tin in this district is peculiar and of much geological interest. The Rooiberg Minerals Exploration Company in the first nine months of 1908 produced 1109 tons of concentrates.

H. Kynaston, in his monograph published by the Transvaal Geological Survey, in describing the principal deposits of cassiterite occurring on Zaaiplaats and the adjoining farms, stated that they are found along a fairly well-defined belt or zone in the granite, extending from the northeastern corner of Zaaiplaats in a general southeasterly direction, across the northern corner of Roodepoort and the southern portion of Groenfontein, thence trending in a slightly more southerly direction across the west corner of Sterkwater into the northern portion of Solomon's Temple. According to *South African Min. Journ.* of Jan. 30, 1909, the results of recent prospecting work appear to indicate a further extension of the belt both to the north and south. The width of this tin-bearing zone varies considerably, the average being probably about a quarter of a mile. On Roodepoort, Groenfontein and Sterkwater it probably does not exceed 1500 ft. On its southwest side it is limited by the line of quartz and pegmatite veins, though on the northeast it does not appear to be sharply defined by any very well-marked or conspicuous geological boundary. Throughout this zone the principal and most important cassiterite deposits are found in the form of peculiar elongated cylindrical pipes or pipe-veins.

That the Waterberg district will soon be producing very much more tin than at present is certain. Apart from the contemplated increase in the plant of the Transvaal Consolidated Land Company and the erection of crushing and concentrating machinery on Rooiberg, it may be pointed out that the new mill of Messrs. Maggs and Munro, near Potgietersrust, and the battery and concentrating plant of the South African Tin Mines on the farm Doornhoek, near Nylstroom, may soon be in operation. Among alluvial deposits the Transvaal Bischoff property

has, according to Jorissen and Hoffmann, three and a quarter million cubic yards of alluvial ground on its property, from which large profits may be won.

Zululand.—A discovery of tin ore in the Melmoth district has been reported. The formation consists of crystalline and garnet schists containing lenses of hematite, and the tin occurs in pegmatite veins and bodies. There is said also to be a large amount of tin as payable alluvial. The Premier Zululand Tin Company has been organized to work in this field.

The Tin Markets in 1908.

New York.—In sympathy with all other metals, the consumption of tin in the United States suffered during 1908. For this reason the statistical position became more unfavorable from month to month, and at the end of the year visible supplies had increased to about 20,000 tons, as against about 13,000 tons in December, 1907. The market for tin is influenced entirely by the operations of a few large concerns in London, and while 1908 did not witness the spectacular gyrations of 1907, it was nevertheless characterized by severe fluctuations up or down, accordingly as the bulls or bears had the upper hand. The domestic market is dependent upon the course which the metal takes on the London exchange, with such exceptions as are brought about by a scarcity or a corner in spot material, when premiums to a larger or smaller extent are exacted.

The market at the beginning of 1908 showed a firm undertone, and prices were quoted at about 26½@27c. for spot delivery. Spot material remained scarce throughout January and a premium of ½@¾c. per lb. was paid for the same. During the first half of February, under the lead of the London market, prices advanced to 29¼c., then declined to 28c., but rallied to 29c., and remained on about that level for the remainder of the month. The announcement by the Dutch Government at the beginning of March of a reduction in the sale of Banka tin during 1907 led to excited speculation in the London market, where prices advanced considerably. Quotations at that time were established at 30½c. in the domestic market. March showed still further advances, and at the beginning of April quotations were about 31¼c. Until the middle of May prices remained on this level, when a bear attack in the London market brought about a severe decline. At the end of May, spot tin was quoted at 28¼c. in this market.

In July the low point of the year was reached, and sales took place at 27¼c. The better feeling prevalent in the copper market toward the end of July and the beginning of August had a sympathetic effect

on tin, and prices advanced to 31c. for spot material. August and September witnessed again a gradual decline in prices brought about by an entire absence of interest among consumers. Quotations were 29½c. at the end of the latter month. In October, tin remained fairly steady at around 29½c.; it advanced during the month of November to 30½c. per pound; but declined again in December, owing to its unfavorable statistical position which became public at the beginning of the month, to 29c., and closed at 29 cents.

AVERAGE MONTHLY PRICES OF TIN PER POUND IN NEW YORK.

Year.	Jan.	Feb.	Mar.	Apr.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.
1896	13.02	13.44	13.30	13.34	13.54	13.59	13.63	13.49	13.15	12.94	13.09	12.96	13.29
1897	13.44	13.59	13.43	13.34	13.44	13.77	13.89	13.80	13.98	13.88	13.79	13.71	13.67
1898	13.87	14.08	14.38	14.60	14.52	15.22	15.60	16.23	16.03	17.42	18.20	18.30	15.70
1899	22.48	24.20	23.82	24.98	25.76	25.85	29.63	31.53	32.74	31.99	28.51	25.88	25.12
1900	27.07	30.58	32.90	30.90	29.37	30.50	33.10	31.28	29.42	28.54	28.25	26.94	29.90
1901	26.51	26.68	26.03	25.93	27.12	28.60	27.85	26.78	25.31	26.62	26.67	24.36	26.74
1902	23.54	24.07	26.32	27.77	29.85	29.36	28.38	28.23	26.60	26.07	25.68	25.68	26.79
1903	28.23	29.43	30.15	29.81	29.51	28.34	27.68	28.29	26.77	25.92	25.42	27.41	28.09
1904	28.85	28.09	28.32	28.13	27.72	26.32	26.57	27 01	27.78	28.60	29.18	29.292	27.99
1905	29.325	29.262	29.523	30.525	30.049	30.329	31.760	32.866	32.095	32.481	33.443	35.835	31.358
1906	36.390	36.403	36.662	38.900	43.313	39.260	37.275	40.606	40.516	42.852	42.906	42.750	39.819
1907	41.548	42.102	41.313	40.938	43.149	42.120	41.091	37.667	36.689	32.620	30.833	27.925	38.166
1908	27.380	28.978	30.577	31.702	30.015	28.024	29.207	29.942	28.815	29.444	30.348	29.144	29.465

London.—The market opened in January at £122, rose to £125¾ for cash warrants on January 15, and after a decline to £122½ closed the month at £124¼. At the beginning of February there was a sharp rise, three months' warrants fetching £131 on Feb. 5. This was followed by a relapse to £126 for cash warrants on Feb. 18. After a recovery to £132 the month closed at £129½ for cash and £129 for three months'. March opened with a decline to £127½ for three months', cash being £129½. On March 4 it was announced that the Dutch Government had revoked its decision to augment the Banka sales by 1500 tons during 1908, so that the year's supply would be approximately the same as in 1907. This altered the tone of the market and prices rose to £136½ for cash and £132½ for three months'. The market was practically in control of the bull party during the remainder of the month, final quotations being £144¼ for cash and £143 for three months'.

April opened with more normal conditions and a recession in the price, but was quickly followed by another advance, followed by a recession and a series of fluctuations, the closing prices being £143 for cash and £142 for three months'. In May there was a recession under realizing sales until on the 13th cash stood at £133 and three months' at £132. From that point there was a recovery, followed by a sharp break to £126¼ for cash and £125½ for three months'. The final prices

for the month were £129¼ for cash and £128¼ for three months'. In June the market fluctuated between comparatively narrow limits, the closing prices being £125 for cash and £126⅛ for three months'.

In July prices rose steadily, touching £140 for three months' on the 23rd. The closing prices for the month were £138½ for cash and £139¾ for three months'. In August the tendency of the market was downward, cash warrants selling for £130¾ and three months' at £131½ toward the end of the month, the closing prices being £132⅜ and £133⅛ respectively. In September the fluctuations were comprised within narrow limits, the closing prices being £134 for cash and £135½ for three months'.

October was also a month of fluctuations, the general tendency being upward, prices at the end of the month being £136¼ for cash and £137¾ for three months'. In November there was a rise to £142½ for cash and £144 for three months', which was followed by a relapse. The month closed with a dull market at £135⅛ for cash and £136¾ for three months'. In December, persistent selling, chiefly by eastern holders, caused weakness from the outset. On Dec. 14, £130½ was accepted for cash warrants and £132¼ for three months'. The closing prices of the month were £131⅜ for cash and £133 for forward tin.

AVERAGE MONTHLY PRICE OF TIN IN LONDON. (*a*)
(In pounds sterling per ton of 2240 lb.)

Year.	Jan.	Feb.	Mar.	Apr.	May.	June.
	£ s d	£ s d	£ s d	£ s d	£ s d	£ s d
1897.....	60 .5 .1	61 .4 .3	59.18 .9	59.18 .1	60.17.10	61.16 .6
1898.....	63 .1 .7	63.15.11	65 .1 .0	65 .3 .0	66 .6 .0	68.15 .0
1899.....	99.16 .4	108.16 .3	107.16 .8	114 .1 .1	117 .9 .6	117.12 .0
1900.....	118 .9.11	137.18 .4	142 .0 .0	137.15 .0	135 .1 .8	139. 9 .3
1901.....	120 .9.10	122 .6.11	116.15 .6	116 .3 .0	123.13 .0	129.16.11
1902.....	106 .6 .5	114 .4 .9	115.10 .6	125.14 .2	134.13.10	129.12.10
1903.....	127.12 .6	133 .8 .1	137 .0 .6	136.19 .2	133.12 .0	127.11 .0
1904.....	130.10 .4	125.13 .6	126 .9 .8	127 .5 .1	125 .7 .2	119.11 .1
1905.....	131 .5.11	131 .3 .6	134.17 .2	140.11 .8	136.11 .8	138 .3 .6
1906.....	164.11.10	166 .0.10	166. 1 .2	176.14 .5	192 .6 .4	178 .0 .7
1907.....	190 .4 .0	191.18 .9	188.17 .6	187 .1 .2	191 .1.10	187.10.11
1908(*b*).....	121.16 .6	129 .0 .0	137.16 .0	144.10 .0	135 .0 .0	127.12 .6

Year.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
	£ s d	£ s d	£ s d	£ s d	£ s d	£ s d	£ s d
1897.....	62 .5 .7	61.10 .1	61.12 .8	62.11 .9	62.11 .9	62.10 .0	61. 8 .0
1898.....	71 .4 .2	73.10 .1	73.15 .7	78.17.10	82 .8 .6	82.10 .7	71 .4 .1
1899.....	132.13 .1	142 .1 .4	146 .7 .2	144.10 .2	129.16 .0	113. 0 .7	122 .8 .7
1900.....	142.16.10	140.19 .1	132.13 .9	130.14 .3	127 .3 .8	119.14 .9	133.11 .6
1901.....	127.19 .9	116 .1 .7	114.10 .6	113 .1 .5	114 .0 .7	108.17.10	118.12 .8
1902.....	127 .3 .2	126.10 .0	121.10 .7	117.11 .3	115 .2 .3	115.13 .5	120.14 .5
1903.....	125 .1 .7	127.16.10	120 .9 .6	115.17 .1	116.13 .9	125.15 .0	127 .6 .5
1904.....	119.18 .6	122 .5 .9	126 .7 .7	130.11 .6	133 .0 .5	133.15 .6	126.14 .8
1905.....	144 .6 .8	150 .5 .6	146.11 .9	148 .3 .6	152 .5 .3	162.14 .3	143 .1 .8
1906.....	170.12 .5	180.19.11	184.15 .3	195.15.11	195.15.10	195.19 .9	180.12.11
1907.....	188 .0 .2	170 .5 .9	166 .6 .6	146 .7 .7	138 .8 .8	125.10 .4	172.12 .9
1908(*b*).....	131 .0 .4	134.14.12	131.18 .9	134 .0 .0	138.12 .6	138.12 .6	133.14 .7

(*a*) As reported by Metallgesellschaft, Frankfurt am Main. (*b*) As reported by Sargant & Sons, London.

THE METALLURGY OF TIN.

Tin Smelting in the Electric Furnace.—Experiments have been made by F. Mattonet, described in *La Metallurgie,* 1908. His experiments were made on a very small scale and with a very impure tin ore. He found an advantage of this kind of smelting to be the production of a slag containing but little tin. On the other hand, loss by volatilization was high, rising to as much as 15 per cent. Nevertheless the total extraction was equal to that of the ordinary process of smelting. The metal produced was very impure. It is to be remarked, however, that the experiments were made with a very impure ore.

Recovery of Tin from Tin Plate Scrap.—According to Dr. K. Goldschmidt, the electrolytic process is attended with numerous difficulties, owing on the one hand to the rapidity with which the caustic soda in the bath becomes inactive through absorbing carbon dioxide from the air, and on the other hand, to the electrochemical troubles arising from the necessity for warming the bath, and to the loose manner in which the scraps must be connected to the anodes, in order that the electrolyte may have free passage through the scrap. Hence, it is only by great care in working that the iron can be freed from tin to a sufficient extent to fit it for use in the open hearth steel furnace. Under the most favorable conditions, 0.05 to 0.1 per cent. of tin is unrecovered; indeed, the amount is usually twice as large, owing to the thicker deposits at the edges of the plates being only imperfectly dissolved; 0.3 to 0.5 per cent. of unrecovered tin is by no means infrequent.

In the Goldschmidt chlorine process, every trace of moisture, together with all organic substances, such as straw, paper, varnish, and anything likely to absorb water, must be removed. The dried scrap is pressed into bundles, which are packed into baskets and lowered into a large upright cylinder. The cylinder having been closed, chlorine gas is introduced. Provision is made for cooling the cylinder, the heat liberated by the reaction of the chlorine on the tin being considerable. The chlorine is forced in under pressure in order to enable it to penetrate the interior of the bundles of scrap; otherwise, the central portion would be but imperfectly detinned. The condensation of the gas on attacking the tin causes a reduction of pressure, and so long as this is apparent, the process is incomplete. Persistence of the pressure at a constant level indicates termination of the reaction. At that stage the residual chlorine and tin chloride in the cylinder are expelled, and the bundles of scrap iron are washed, after which they are ready to be transferred to the open hearth furnace.

G. O. Seward and F. von Kügelgen have also described a process, assigned to the Tin Products Company, for detinning scrap with chlorine gas. (U. S. patent 915,029, March 9, 1909.)

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Extraction Process.—Improvements in and Relating to the extraction of Tin from Ores, Slag or Refuse. Hermann Mühlinghaus, Wiesbaden, Germany. (Brit. No. 25,460 of 1907.)

Recovery of Metallic Tin.—Cyclic Process for Recovering Metallic Tin. Franz von Kugelgen, Holcombs Rock, Va., and George O. Seward, New York, N. Y. (U. S. No. 883,139 of 1908.)

Slag Treatment.—Process for the Treatment of Slag from Tin-Smelting Furnaces. George T. Holloway, London, England. (U. S. No. 891,477 of 1907.)

Tin Chloride.—Improved Manufacture of Tin Chloride or the Like. Heinrich Brandenburg, Kempen-a-Rhein, Germany. (Brit. No. 1461 of 1908.)

Treatment Process.—Improvements in the Process of Treating Tin Ores or Tin Slag. Communicated from Hermann Mühlinghaus, Wiesbaden, Germany. (Brit. No. 15,039 of 1907.)

TITANIUM.

Titanium ore in the form of rutile is produced by the American Rutile Company, from its mines at Roseland, near Arrington, Nelson county, Va. The deposits at that place occur in the form of great pegmatitic dikes, consisting chiefly of orthoclase and albite feldspars and blue quartz, together with rutile in a widely variable proportion. At places there is much hornblende; more rarely the rock contains apatite. At certain points the rutile has segregated in considerable quantities and forms from 5 to 35 per cent. of the mass. The American Rutile Company works areas containing such segregations. At a place five or six miles distant the General Electric Company has been making developments upon a similar dike, which attains a width of 5 ft. and appears to carry as high as 50 per cent. of rutile.

Rutile has been used for a long time in the dental industry (on a small scale) for the manufacture of artificial teeth. It is also employed in the preparation of a glaze for earthenware. Besides these, there have been several other more or less trifling uses. The recent increase in the demand for this mineral has been due to the electrical industry, which employs it as a component of the electrodes for arc lamps. However, the demand for the mineral is very irregular. There was no production of this mineral in 1908, but there has been in the early part of 1909. Besides that which has been used by the domestic electric companies, a considerable exportation from Virginia has been made in 1909. The value of rutile, about 97 per cent. pure, has varied from $100 to $160 per 2000 lb. For use in the electric industry, the mineral must be quite free from most foreign substances. Anything that slags easily is bad for an arc lamp.

Outside of the United States, rutile is produced at Kragerö in southern Norway, on the coast northeast of Kristiansand, whence 50 or 60 tons of nearly pure rutile is obtained annually, besides a certain amount of lower grade mineral. A little rutile is occasionally saved as a by-product at Para Wira and at Talunga, in South Australia. In 1908 Francis J. Spence, of Adelaide, developed a lode mine near Mount Crawford, South Australia. Rutile has also been observed in Queensland.

Experiments have continued for the use of titaniferous iron ore in the manufacture of ferrotitanium. A. J. Rossi, of the Titanium Alloy Mfg. Company, who has been investigating titanium and its alloys for many years, has recently patented the use of a small quantity of titanium in the production of sound copper castings (U. S. patent 905,232, Dec. 1, 1908). The titanium is introduced into the molten copper by means of an alloy of copper and titanium containing from 8 to 10 per cent. of titanium. From 1 to 2 per cent. of this alloy is used. There is introduced, therefore, from 0.10 to 0.20 per cent. of metallic titanium. The inventor states that when this quantity of titanium is used there is found no trace of it in the finished copper casting. Its function, therefore, is to reduce the oxides and combine with the gases in the molten metal. Oxide of titanium is thus formed which floats to the top of the copper.

The Occurrence and Uses of Titanium.

By Charles Baskerville.

Titanium is almost invariably present in igneous rocks and in the sedimentary material derived from them. It occurs in soils and clays, and 784 out of 800 igneous rocks analyzed in the laboratory of the United States Geological Survey contained titanium. Extensive deposits of titanium ores are also known. The important titanium minerals are rutile, brookite, octahedrite, perofskite, ilmenite and titanite.

The addition of titanium in minute amounts to iron and steel increases the tensile strength greatly; ferro-titanium is useful in cases where a metal of considerable density and hardness is required, as in the manufacture of car wheels. Titanium steel is employed for cutting tools. The addition of titanium to steel containing 0.8 to 1.2 per cent. of carbon scarcely increases the hardness, but greatly raises the elasticity. On account of the beautiful light effect it produces during combustion, titanium has been used in pyrotechnics. It is also used in the manufacture of filaments for incandescent electric lamps, and for incandescent media for gas lights. With the development of the electric furnace treatment there will be a market for titaniferous iron ores.

Rutile, or nigrine, occurs in reddish brown to nearly black prismatic crystals, which are often acicular in form and as such included in other minerals. It is found occasionally massive, especially when black and ferriferous. Rutile occurs as a pyrogenic mineral in eruptive rocks, but is more common to gneiss, mica schist, and the phyllites. It also is found as a secondary mineral derived from ilmenite and titanite. Rutile is a brittle mineral, having a hardness of 6-6.5, and a specific gravity

of 4.18-4.25. Its luster is adamantine to almost metallic, and the cleavage is prismatic and pyramidal. The streak is pale brown. It is found in Maine, New Hampshire, Massachusetts, Connecticut, New York, Pennsylvania, New Jersey, Virginia, North Carolina, Georgia, Arkansas, Colorado, and in the Black Hills, Dakota. The most prominent of these localities are Nelson county, Va.; Lincoln county, Ga.; Habersham county, Ga.; Warwick, N. Y.; Warren, Me., and Magnet Cove, Arkansas.

Rutile is infusible before the blowpipe. It dissolves very slowly in salt of phosphorus in the oxidizing flame to a yellowish bead, which assumes a violet color in the reducing flame on cooling. Ferriferous varieties give a brownish yellow or red bead in the reducing flame, so that a violet color is only obtained after treating the bead with metallic tin on charcoal. It is insoluble in acids. However, it dissolves by fusion with an alkaline carbonate; the solution, after the addition of an excess of acid and tin-foil, yields a violet color on boiling.

Rutile is of a more decided red color and of lower specific gravity than cassiterite; and may readily be discriminated from garnet, pyroxene, vesuvianite, and tourmaline by its luster, weight and infusibility.

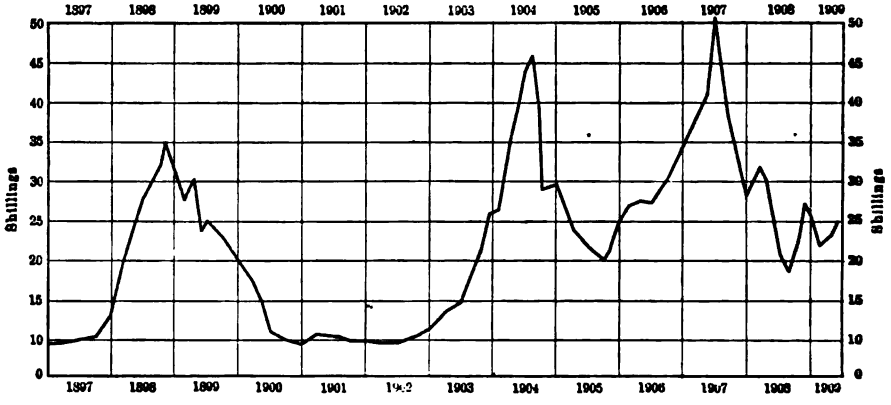
Ilmenite, menaccanite, or titanic iron-ore is an iron-black mineral, which is found usually massive, in thin plates, in embedded grains, or loose as sand. It occurs in Orange county, N. Y.; Massachusetts; Litchfield, Conn.; Troy, Vt., and in California. Extensive beds of ilmenite occur in Quebec, Canada. Its hardness is 5-6, the specific gravity is 4.5-5, and the luster is submetallic. The fracture is conchoidal. The streak is black to brownish red. Ilmenite slightly influences the magnetic needle.

Ilmenite is infusible in the oxidizing flame, but is rounded on the edges in the reducing flame. It reacts for iron with borax and salt of phosphorus in the oxidizing flame, and yields a brownish-red bead in the reducing flame with the latter salt; this changes to a violet-red color when treated with tin on charcoal. The powdered mineral is soluble in hydrochloric acid to a yellow solution, which assumes a blue or violet color when boiled with tin. It is decomposed on fusion with sodium or potassium disulphates.

Ilmenite is distinguished from magnetite and hematite by the titanium reactions described above. It is used in the preparation of a lining for puddling furnaces, but is not very desirable as an iron ore on account of the difficulty in reducing it.

TUNGSTEN.

The production of tungsten concentrate in the United States in 1908 was 497 tons of 2000 lb., valued at $126,281, which was approximately only one-third that of 1907, and was the smallest output reported since 1903. This great falling off in production is traceable primarily to the depression in the steel industry during 1907 and 1908, which caused the closing down of many of the Colorado tungsten mines. As in previous years, the bulk of the production in 1908 came from Colorado; South Dakota and California contributed less than 100 tons to the total. A



PRICES PER UNIT OF WO_3, 1897-1909 (STEINHART.

notable feature of the industry in Colorado was the consolidation of a number of properties by the most active interests in the tungsten industry. The accompanying table shows the production of tungsten concentrate in the United States for a period of years.

PRODUCTION OF TUNGSTEN CONCENTRATE IN THE UNITED STATES. (a)
(In tons of 2000 lb.)

Year.	Production.	Value.	Average per Ton.	Year.	Production.	Value.	Average per Ton.
1901.....	179	$27,720	$155	1905.....	834	$257,463	$308
1902.....	184	33,112	180	1906.....	1,096	443,150	404
1903.....	292	43,639	149	1907.....	1,468	715,031	487
1904.....	740	184,000	249	1908.....	497	126,281	254

(a) Statistics reported by the U. S. Geological Survey, except for 1905-08.

Prices.—Owing to over-production in 1907 and a lack of demand early in 1908, the price for tungsten concentrate suffered a large decrease in 1908. The average price per ton received during 1908 was $254, as compared with $487 in 1907. The lowness of the average for 1908 was due to the sale of some inferior ore which fetched only $26.50 per ton. The bulk of the output was sold at prices ranging from $280 to $310 per ton. During the latter part of 1908, when the industry showed increased activity, prices manifested a hardening tendency.

Tungsten Mining in the United States.

California.—In point of production, the Atolia region, in San Bernardino county, is the most important district in this State. The production in 1908, however, was small, owing to the closing down of the chief producers early in the year. The ore in this district is scheelite; it occurs in massive form in bunches and seams in the veins, which vary from stringers to about 4 ft. in width. The vein matter is quartz, altered country rock and calcite; the country rock is granitic. The veins strike easterly and westerly and dip from 40 to 80 deg. north. The Atolia Mining Company is the chief operator in this district. Wolframite has been found in the foothills of the Sierra Nevada mountains, about 12 miles north of Raymond, Madeira county. The ore occurs in bunches up to 2 or 3 lb. in weight, in a glassy quartz vein, from 4 to 16 in. in width.

Colorado. (By R. D. George.)—The financial stress which developed in the latter part of 1907 paralyzed the tungsten-mining industry early in 1908, and for several months the district was very quiet. Several of the larger producers in Boulder county closed down completely, a number of the smaller companies went out of business, and consolidations still further reduced the number of producers. It was not until the latter half of 1908 that the camp began to come to itself again. Since that time activity has increased, and 1909 promises a large measure of prosperity.

One of the notable events of the year was the organization of the Primos Mining Company, which bought the tungsten mines of the Cardinal company, and those of the Stein & Boericke company. The Cardinal company kept a force of men on development work, and has opened in the Conger, one of the largest, if not the largest, body of tungsten ore yet found in Boulder county. Promising finds have been made in other mines formerly belonging to this company, but now held by the Primos Mining Company.

The principal producers in 1908 were: The Primos Mining Company, the Philipp Bauer Company and the Wolf Tongue Company. Probably

75 small producers contributed to the total output largely, however, as a result of the development work undertaken. Most of their ore was treated at the Wolf Tongue mill at Nederland. A number of good finds were made in the southern part of the district, particularly near Manchester lake. High-grade ore in promising quantities has been opened in the Boulder Falls part of the district, and this area is likely to make a good record when mining activity is fully restored. Some development has been done in the vicinity of Ward, not far from Gold lake, but no shipments have been made.

As to the future of the district, the producing area is today larger than ever and the deepest workings show no falling off either in quantity or quality of ore. With satisfactory prices for the product, the camp would show an increased production. The production for 1908 was as follows: Ore mined and concentrated, 3450 tons; high-grade ore ranging above 25 per cent. of tungstic oxide, 180 tons; concentrates, 407 tons; total value, $167,257.

Idaho.—No production was reported from the two known tungsten deposits in this State, namely, those of the Golden Chest and Golden Winnie mines at Murray. The ore of this district is scheelite.

Montana.—There were no new developments at the Birdie mine, in the hills east of Butte, during 1908. The ore here is hübnerite. Although pockets of scheelite have been encountered at various points in Park county, there was no activity in this section during 1908. The only tungsten shipment ever made from this district was from the gold mine of the Kimberly-Montana company at Jardine.

Tungsten in Foreign Countries.

By O. J. Steinhart.

The world's production of tungsten concentrate (60 per cent. WO_3) in 1908 showed a slight decrease as compared with the output of 6000 short tons in 1907. Moderate stocks are said to exist in Europe (April, 1909) but sellers are not anxious to dispose of their product at 25s. per unit, anticipating higher prices. The setback in prices during 1908 was partly due to the increased output in the United States and to some extent to the depression of the general and engineering trades in the United States, which country has been a ready consumer of large quantities of high speed steels, into which the tungsten ultimately finds its way, after being reduced to metallic state or to the form of a ferro-alloy.

The following figures show the approximate high and low price in Europe in shillings per unit (22.4 lb.) of WO_3 per long ton, for a period of years. 1900, 20s.-10s.; 1901, 11s.-9.75s.; 1902, 12s.-10s.; 1903, 27s.-12s.; 1904, 45.5s.-27s.; 1905, 29s.-20s.; 1906, 34.5s.-25s.; 1907, 51s.-28s.

During the first few months of 1908 the price advanced from 28s. to 31.5s. then rapidly declined to 19s. shortly after mid-year. Quotations then quickly rose to around 27s. at which point they closed the year.

Australia.—In Queensland a great number of small easily mined surface deposits of wolframite were worked in 1905. Wolfram Camp in the Herberton-Chillagoe district and also Kooboora and Bamford, seem to have been the chief producing localities. Since 1905 the production has fallen off. In New South Wales both scheelite and wolframite, in about equal quantities, were mined in 1908. Hillgrove is the center of the district. The Northern Territory also is producing tungsten-bearing ores in increasing quantities, near Pine creek and 200 miles from Palmerston. The mineral occurs mixed with copper ores.

England.—The production in 1908 was 229 long tons against 250 long tons in 1907. The chief producers are the following mines in Cornwall: South Crofty, East Pool & Agar, Kingston & Clitters United, Carn Brea, and a few smaller concerns. A little mixed tin-wolfram has come from the Bodmin Moor district where the ancient workers have left alluvial ground behind, mixed with tin. For the present this production is insignificant, although the deposits, if worked by up-to-date dredging methods, would yield a great deal of tin and wolfram.

Malaya.—The States of Perak, Selangor, and Negri Sembilan have produced some wolfram and it is stated that many of the refuse heaps left behind by the Chinese contain considerable quantities of low-grade mixed tin-wolfram ores. One or two electro-magnetic plants are installed in the country.

Portugal.—Considerable quantities of high-class ores have come from this country. The most important mine, the Panasqueira in the Castello Branco district is said to have produced about 300 tons in 1908 at a cost of about £30 to £35 per ton. Water power is being installed which may cheapen production. Other mines are to be found at Braga (Rivaes), Aguas Ferrens and Mina Rebordosa (Porto), Borralhas (Villa Real), Alvarengo (Areiro), Laborim (Villa Nova de Velha), Carvalhal de Eastanlo e Bodiosa (Vizen), S. Margarida (Proenca a Velha), Quarta-Feira (Sabugal), Pinhel (Guarda), Serra das Pedras de Valle das Mocas (Villa Nova da Paira), Fonte da Batalha, Valle de Milho (Braganea).

South Africa.—During 1907, 211 tons of wolfram and scheelite were produced, most of which came from Rhodesia. The output seems likely to increase.

Spain.—Most of the wolfram mines are situated in the northwestern part of Spain, in the province of Galicia. In some respects the deposits resemble those of Cornwall, although the mineral is of a much coarser structure and not quite so closely associated with tin. The most impor-

tant mines are those of Tyre and Sideon, Angelita, and San Finx near Santiago de Compostella. These mines produced about 250 tons of wolfram in 1908. There is also La Porpresa mine in the province of Cordoba, in the Las Cabezas mountains. Scheelite here occurs mixed with wolframite and the production is said to be about 120 tons per annum, analyzing over 70 per cent. WO_3.

Classification, Occurrence, Identification, and Properties of Tungsten Ores.

By O. J. Steinhart.

Ores of Tungsten.—The principal ores of commercial importance containing tungsten are wolframite, hübnerite, ferberite and scheelite. There are further a number of less common wolfram-bearing minerals such as stolzite, powellite, reinite, raspite, wolframine, or tungsten ocher, meymacite, cupro-tungstate, etc. Wolframite is essentially tungstate of iron and manganese containing theoretically up to 75 per cent. WO_3. Most of it, however, is marketed in the form of concentrate, carrying between 60 and 70 per cent. WO_3. It is generally dark brown to black in color, with sub-metallic luster and good cleavage; specific gravity, 7.2 to 7.5; hardness about 5; streak, yellow to reddish brown. Ores under 60 per cent. WO_3 are not in demand and must be sold at a somewhat reduced price. Wolframite from the northwestern part of Spain showed the following analysis: WO_3, 64.13 per cent.; SnO_2, 0.68; MnO, 6.42; FeO, 10.88; SiO_2, 7.71; Al_2O_3, 5.32; CaO, 1.21; MgO, 3.16; CrO, 0.38.

Ferberite, which contains but little manganese, is mined in considerable quantities in Boulder county, Colorado, and is a nearly pure tungstate of iron, carrying but little manganese. Typical concentrates[1] assayed as follows, the first figure being for material from the Clyde mine, Nederland, and the second for material from the Baker ranch, Nederland: WO_3, 61.15, 63.88 per cent.; SiO_2, 16.10, 6.45; FeO, 19.33, 20.44; Fe_2O_3, ——, 0.35; CaO, 0.38, 0.35; MnO, 0.51, 0.37; Al_2O_3, 2.49, 2.19; MgO, 0.39, 0.50.

Electromagnetically separated ore, originally associated with tin ore from mines in Cornwall,[2] showed the following range of composition in four samples: WO_3, 60.3 to 72.4 per cent; SnO_2, 0.66 to 8.75; insoluble, 2.20 to 4; Fe_2O_3, 2.4 to 9.4; FeO, 13.5 to 16.2; MnO, 5.2 to 5.4. Hübnerite, chiefly tungstate of manganese, from the Natalie mine, Silverton, Colo., contained[1] WO_3, 70.21 per cent.; SiO_2, 4.91; FeO, 2.03; CaO, 0.37; MnO, 21.72; Al_2O_3, 0.56.

[1] *MinWorld*, Feb. 6, p. 280, 1909.
[2] H. W. Hutchin, Cornish Consolidated Tin Mines.

The majority of these minerals occur in granite country, in quartz veins, showing more or less continuity, rarely going to considerable depth and very often in pockets. Occasionally, as for instance in Cornwall, they are found between the junction of granitic and schistose rocks (locally called killas). Considerable quantities of water-worn alluvial wolframite deposits are met with, as for instance, in Malaya, Cornwall and Queensland. These varieties are generally very pure although they are sometimes mixed with varying quantities of cassiterite, from which they can, however, be easily separated by electromagnetic treatment.

The need of the separation of tin and wolfram has been much felt by some of the smaller producers, who are unable to provide electromagnetic separation plants, the manipulation of which requires considerable experience and careful chemical supervision. Considerable quantities of such mixed ores are now being purchased by the Hingston & Clitters Mines, near Plymouth in Cornwall (56 Cannon street, London, E. C.). This concern either purchases mixed lots outright or treats them for a reasonable returning charge.

Scheelite, essentially tungstate of lime, containing from 71 to 78 per cent. WO_3, is produced in fairly large quantities but is not quite so much liked by manufacturers of tungsten and its alloy as wolframite. It generally sells at from 2 to 3s. per unit less than wolframite. In appearance scheelite varies from a glassy white to a dull yellow, and has a stony appearance, some specimens resembling coarse marble. It occasionally crystallizes in the tetragonal system. Its specific gravity is 5.6 to 6; it can be easily scratched with a knife as it is somewhat softer than limestone. Analyses of scheelite from various countries show the following range of composition: Tungsten trioxide, 69 to 78 per cent.; lime, 16 to 19; iron oxide, 0.5 to 2.2; manganese oxide, 0.18 to 0.77; phosphoric acid, 0.14 to 0.2; silica, 2.5 to 12.5. An interesting specimen of mixed cassiterite and scheelite of apparently alluvial origin, from the Malay States, was recently shown to me by B. Kitto, of London. The material consisted of about 70 per cent. scheelite and 30 per cent. tin oxide. If considerable quantities of this complex ore are forthcoming this will add another concentration problem.

Identification.—Other minerals, such as titaniferous iron, magnetite, tantalite, cassiterite and tourmaline, are often mistaken for wolframite and hübnerite; scheelite sometimes is mistaken for barytes and similar heavy minerals. A very simple and quick way to identify tungsten minerals is as follows: Boil the finely powdered mineral for 20 or 30 minutes in hydrochloric acid and after dilution with water, drop a piece of zinc, tin, or even iron, into the solution. The appearance of a blue

color, ultimately changing to claret and finally to brown, caused by the gradual reduction of WO_3 to its intermediate lower oxides, is a pretty certain indication of the presence of tungsten, although molybdenum behaves in a somewhat similar manner. Wolframite fuses easily before the blowpipe to a globule, which has a crystalline surface and is magnetic. With salt of phosphorus it gives a reddish yellow gloss while hot, which is paler on cooling; in the reducing flame this gloss becomes dark red and opaque.

Physical Properties.—The physical properties of tungsten depend on the manner of its preparation. Reduction by hydrogen at dull redness gives an amorphous steel gray powder of a fine luster. The reduction of tungstic acid by potassium by Wohler's method gives a more brilliant crystalline powder. By electrolyzing molten sodium tungstate, Zettnow obtained the metal in the state of a black amorphous powder. It is pyrophoric when prepared according to Feree's method by using mercury as a cathode, forming an amalgam and then distilling off the mercury.

Tungsten is difficultly fusible, its melting point having been recently given as 2800 to 3000 deg. C. It can be drilled and filed easily and is not magnetic. Its density depends on the method of preparation, varying from 17 to 18 for the pulverized varieties. Pure tungsten melted in the electric furnace has a density of 18.7. Heated to 1000 deg. C. and compressed under a pressure of 200 kg. per sq.cm. (200 atmospheres) gives a density as high as 18.92. Its specific heat is about 0.034 deg. C., and increases with the temperature. Its electrical resistance as determined by use is $86 \times 10^{-6} = 0.000086$ ohms per centimeter at 10 deg. C.

Chemical Properties.—Tungsten is attacked by fluorine at ordinary temperatures with intumescence, forming a volatile fluoride. Dry chlorine attacks it at about 300 deg. C., forming the hexachloride; moist chlorine gas forms the red oxychloride. Bromine acts on it only at low redness, forming the pentabromide; iodine attacks it only at a bright cherry red. Oxygen, either dry or moist, has only a feeble action at ordinary temperatures; however, the metal blackens little by little in the air. At redness tungsten is attacked by oxygen, forming tungstic acid; it burns likewise in air at bright redness, swelling up and forming a powdery mass of tungstic acid. Melted sulphur slowly transforms tungsten into sulphide. Nitrogen has no action at redness; the vapor of phosphorus attacks powdered tungsten at redness. In the electric furnace carbon, boron and silicon unite with tungsten to form carbides, borides and silicides.

Tungsten is not sensibly attacked by water at ordinary temperatures; it is attacked rapidly by vapor of water at redness. Hydrofluoric and

sulphuric acids attack it slowly; nitric acid and aqua regia transform it into tungstic acid. Carbon bisulphide and hydrogen sulphide attack tungsten at redness, forming the bisulphide. Several fused oxidizing agents, such as lead peroxide and potassium chlorate, attack the powdered metal with incandescence. Fused sodium carbonate and the alkalies dissolve it slowly. A mixture of carbonate and nitrate of potassium oxidizes it rapidly, forming potassium tungstate. Arsenide of tungsten heated with copper out of contact with air up to the melting point of copper is completely decomposed forming copper arsenide and metallic tungsten. Tungsten forms compounds with B, C, Si, and alloys with Al, Ni, Fe, Mn, Cr, and Mo.[1]

Mushet, whose name is so well known with regard to many inventions in connection with steel, in 1857 took out his first English patent for tungsten steel. At the same time several Continental inventors, including Jacob and Koeller and later F. Meyer, were at work in the same direction. Mushet made a more or less impure ferro-alloy of tungsten direct from the ore, which he added to the steel. Later, in 1866, L. Biermann in Hanover started the manufacture of a 25 per cent. alloy in crucibles, which met the requirements of that period; later he manufactured small quantities of an alloy containing 78.76 W, 15.94 Fe, 5.03 per cent. C., an alloy totally unsuitable for the present day use for high percentage tungsten steels, which contain from 10 to 20 per cent. of tungsten and about 0.6 to 0.8 per cent. of carbon, besides several per cent. of chronium and some manganese.

The manufacture of metallic tungsten powder[2] 96/98, as now employed by the steel maker, is in the hands of about half a dozen German makers and seems to have first been perfected at some of the Hanoverian works. In England the Tungsten and Rare Metals Company, Ltd., began the manufacture of metals of the following composition: W, 97.02 per cent.; Si, 0.72; C, 0.32; Al, 0.47; Fe, 0.61; Mg, 0.32; Mn, 0.16; S, nil; P, nil; O, etc., 0.33; total, 100. Also W, 98.63 per cent.; Si, 0.32; Fe, 0.59; Al, 0.21; C, 0.12; Mg, Ca and O, 0.13; total, 100. This firm stopped manufacturing metal in 1907, but is now erecting works near Liverpool.

Briefly, the production of the metal is carried out as follows: The ore is first ground to a fine powder and heated in a reverberatory furnace with the necessary quantity of sodium carbonate in order to form tungstate of soda. The melt, which also contains the FeO, MnO, Al_2O_3, SiO_2 and any SnO_2 present, is again crushed, leached with boiling water, pressed, and the above-mentioned oxides left behind in the filter press in which the last operation is performed. If the furnacing has been

[1] O. J. Steinhart. "Metals and Ferro-alloys for Steel," Trans., I. M. M., Jan., 1906.
[2] O. J. Steinhart, loc. cit.

carried on correctly, not a trace of tin need go into the solution; in fact, tin is often recovered from the residues. From the sodium tungstate the yellow WO_3 is thrown down by means of hydrochloric acid. The tungstic acid is dried and reduced in crucibles to metallic tungsten, which should not contain more than 0.5 per cent. of carbon as sent to the steel makers. The manufacture is a lengthy and complicated one and requires careful and highly-trained chemical supervision.

Without going fully into the cost of the manufacture of tungsten by the process outlined above, it may be taken to be between £80 and £100 per ton of metal made, including the manufacturer's profit. The yield is by no means a theoretical one, as it requires from 140 to 145 units of WO_3, as compared with a theoretical consumption of 120 units, to produce 100 parts of 96 to 98 per cent. tungsten. If, therefore, we take the present price of ore to be 26s. per unit, we have the following: (26s. $\times$140)+£90=£272 per ton.

Besides tungsten powder, ferrotungsten is now produced regularly in France by Girod and others in the electrical furnace and for some purposes may be used with advantage, although it is not materially cheaper as far as its tungsten contents are concerned than tungsten powder. Ferrotungsten is produced in three grades with varying carbon contents as follows: (1) tungsten, 85.15 per cent.; iron, 14.12; carbon, 0.45; silicon, 0.13; manganese, 0.085; phosphorus, 0.018; sulphur, 0.021. (2) Tungsten, 79.48 per cent.; iron, 18.60; carbon, 1.49; silicon, 0.16; manganese, 0.21; aluminum, 0.001; phosphorus, 0.017; sulphur, 0.016. (3) Tungsten, 71.80 per cent.; iron, 24.35; carbon, 2.58; silicon, 0.36; manganese, 0.78; aluminum, 0.07; copper, 0.008; phosphorus, 0.008; sulphur, 0.02. The manufacturers of tungsten steels seem to prefer the powder, as it is easily added to the steel and can be relied upon as to composition. The annual consumption in Sheffield alone is about 1000 to 1500 tons, practically all of which is used for high speed rapid cutting tool steels.

Besides tool steel tungsten is used for few other purposes. Permanent magnets for telephones are made of tungsten steel and are said to answer well. Within the last two or three years a great number of patents have been taken out for the employment of metals with high fusion points for the manufacture of metallic filaments for electric lamps. The so-called Osram lamp which has a practically pure tungsten filament is one of the best known examples. According to manufacturers, the superiority of tungsten over other metals such as osmium, molybdenum, tantalum, vanadium, etc., has now been ascertained by research. A 30 c.p. 100-volt Osram lamp has a filament of 0.045 mm., and a length of 20 in. A 16 c.p. 100-volt lamp has a 0.03 mm. filament 18 in. long.

The essential feature of the patented process for producing these tungsten filaments consists of mixing the powdered metal into the form of paste with an organic binding agent, such as for instance, gum arabic, which of course is rich in carbon; and squirting this paste into a thread. The filaments afterward go through various processes by means of which all carbon is said to be removed, leaving the metal absolutely pure. These lamps consume only about one-third the amount of current of the ordinary carbon filament of the same candle power. From the figures given above it will be readily understood that the consumption of wolfram ores or metallic tungsten for lamps is an insignificant one and all the rumors circulated as to largely increased consumption of the mineral some time back were utterly unwarranted. One ton of 70 per cent. wolframite will suffice to make 18 million lamps.

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VANADIUM.

This substance was discussed rather fully in THE MINERAL INDUSTRY, Vol. XVI. In 1907 vanadium oxide, from which vanadate of iron is made, was produced by the Vanadium Alloys Company at Newmire, Colorado, and a small amount of sodium vanadate was produced by the Dolores Refining Company at Cedar, Colorado. The United States Vanadium Company, of Telluride, produced some roscoelite, which was sold to the Vanadium Alloys Company. The total quantity of ore produced in 1907 was 1042 tons, averaging a little over 2 per cent. vanadium. The production in 1908 was about 1500 tons, yielding 64,800 lb. of vanadium. Operations were continued in 1908 by the Dolores Refining Company, which has introduced the Engle-Haynes process of extraction by an alkaline carbonate as the solvent, and precipitating the uranium and vanadium therefrom successively by alkaline and alkaline earth hydrates respectively. On Rock creek, Colo., according to the State Commissioner of Mines, a number of important discoveries have been made.

The most important source of vanadium appears to be the mines in Peru, where a vanadium-bearing asphaltite occurs. These deposits were described by D. Foster Hewett in a paper read before the American Institute of Mining Engineers, February, 1909. From June, 1906, to January, 1909, these mines in Peru produced and shipped to the United States 1800 tons of oxidized ore, containing about 20 per cent. vanadic oxide, together with the product from roasting 400 tons of sulphide ore. This ore is treated by the American Vanadium Company of Pittsburg, Penn., which has works at Bridgeville, Allegheny county, Penn. This company owns the mines in Peru. According to James J. Flannery, its president, the vanadium content and the extent of vanadiferous deposits in the United States are of such nature as to preclude the possibility (in so far as such deposits have been discovered) of providing to the steel and iron producers of this country sufficient vanadium alloy to meet the existent demand, or to provide for the growing demands of the future.

The American Vanadium Company was instrumental in exploiting vanadium for use in iron and steel, copper, brass, bronze, and aluminum, and its use in these metals is being rapidly extended, which growth will be accentuated if consumers are encouraged to believe there will

be either no advance in the present market price for the vanadium alloy or a reduction thereof; prior to the development of its ore deposit and manufacturing process about 90 per cent. of the vanadium alloys consumed in the United States were imported from Europe.

Ferrovanadium is used in the production of practically all classes of wrought steel and steel castings and in cast iron, and by all processes of steel and cast-iron manufacture, the common purposes for which such vanadium steels and vanadium cast irons are employed being—

(*a*) In wrought (or worked) steel:

(1) General forging steel, for locomotive piston rods, and piston rods for other purposes, connecting rods, crankshafts, axles, side and main rods, gears, hammer rods, bolts, gun barrels, tires, springs, high-tensile structural material, etc. The price charged therefor by steel producers varies from 4½c. per lb. to as high as 13c. per lb. for specially finished vanadium crucible steel, on which more or less rough forging work has been done.

(2) Cutter and punch and die steel, for rotary rock cutters, punches and dies, saws, edged tools, etc.; the base price charged by producers ranges from 10 to 13c. per pound.

(3) High speed tool steel, containing from 15 to 25 per cent. tungsten, from 3 to 5 per cent. chromium, and about 0.3 per cent. vanadium, used for cutters on lathes, planers, etc. The price charged by producers ranges from 55c. per lb. to 65c. per lb.; the use of vanadium in such steel consumes a comparatively unimportant quantity of the ferrovanadium produced.

(4) "Governmental" steel, for use in protective deck plate, armor plate, gun shields, torpedo tubes, projectiles, defense guns, and artillery, in which vanadium—owing to four properties it imparts, namely, scavenging, toughening, strengthening, and prevention of erosion—is either now being used, or will be used in the immediate future, in considerable quantities.

(*b*) In steel castings, for locomotive frames, transmission bars, drawheads, crossheads, and the multifarious other uses for which steel castings are employed, the price charged by producers is from 2 to 3c. per lb. in excess of that charged by them for steel castings not containing vanadium.

(*c*) In cast iron for car wheels, piston-rod bushings, cylinder-plow moldboards, water-chilled rolls for steel mills, etc.

The addition of vanadium to steel greatly increases the static strength of the steel and imparts the highest attainable resistance to molecular disintegration arising from shock, impact, and fatigue, and as vanadium

is likewise efficacious in other metals, the lower the price charged therefor the more rapidly will its use be extended.

In the early part of 1908 the market price for metallic vanadium in the alloy, free from carbon, was $5 per lb. That is to say, ferro vanadium fetched $5 per lb. for its vanadium content. When the alloy contains carbon, 1 to 4 per cent. perhaps, the price is much cheaper.

Occurrence of Vanadium Near Telluride, Colorado.

By Edward R. Zalinski.

It is the purpose of this article to describe the occurrence of vanadium on Big Bear creek, about 14 miles west of Telluride, Colo., and about two miles south of Wilson station, or Newmire, on the Rio Grande Southern railroad. A plant for the reduction of vanadium ores is situated at Newmire, and the supply is obtained from the deposits on Big Bear creek. Brief mention of the mill of the Vanadium Alloys Company has appeared at various times, but as far as I am aware nothing has been published regarding the ore occurrence.

The deposits occur in a bed of white sandstone lying beneath a bed of black limestone and above thick, red sandstone beds. As far as developed it is only workable for vanadium at certain points, though both vanadium and uranium minerals are found widely distributed in San Miguel and Montrose counties, and the occurrence extends into southeastern Utah. The vanadium exists as an impregnation of the sandstone. The supply for the Vanadium Alloys Company is at present being mined about 1.5 miles south of Newmire on Big Bear creek. The sandstone is being worked on the east side of the cañon by open cuts and short adits. The bed of sandstone mined here is up to 9 ft. thick. It is impregnated with an olive-green mineral which is probably the vanadium mica, roscoelite. The zone of vanadium impregnation is parallel to the bedding, and varies in thickness and intensity. The sandstone changes from light buff in color to dark olive green where richest in roscoelite. Occasional lighter streaks and irregular bands of sandstone are inclosed in the richest portions. The ore is said to average from 3 to 5 per cent. vanadium oxide, though portions of the bed run higher.

In the Placerville deposits some carnotite occurs in minute yellow specks in the sandstone near the bottom of the vanadiferous zone, but here as on Bear creek the chief vanadium mineral is roscoelite. On La Sal and Rock creeks, in the western portions of Montrose county, carnotite is the most abundant mineral. Hand specimens of the rock from Bear creek show a medium fine-grained sandstone of dull olive-green color containing small silvery-white specks of mica and occa-

sional spots and patches stained with limonite. Rock sections of the Placerville sandstone examined by Ransome showed the quartz grains often to be surrounded or cemented together by roscoelite. This is grass green in thin section and resembles chlorite; it sometimes shows an indistinct foliated or fibrous structure. Both the Bear creek and Placerville occurrences are in the lower bed of the La Plata sandstone directly beneath the bed of limestone or dark calcareous shale.

Treatment of the Ore.—The vanadiferous sandstone milled at the plant of the Vanadium Alloys Company at Newmire carries from 2 to 6 per cent. of vanadium oxides, mostly as V_2O_3, with a small percentage of V_2O_5. The treatment consists of crushing and roasting the ore with salt, leaching with water and precipitating with ferrous sulphate, the final product being a ferric vanadate which contains from 40 to 70 per cent. V_3O_5. The present capacity of the mill is 12 tons a day. Roasting with salt forms sodium vanadate and sets free chlorine. Sodium vanadate is easily soluble in water; the roasted ore, after cooling, is put into tanks with water and agitated by blowing air through it. It is possible that some vanadium chlorides are formed during the roasting; there is little or no vanadium left in the roasted and leached residue, so that if vanadium chlorides or oxychlorides are formed, they are undoubtedly decomposed and leached out. The solution containing sodium vanadate, possibly also some vanadium chloride, is pumped into other tanks and the vanadium precipitated as ferric vanadate by the addition of ferrous sulphate. The precipitate is collected on press filters and taken to the drying room where it is dried slowly to a lumpy powder of dark olive-green color. The mill has been in operation since 1906. A very small amount of uranium occurs with the ore; this is probably due to fine particles of admixed carnotite, which, however, are not separately visible.

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ZINC.

By W. R. Ingalls.

The production of spelter in 1908 is given in Table I, which is based on reports from all of the smelters. There was no addition to smelting capacity in 1908, excluding the new plant at Danville, Ill., and the two new works at Bartlesville, Okla., which went into operation just before the end of 1907 and were completed early in 1908. Otherwise no smelter added so much as a furnace. Indeed at least one plant begun in 1907 was abandoned. Several small works remained idle throughout 1908. Others were idle for longer or shorter periods. No one operated at full capacity. So far as I am aware no plans for new plants were prepared during the year.

All of the smelters but three reported the stock of spelter at their works on Dec. 31, including that held for speculators. The total of their reports is 19,848 tons. Allowing for the companies that did not report I estimate the stock in first hands on Dec. 31 to have been 25,000 tons. The delivery into consumption is computed in Table VII. In considering the figures in Table VII it is important to make a distinction between "deliveries" and "consumption." The zinc industry joins hands with iron and copper, which are represented by the galvanizers on one hand and the brass-makers on the other. The consumption of iron and copper in 1908 was not more than 70 per cent. of that in 1907.

Every smelter in the United States reported his receipts of ore, enumerated according to State and country of origin. The totals are given in Table X.

The bulk of the production in the United States continues to be derived from the Joplin district. The fact that the resources of that district have been seriously diminished is of determining influence in the spelter industry. If it had not been for the appearance of the Rocky Mountain zinc ore, the rejuvenation of mining in Wisconsin, and the importation of Mexican ore during the last five years, there would undoubtedly have been a spelter famine. In spite of the new supplies it has been necessary to mine a very large quantity of ore at Joplin that yields only 2½ to 3 per cent. of concentrated mineral, which could be done only at an advance in price for the ore and consequently for spelter also. This condition is becoming more and more emphasized.

There is not yet in sight any supply of cheap ore that may oust the Joplin district from its present position of premier zinc producing district. Consequently continuance of a high level of price for spelter must be expected, because it will be necessary to enable the Joplin sheet ground mines to continue to operate. The price might rise inordinately high under the demand for increasing consumption, but it will be restrained by supplies of ore from other districts. However, the other districts, except perhaps Wisconsin, will be unable to furnish cheap ore; because in spite of their advantage in a higher grade of crude ore, their cost of mining and milling per ton of crude ore is high (in most cases inevitably high), and their concentrated product is subject to high freight charges, owing to remoteness from the smelting centers. The experience of 1907-08 has shown that a price of less than 5c. for spelter tends to check production from these mines.[1]

Yet it would be unjustified to say that this condition will continue indefinitely, inasmuch as new zinc-producing districts may be discovered. None that can be pronounced as promising a substitute for Joplin is now in sight, but geologists consider that similar deposits may reasonably be expected along the skirts of the Ozark uplift, and the development of important mines at Miami, Okla., in 1908, lends color to this belief. Dr. Keyes, formerly State geologist of Missouri, has recently expressed the opinion that the name of "Joplin district" has become a fetish, upon which attention has been concentrated to the exclusion of other prospects that may develop into important zinc-producing centers. If this be so, and Dr. Keyes is not alone in the opinion, it is to be expected of course that sooner or later further zinc resources will be discovered in Missouri, Kansas and Oklahoma.

The spelter market was very unsatisfactory in 1908. It began with a greatly decreased production, a stock of 33,000 tons on hand, and the price 4⅜c., New York. Spelter is a good criterion of domestic business, inasmuch as practically the whole production is consumed in this country, while 80 per cent. of it was used by the galvanizers and the brass makers, wherefore spelter connects with both the steel and the copper manufacturing industries. During the first half of 1908 the demand for spelter was slack, but the smelters laid off so many furnaces that the small demand led to an advance in the price; however, by mid-year the latter was only ⅛c. above that of the beginning and the unsold stock had increased, estimates varying from 35,000 to 50,000 tons. The gradual improvement in the demand for consumption induced the smelters to fire more furnaces in the second half of the year, and the price averaged

[1] These remarks were written without respect to the effect that a tariff on zinc ore might have. Such a tariff, though contemplated by the bill pending before Congress, has not yet been enacted. If enacted its effect will be disturbing and serious as has been pointed out in *Eng. and Min. Journ.*

materially better, being a little over 5c. in November and December, but 1908 closed with a large unsold stock in the hands of the smelters, including what they were holding for speculators, the account of the latter being rather large. In this industry it is not common to have an unsold stock in first hands of more than 2 or 3 per cent. of the annual production.

The statistics given in an accompanying table show a large decrease in the spelter production and also in the consumption as compared with 1907. The Joplin district does not show a corresponding decrease. Its production of ore in 1907 was 286,589 tons; for 1908 its output was 259,609 tons. These figures correspond with about 143,000 and 130,000 tons of spelter respectively. Wisconsin made a small increase in 1908. It appears, therefore, that the Rocky mountain districts and the exporters of ore from Mexico suffered most. The year was one of the most disastrous on record for the smelters, whose competitive bidding for the necessary Joplin ore wiped out all margin for profit, of course to the advantage of the miners. This was a powerful factor in sustaining the Joplin production.

In spite of this mulcting of the smelters, the operators in the Joplin district found it hard to realize any profit and there was great complaint about the adverse conditions. The fact is, of course, that the old lens deposits, yielding 5 per cent. ore, have been to a large extent exhausted, during the last five years and a large part of the production has had to be derived from the "sheet ground," which yields only 2½ to 3 per cent. Prices for the concentrated product rose correspondingly and the producers were able to do finely up to the latter part of 1907. Yet, even under the adverse conditions of 1908, the Joplin district was immensely profitable, although not to the operators. The latter work generally upon leased land and pay the fee-owner a royalty upon what they produce. This royalty may have been reasonable when richer ore was mined, but under present conditions it is outrageous. Consequently the mining

I. PRODUCTION OF SPELTER IN THE UNITED STATES.

States.	1900	1901	1902	1903	1904	1905	1906	1907	1908
Colorado	.....	.....	.....	877	4,906	6,599	6,260	5,200	3,079
Illinois (a)	37,558	44,896	49,672	49,526	47,607	45,357	48,238	56,103	50,244
Kansas	57,276	74,270	87,321	87,406	103,721	114,948	129,741	133,561	99,136
Missouri	20,138	13,083	10,548	9,894	12,056	11,800	11,088	11,594	10,196
Oklahoma	.....	.....	.....	.....	.....	.....	.....	5,094	14,867
South and East (b)	8,259	8,603	10,698	10,799	13,513	23,044	30,167	38,060	32,989
Total tons of 2000 lb	123,321	140,822	158,239	158,502	181,803	201,748	225,494	249,612	210,511
Total tons of 2240 lb	110,028	125,734	141,283	141,520	162,324	180,132	201,343	222,868	187,776
Total metric tons	111,794	127,751	143,552	143,792	164,921	183,014	204,548	226,398	190,933

(a) Up to 1903, inclusive, includes also the production of Indiana. (b) New Jersey, Pennsylvania and Virginia, and (since 1903) West Virginia.

profit is going to the fee-owners. The operators command pity in their present situation, but they have made no serious or concerted attempt to secure a readjustment of the the royalty question. On the contrary, they have joined in an effort to secure a tariff on zinc ore for the avowed purpose of raising the price for spelter and consequently for ore.

II. PRODUCTION OF ZINC IN EUROPE AND AMERICA. (*a*)
(In metric tons.)

Year.	Austria.	Belgium.	France.	Germany.	Holland.	Italy.	Russia.	Spain.	United Kingdom.	United States.	Totals.
1896...	6,888	113,361	45,585	153,082	4,770	*Nil.*	6,257	6,133	25,278	70,432	421,786
1897...	6,236	116,067	38,057	150,739	6,600	250	5,868	6,244	23,805	91,070	444,946
1898...	7,302	119,067	37,155	154,867	6,700	250	5,664	6,031	28,387	103,514	468,937
1899...	7,192	122,843	39,274	153,155	6,235	251	6,331	6,184	32,322	117,644	491,331
1900...	6,742	119,315	36,305	155,799	6,845	547	5,963	5,611	30,207	111,794	465,438
1901...	7,558	127,170	37,600	166,283	7,855	511	6,090	5,354	29,877	127,751	516,049
1902...	8,309	124,780	36,300	174,927	9,910	485	8,280	5,569	40,244	143,552	552,356
1903...	8,949	131,740	37,416	182,548	11,515	126	9,901	5,134	44,110	143,792	569,971
1904...	9,159	137,323	41,600	193,058	12,895	189	10,607	5,887	46,218	164,921	621,857
1905...	9,204	142 555	43,200	198,208	13,550	5	7,520	6,184	50,125	183,014	653,565
1906...	10,711	148,035	46,536	205,691	14,650	69	9,610	6,209	52,587	204,548	698,646
1907...	11,359	154,492	(*c*)49,733	208,195	14,990	(*b*)	9,738	(*c*)6,000	55,595	226,398	736,500
1908...	14,224	165,018	(*c*)49,800	216,874	17,255	(*b*)	9,753	(*c*)6,018	54,472	190,933	724,347

(*a*) From the official statistics of the various Governments, except 1906, 1907 and 1908, for which years the figures reported by Henry R. Merton & Co. have been used where the official statistics were unavailable. In addition to the production reported in this table, Australia produced 286 long tons in 1903, 299 in 1904, 544 in 1905, 1008 in 1906, 980 in 1907, and 1086 in 1908. (*b*) Included in Austria. (*c*) An approximate separation of the total which is reported for "France and Spain."

III. ZINC SMELTING CAPACITY OF THE UNITED STATES.

Name.	Location.	Furnaces.	Retorts.
Grasselli Chemical Co	Clarksburg, W. Va	10	5,760
Matthiessen & Hegeler Zinc Co	Lasalle, Ill	5	4,320
Illinois Zinc Co	Peru, Ill	7	4,800
Sandoval Zinc Co	Sandoval, Ill	4	896
Mineral Point Zinc Co	Depue, Ill	6	4,800
Hegeler Bros	Danville, Ill	2	1,700
Edgar Zinc Co	St. Louis, Mo	9	2,016
Edgar Zinc Co	Cherryvale, Kan	24	4,800
Lanyon Zinc Co	Iola, Kan	5	3,000
Lanyon Zinc Co	Iola, Kan	5	3,000
Lanyon Zinc Co	Iola, Kan	5	3,000
United Zinc and Chemical Co	Iola, Kan	4	2,304
United Zinc and Chemical Co	Iola, Kan	2	(*a*)480
United Zinc and Chemical Co	Springfield, Ill	2	640
Cockerill Zinc Co	Iola, Kan	4	2,520
Cockerill Zinc Co	Iola, Kan	3	1,856
Cockerill Zinc Co	Altoona, Kan	6	3,840
Cockerill Zinc Co	Pittsburg, Kan	6	1,344
Cockerill Zinc Co	Nevada, Mo	3	672
Cockerill Zinc Co	Rich Hill, Mo	3	672
Granby Mining and Smelting Co	Neodesha, Kan	6	3,840
United States Zinc Co	Pueblo, Colo	6	(*a*)1,440
American Zinc, Lead and Smelting Co	Caney, Kan	6	3,720
American Zinc Lead and Smelting Co	Ikesring, Kan	6	3,840
New Jersey Zinc Co	Bethelehem, Penn	3	672
New Jersey Zinc Co	Palmerton, Penn	12	2,400
Bertha Mineral Co	Pulaski, Va	10	1,400
Prime Western Spelter Co	Iola, Kan	9	5,344
Prime Western Spelter Co	Iola, Kan	5	3,220
Bartlesville Zinc Co	Bartlesville, Okla	6	3,456
Pittsburg Zinc Co	Pittsburg, Kan	4	896
Lanyon-Starr Smelting Co	Bartlesville, Okla	5	2,880
National Zinc Co	Bartlesville, Okla	4	2,432
Chanute Zinc Co	Chanute, Kan	8	1,600
Totals		205	89,560

(*a*) Rhenish furnaces, with muffles.

IV. EXPORTS OF ZINC ORE AND ZINC OXIDE FROM THE UNITED STATES. (a)

Year.	Ore.			Oxide.		
	Short tons.	Value.	Value per ton.	Short tons.	Value.	Value per ton.
1896	(b) 2,324	$47,408	$20.40	(c)	.....	.....
1897	9,251	211,350	22.85	1,859	$104,140	$56.02
1898	11,782	299,970	25.50	3,925	252,194	64.25
1899	28,221	725,944	25.90	5,343	366,598	68.61
1900	42,062	1,134,663	26.98	5,656	496,380	87.76
1901	44,146	1,167,684	26.45	4,561	393,259	86.22
1902	55,733	1,449,104	26.00	5,358	433,722	80.93
1903	39,411	987,000	25.04	7,215	578,215	80.14
1904	35,911	905,782	25.22	8,157	628,494	77.05
1905	30,946	848,451	27.41	11,280	810,203	71.83
1906	27,720	733,300	26.45	15,578	1,149,297	73.78
1907	20,352	579,490	28.47	13,256	1,069,924	80.71
1908	26,108	877,745	33.60	12,008	845,070	70.37

(a) In addition to the exports of ore, 15,887 short tons of zinc dross (galvanisers' waste) were exported in 1906, 9,593 short tons in 1907, and 8,405 short tons in 1908. (b) Includes oxide. (c) Included in ore.

Zinc Mining in the United States.

Arkansas.—Even under the influence of high prices for spelter, production in this State fails to increase. With the lower values for spelter and ore in 1908 the mines showed a noteworthy falling off in production.

Colorado.—The production of zinc in this State was adversely affected by the low prices for lead and spelter, the zinc ore being produced almost without exception in connection with lead ore. As in 1907, the bulk of the production in 1908 was obtained from Leadville. The Yak Tunnel properties there maintained a large output, but the Moyer and Tucson mines of the Iron Silver company were idle throughout the year. Operations were suspended also by the Western Mining Company.

Iowa. (By James H. Lees.)—The financial difficulties of the autumn of 1907 put a stop to the developments which were in progress or contemplated. As in the case of lead, so with the zinc producers; operations were carried on for a few weeks of 1908, and during this time about 275 tons of blende was raised. At the beginning of the year the Avenue Top concentrator had some concentrates in its bins and during the early part of 1908 its sales, including this and what was raised and milled after the first of the year, amounted to 400 tons. This mill also concentrated some ore from other properties, so that a total of 435 tons was marketed at an average price of $48 per ton. After the early part of 1908 no ore was raised, and the Avenue Top mill, the only one at present in the district, has been closed and the entire plant has gone into a receivership.

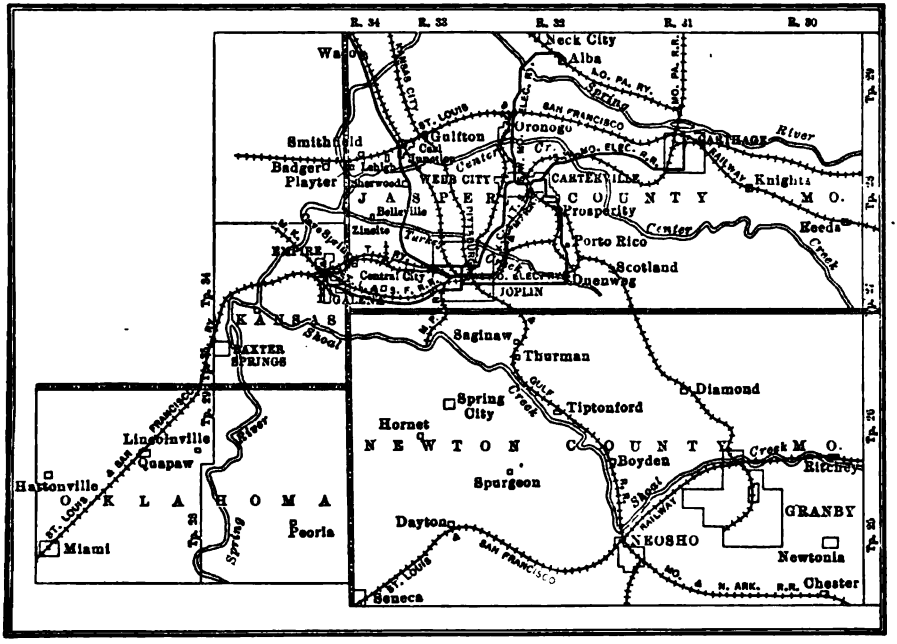
Missouri and Kansas. (By Jesse A. Zook.)—At the beginning of 1908 conditions in the Joplin district were bad owing to the general depression in business. This condition continued until toward the end of September. However, as soon as the mine owners and miners adjusted

V. EXPORTS OF DOMESTIC SPELTER FROM THE UNITED STATES. (a)

Year.	Plates, Sheets, Pigs and Bars.		Wares.	Total Value.
	Short Tons.	Value.	Value.	
1896	10,150	$1,013,620	$51,001	$1,112,029
1897	14,245	1,356,538	71,021	1,743,049
1898	10,499	1,033,959	138,165	1,724,188
1899	6,755	742,521	143,232	1,978,295
1900	22,411	2,217,963	99,288	2,317,251
1901	3,390	228,906	82,046	310,952
1902	3,237	300,557	114,197	414,754
1903	1,521	163,379	71,354	234,733
1904	10,073	1,094,490	117,957	1,212,447
1905	5,516	682,254	159,995	842,249.
1906	4,670	583,526	204,269	787,795
1907	563	75,526	186,283	261,807
1908	2,640	250,254	88,485	338,739

(a) There is also a comparatively insignificant re-export of foreign-made spelter and zinc wares.

the wage scale an increasing production followed, and, although the margin of profit for the mines in the sheet-ore area was comparatively small, yet prices for zinc and lead concentrates were high enough for



MAP OF JOPLIN DISTRICT.

the richer properties to resume production. Shipments increased from 14,000 tons in January to 20,000 tons in February. The average per month continued at approximately this figure until October, when the

output increased to 25,000 tons for the month. The more rapid recovery of lead prices, from the extremely low point at the close of 1907, made it possible to resume mining operations at many lead-zinc mines, especially in the sheet-ore area, that otherwise could not have been operated. New orebodies were opened in outlying camps and in the areas where the ore is easy to concentrate; these deposits helped to maintain the average production.

Much development work was done during 1908 along Spring river northwest from Carthage to Alba, and northeast from Alba. The richer

VI. IMPORTS OF ZINC AND ZINC OXIDE INTO THE UNITED STATES.
(In pounds.)

Year.	Sheets, Blocks, Pigs and Old.		Manufactures	Total Value.	Oxide.	
	Amount.	Value.			Dry.	In Oil.
1896	856,044	$25,904	$15,728	$41,632	4,672,781	311,023
1897	2,557,341	95,883	19,431	115,314	5,564,753	502,357
1898	2,742,357	109,624	13,448	123,072	3,342,235	27,050
1899	2,985,463	151,956	14,800	166,756	3,012,709	41,699
1900	2 013,196	97,772	36,836	134,608	2,618,808	38,706
1901	775,881	30,920	42,643	73,563	3,199,778	128,198
1902	1,238,091	46,713	37,191	83,904	3,271,385	163,061
1903	728,614	30,900	18,938	49,838	3,487,042	166,034
1904	933,474	44,326	11,918	56,244	2,585,661	224,244
1905	1,042,081	51,052	12,390	63,442	3,436,367	342,944
1906	4,407,481	253,310	17,385	270,695	4,191,476	292,538
1907	3,555,890	210,322	16,282	226,604	5,311,318	362,418
1908	1,551,676	85,885	7,474	93,359	4,635,101	210,166

VII. DELIVERIES OF SPELTER IN THE UNITED STATES
(In tons of 2000 lb.)

	1905	1906	1907	1908
Stock, Jan. 1	6,500	4,000	4,550	32,883
Production	201,748	225,494	249,612	210,511
Imports	521	2,203	1,778	881
Total supply	208,769	231,697	255,940	244,275
Exports	5,515	4,670	563	2,640
Stock, Dec. 31	4,000	4,550	32,883	25,000
Deliveries	199,254	222,477	222,494	216,635

of these newly found orebodies added materially to the output of this part of the district; many of the older mines of Alba-Neck have been abandoned. New discoveries in the Spring City-Spurgeon mines in Newton county south of Joplin kept pace with the abandonment of worked-out mines. This area, aside from the channel deposits, produces principally calamine. The panic halted underground development around Sarcoxie and Reeds, southeast from Carthage, but a large amount of prospecting was done with drills.

VIII. USES OF SPELTER IN THE UNITED STATES.
(In tons of 2000 lb.)

Purpose.	1905	1906	1907(b)	1908(b)	1905	1906	1907	1908
Galvanizing	100,000	124,000	150,000	119,000	50%	55%	62.7	62.2
Brass-making	52,000	57,000	44,000	33,000	26	25¼	18.4	17.2
Sheet zinc	34,000	36,000	30,000	27,000	17	16	12.4	14.1
Lead desilverization	2,400	2,500	2,600	2,500	1¼	1	1.1	1.3
Other purposes (a)	10,854	6,000	12,894	10,000	5¼	2¼	5.4	5.2
Total	199,254	225,500	239,494	191,500	100	100	100	100

(a) The apparent falling off in the consumption of zinc for "other purposes" in 1906 is explained by a more complete itemization of the consumption in 1905: in other words, there was probably more spelter used for brass-making in 1905 than the above table shows. (b) The statistics of consumption for 1907 have been revised from those stated in *The Mineral Industry*, Vol. XVI, so as to include the consumption of reclaimed spelter, which is also included in the statistics for 1908. For a full discussion of this subject see *Eng. and Min. Journ.*, June 12, 1909.

South and southeast from Galena much drill prospecting was done in 1908, and some ore was found. Low ore prices had little effect on curtailing the work of drill prospecting, but it retarded development work all over the district. A few lens deposits were developed west and southwest of Joplin, but no effort was made to develop the disseminated deposits west of town, as that ore requires higher prices than those prevailing in 1908 to be profitably mined. Aside from a general resumption of operations in the sheet-ore area that may be looked for with higher metal prices, new production will come from the points where prospecting and development work during the last two years have indicated favorable ore deposits.

The increased production of zinc ore from Miami, carrying an average of 5 per cent. of iron, was the foundation of a new system of basing ore values. As prices advanced toward the close of 1908, and all grades of ore grew in demand, one of the smeltermen discovered that he could afford to buy this ore with no deduction for iron, and a price based on ore containing 40 per cent. zinc was made, with $1 per ton advance for each 1 per cent. of zinc above 40 per cent., and with no deduction for the iron in the ore. As high as $22.50 per ton was paid for ore containing 40 per cent. zinc. This was equivalent to $42.50 per ton on the

IX. PRODUCTION OF ZINC OXIDE IN THE UNITED STATES. (a)

Year.	Quantity.		Value.		Year.	Quantity.		Value.	
	Short Tons.	Metric Tons.	Totals.	Per Short Ton.		Short Tons.	Metric Tons.	Totals.	Per Short Ton.
1897	26,262	23,285	$1,686,020	$64.26	1903	59,562	54,034	$5,005,394	$83.69
1898	32,747	29,708	2,226,796	68.00	1904	59,613	54,081	4,523,414	75.88
1899	39,663	35,982	3,331,692	84.00	1905	72,603	65,859	5,808,240	80.00
1900	47,151	42,775	3,772,080	80.00	1906	77,800	70,573	6,257,361	80.43
1901	46,500	42,266	3,720,000	80.00	1907	85,390	77,449	7,731,100	73.28
1902	52,730	46,929	4,023,299	76.30	1908	65,100	59,046	5,876,342	90.26

(a) The figures for 1905 and 1906 include zinc-lead pigment, which was not included in the statistics for previous years.

60-per cent. zinc basis, and as this ore carried 6 per cent. of iron without penalty it equalled a base-price of $48.50 per ton, while the best ore in the district was selling at $42.50 to $43 on a 60-per cent. basis. The smelting company making the innovation abandoned the new base within a few weeks, but other smelting companies continued to use it until the close of 1908. This may eventually lead to a complete change in the base-price for zinc ores. At present the system of making zinc-ore prices is exceedingly crude.

X. PRODUCTION OF ZINC ORE IN THE UNITED STATES.
(In tons of 2000 lb.)

State.	1904	1905	1906	1907	1908
	Tons.	Tons.	Tons.	Tons.	Tons.
Arkansas	(e) 1,900	2,200	4,200	4,088	2,582
Colorado	(a) 94,000	105,500	114,000	142,510	85,052
Idaho	Nil.	1,700	2,150	11,847	1,558
Kentucky	(d) 958	(d) 414	975	1,005	341
Miss.-Kan.	(b)273,238	(b)258,500	(b)280,260	297,126	273,430
Montana	Nil.	2,000	4,900	1,218	2,783
Nevada	Nil.	Nil.	7,090	4,593	1,445
New Mexico	(e) 21,000	17,800	30,000	4,281	2,290
New Jersey	(d)280,029	(d)361,829	404,690	368,710	399,232
Oklahoma				3,240	9,300
Utah	Nil.	9,265	10,700	9,043	709
Wisconsin	(c) 19,300	32,690	42,130	53,011	58,135
Others	(a) 2,600	(f) 3,800	(h) 850	(h) 2,241	(h) 1,530
Totals	693,025	795,698	905,175	902,923	838,377

(a) Estimated. (b) Production of Joplin district, plus output of southeastern Missouri, the latter as reported by the State mine inspector. (c) According to H. F. Bain, "Contributions to Economic Geology," 1904. (d) Report of State Geologist; crude ore. (e) Partly estimated. (f) Arizona, Nevada, Illinois, Iowa, Tennessee and Virginia. (h) Tennessee, Arizona and California.

XI. IMPORTS OF ZINC ORE INTO THE UNITED STATES.
(In tons of 2000 lb.)

Source.	1904	1905	1906	1907	1908
British Columbia	2,100	8,561	600	1,157	6,157
Mexico	?	(a)32,164	(a)88,900	(a)108,800	66,383
Totals	?	40,725	89,500	109,957	72,540

(a) The actual tonnage of ore imported was somewhat greater than this figure, but it included some mixed ore, which for statistical purposes has been reduced to the zinc ore equivalent. This table is based on reports from the smelters of the ore received by them from these countries.

Zinc ore sold at $39 per ton in the first week of 1908, an advance of $3 per ton over the closing price of 1907. In two weeks it jumped to $42.50 and the following week to $44 per ton. Beginning February at $42 it declined to $39, then rose to $41, and declined to $39.50, where it stood for three weeks. The two last weeks of April and the first half of May it was $39, the last half of May $38, and throughout June it fluctuated between $36.50 and $37.75. During all of July and the first week of August the price was stationary at $38, advancing to $40 at the end of August. September showed a range from $39.50 to $41.

In October it declined again, opening at $40.50, dropping to $39.50 and then to $39. November opened at $40, going up to $41, $43 and to $45 at the close. The first week of December the high price was $46.50, the next week $47, and the third week $46.50, closing the year at $47.

Montana. (By H. K. Welch.)—The principal zinc district in Montana lies in and around Butte. La France Copper Company's Lexington mine produced a good deal of zinc during 1908. A mill was erected for the separation of the zinc product, but it did not prove to be an unqualified success and has been closed. On the upper levels of the Elm Orlu and Butte & Superior properties large bodies of zinc ore were discovered in a search for copper. The Comet mine, near Basin, in Jefferson county, produced considerable zinc, and much experimenting has been carried on for the separation of this product.

New Jersey. (By Henry B. Kümmel.)—The mine of the New Jersey Zinc Company at Franklin Furnace was the only active zinc mine in New Jersey during 1908. The mine sent to the mill during the year 356,457 long tons of ore, an increase of 27,252 tons above the previous year.

New Mexico. (By Reinold V. Smith.)—The mines at Magdalena, in Socorro county continued to ship zinc carbonate, though the camps in the southern portions of the Territory were mostly closed down except for development work which was pushed with vigor in a few places. In Magdalena the lessees in the Kelly mine shipped 5300 tons in seven

XII. MARGIN ON JOPLIN ORE.

Month.	Spelter.	Ore.	Margin.(a)	Month.	Spelter.	Ore.	Margin.(a)
January	44.50	37.60	6.90	July	44.24	34.55	9.69
February	47.30	36.63	10.67	August	46.46	36.53	9.93
March	46.17	36.19	9.98	September	47.11	37.63	9.48
April	45.84	35.40	10.44	October	47.44	35.95	11.49
May	45.47	34.19	11.28	November	50.06	39.13	10.93
June	44.80	33.06	11.74	December	50.86	42.75	8.11

(a) Margin=difference between value of 1020 lb. of spelter at St. Louis and 2000 lb. of 60 per cent. ore at Joplin.

XIII. AVERAGE MONTHLY PRICE OF ZINC BLENDE ORE AT JOPLIN, MO. (a)
(Dollars per 2000 lb.)

Year.	Jan.	Feb.	Mar.	April.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
1900	30.23	29.36	28.45	28.42	26.92	25.00	24.23	25.67	24.25	24.25	24.45	25.40	26.50
1901	23.72	23.96	23.70	24.58	24.38	24.22	24.68	23.88	21.63	21.63	26.15	28.24	24.21
1902	26.75	27.00	28.00	28.85	29.23	34.10	34.37	32.50	33.58	33.58	32.10	29.25	30.73
1903	31.50	32.50	35.75	37.75	36.60	36.50	36.00	35.00	34.40	34.40	30.75	30.00	34.44
1904	32.12	34.00	36.00	36.40	34.63	32.62	35.00	37.00	40.40	40.00	44.25	46.13	37.40
1905	51.94	53.65	47.40	43.93	43.74	40.75	43.00	50.24	46.80	49.37	50.37	47.67	47.40
1906	49.33	49.25	45.60	44.00	41.50	44.20	43.88	44.38	43.20	42.50	44.43	45.55	44.32
1907	46.90	48.20	49.75	49.25	46.90	47.00	46.80	44.56	41.00	41.75	38.60	31.50	44.36
1908	37.20	36.63	36.19	35.40	34.19	34.06	34.55	36.53	37.63	35.95	39.13	42.75	36.63

(a) Base prices for 60 per cent. zinc ore.

months, sending out a car a day at the close of the year. The Kelly has been equipped with a 100-ton concentrator using the Sanders flotation process to separate the zinc sulphide. In the Graphic systematic development work was conducted continuously, with a force of from 20 to 40 men, still further blocking out the large orebodies already exposed.

Oklahoma. (By Charles N. Gould.)—The present workable deposits of lead and zinc in Oklahoma are all found in Ottawa county in the northeast corner of the State. The formation in which these minerals occur is either the Boon chert or the rocks immediately above. This is a continuation of the Joplin region. Prospecting has been done over several widely separated areas in Oklahoma but outside of Ottawa county no deposits of economic importance have been officially reported.

In Ottawa county there are three well-developed camps; in the order of their establishment they are: Peoria, Quapaw and Miami. The first shaft was sunk in the Peoria district in 1891. Since then there has been a steady output of ore from the camp. In the Quapaw district, the

XIV. SHIPMENTS OF ORE FROM THE JOPLIN DISTRICT.
(In tons of 2000 lb.)

Year.	Zinc Ore.	Lead Ore.	Year.	Zinc Ore.	Lead Ore.
1895.....	144,487	31,294	1902.....	262,545	31,625
1896.....	155,333	27,721	1903.....	234,873	28,656
1897.....	177,976	30,105	1904.....	267,240	34,362
1898.....	234,455	26,687	1905.....	252,435	31,679
1899.....	255,068	23,888	1906.....	278,930	39,189
1900.....	248,446	29,132	1907.....	286,589	41,742
1901.....	258,306	35,177	1908.....	259,609	38,532

next to be opened, ore was first secured by shaft in 1897. The Miami camp has been in existence scarcely three years. The first steam concentrating mill was completed less than a year ago. But in spite of extreme youth, development has been pushed rapidly forward in this camp and it bids fair to become, in a short time, the most important of the three. However, at the present time the Quapaw camp shows the most extensive development in the region, there being about 25 steam concentrating plants in operation besides a number of hand jigs.

In 1908 Oklahoma made great strides in the production of lead and zinc. Something like 6500 tons of zinc ore and 1000 tons of lead ore were produced in the last nine months of the year. Very few camps in the Ozark region equal this record. There are 12 concentrating plants either in operation or in course of erection. The ore runs from 8 to 35 per cent. lead and zinc.

There has not yet been enough development in the district to give a comprehensive idea of the amount of ore in the field but there is no

XV. PRODUCTION OF THE JOPLIN DISTRICT.
(In tons of 2000 lb.)

	Zinc Ore.			Lead Ore.		
	1906	1907	1908	1906	1907	1908
Oronogo	8,768	10,537	9,034	359	578	410
Webb City—						
Carterville	64,172	78,491	17,134	17,262	18,864	18,019
Prosperity	4,498	8,010	8,593	2,754	3,026	2,422
Duenweg	17,984	19,032	12,890	3,963	3,555	1,940
Totals	95,422	116,070	108,651	24,338	26,023	22,791
Carl Junction	46	997	1,531	.....	114	71
Zincite—						
Sherwood	4,798	3,032	2,296	382	100	85
Cave Springs	1,151	1,030	1,028	.....	17	6
Joplin	68,744	59,336	56,263	6,813	7,478	7,201
Spurgeon—						
Spring City	5,691	6,898	7,783	870	1,575	1,686
Diamond	44	38	.....	.....	.....	.....
Seneca	.....	.....	18	.....	.....	197
Granby	10,895	13,826	11,800	985	1,343	910
Totals	91,369	85,157	80,721	9,050	10,627	10,156
Alba-Neck	25,270	22,924	12,888	365	157	94
Carthage	3,575	2,428	4,289	62	9	11
Reeds	87	403	162	.....	.....	.....
Sarcoxie	229	1,473	2,293	5	.....	.....
Wentworth	368	569	655	.....	.....	.....
Totals	29,529	27,807	20,287	432	166	105
Stott City	1,161	918	159	20	.....	.....
Aurora	15,147	13,300	9,293	345	342	323
Ash Grove	.....	.....	19	.....	33	41
Springfield	324	193	1,508	44	255	19
Totals	(a)16,767	14,411	10,979	(a)413	630	283
Galena	29,553	27,762	18,249	4,190	3,572	2,909
Playter	80	298	249	4	17	37
Badger	14,100	11,900	10,100	43	46	509
Lawton	.....	28	.....	.....	.....	.....
Totals	43,733	39,988	28,598	4,237	3,635	3,455
Quapaw-Baxter	3,242	2,775	3,389	669	661	363
Peoria	.....	321	332	.....	.....	1
Miami	.....	60	6,652	.....	.....	1,378
Totals	3,242	3,156	10,373	669	661	1,742
District	280,062	286,589	259,609	39,139	41,742	38,532

(a) Morgan county is also credited with 135 tons of zinc ore and 4 of lead ore.

reason for doubting that ore of better quantity may be found below the level of the present workings. During 1908 development work was done largely by small companies. Several of the large plants were idle. The prospecting has been extensive and the gaps between Quapaw, Peoria and Miami have been practically closed up. Although there are still a number of small barren areas, the statement may be made that prospecting has shown the presence of very large bodies of ore yet undeveloped.

(By Jesse A. Zook.)—The Quapaw district should show an increased production in 1909, as wild-catting has stopped and legitimate development work is being pushed. Miami will increase its output, while some of the recent discoveries by drill in other parts of Oklahoma may become producing mines before the end of 1909. The remarkable increase in the production of Oklahoma in 1908 came almost wholly from the new camp of Miami, where a wonderfully rich deposit of zinc and lead ore was discovered in 1907. This deposit lies in blanket form. Thick sheets of rich ore are found throughout an area 1½x3 miles, while thinner sheets extend outward from the central body. The ore is coated with oil and asphaltum, on which account concentration on tables is practically impossible, the lighter ore particles being washed away with the oily water. The production of zinc ore in Oklahoma in 1907 was 2500 tons. At the end of 1908 the Miami district was producing a tenth of that amount

XVI. JOPLIN DISTRICT. PRODUCTION BY COUNTIES AND STATES.
(In tons of 2000 lb.)

	Zinc Ore.			Lead Ore.		
	1906 (a)	1907	1908	1906 (a)	1907	1908
Jasper county	199,322	207,703	189,403	31,965	33,898	30,259
Newton county	16,998	21,331	20,256	1,855	2,918	2,793
Lawrence county	16,308	14,218	9,452	365	342	223
Green county	324	193	1,527	44	288	60
Missouri	233,087	243,445	220,638	34,233	37,446	33,335
Kansas	43,733	39,988	28,598	4,237	3,635	3,455
Oklahoma	3,242	3,156	10,373	669	661	1,742
District	280,062	286,589	259,609	39,139	41,742	38,532

(a) In 1906 Morgan county produced 135 tons of zinc ore and 4 tons of lead ore.

in a single week. Although some of the leases at Miami are decreasing in output, several new mills are about ready to produce, and it is believed that in 1909 Miami will double or treble its output of 1908. The district is burdened by an excessive royalty, the rate being universally 30 per cent. The land was leased in large tracts by the persons who discovered the ore while drilling for water, and they placed the royalty basis at the maximum assessed in the older camps of Missouri. Toward the end of 1908 drill prospecting indicated the occurrence of ore in other parts of Ottawa county; at least three of these new areas look promising.

(From *Eng. and Min. Journ.*, March 6, 1909.)—The new mining camp is three miles due north of Miami, and four miles southwest of Quapaw. The country is flat prairie land. The ore consists of zinc and lead, and is found at a depth of 100 to 125 ft.; a face of from 16 to 30 ft. has been developed. Some timbering is required, but the roof is good and

machine drills can be used. The formation is unlike that of any other camp. Under about 30 ft. of top soil and clay is found about 5 ft. of loose gravel, and under the gravel 30 to 35 ft. of Cherokee shale. Below the shale is about 20 ft. of black, oily limestone and flint. Below this cap rock the orebody is found, impregnated with bitumen or crude oil, in flint and limestone in old solution-channels. The orebody averages 10 to 30 per cent. mineral in the proportion of six parts zinc to one of lead. Fine-grained iron pyrites is scattered throughout the ore. The zinc concentrates assay about 52 per cent. zinc and 7 per cent. iron, and bring about $15 under base price for 60 per cent. zinc. The lead is of good grade and brings nearly base price. The water problem was a hard one at the outset, but with several mines pumping, the water supply was soon no more than sufficient for use. There are now eight mills in operation, four mills contracted for and several shafts being sunk on drill holes. One square mile of territory has already been proved and 12 to 15 drills are at work in the camp, extending its limits every day.

Owing to the nature of the ore, the local concentration methods are not yet well adapted to handle the Miami product. The bitumen balls up in the jig beds and they require frequent cleaning. The fines float off in the oily water and are not saved by the tables as they should be. The richness of the ore is such that a great loss is entailed in handling too large a quantity and the tailings will, in every instance, have to be treated again. The iron in the concentrates is so great that roasting and magnetic separating will eventually have to be resorted to in order to raise the grade of the ore. One company is already reported to be planning to install roasters. The camp is on Indian lands. The Miami Royalty Company and others have first leases on most of it and pay about 5 per cent. royalty, subleasing to the operating companies at royalty of 25 to 35 per cent.

The result of the first year's work in the Miami field was a production of 6,202,390 lb. of zinc ore valued at $75,606 and 1,007,720 lb. of lead ore valued at $30,095; total, $105,701. The average price of zinc ore was $24.38, and of lead $59.73. This record was made by an average of three mills, which were shut down from Nov. 1, 1907, to April 15, 1908, operating only five and one-half months of the year and with ore prices at the lowest level for several years. The second year should see an average of at least 12 mills running full time.

The royalties are too high and will have to be reduced; other companies will follow the example of the Miami-Yankee and purchase a reduction of royalty, if it be not granted. The methods of concentration will be changed and improved as experience is gained in the field. The grade

of the concentrates will be raised by removing some of the iron and bitumen.

Virginia.—With one or two unimportant exceptions, the zinc industry in southwestern Virginia centers in the operations of the Bertha Mineral Company, of Pulaski. In 1908 the production of this company was far below normal. The Bertha mine, on the Cripple Creek extension of the Norfolk & Western railroad, ceased production in 1898. Parts of the old tailings dump, however, are now being worked over, and, after concentration, used in the furnaces at Pulaski. The Clark mine, situated a few miles beyond Bertha on the same line of railroad, was closed in 1903. The old Wythe lead and zinc mine, at Austinville, which was acquired by the Bertha company late in 1902, is producing but little ore at present. Here the energies of the company are being directed toward solving the problem of concentrating the "hard ore," a name locally applied to the fine-grained zinc-lead sulphide ore, carried in a limestone gangue. There is a good deal of this ore blocked out in the mine, and it is thought that the mill, with slight modifications, will be able to treat it successfully.

The smelting plant of the Bertha Mineral Company is situated at Pulaski, Va., on the main line of the Norfolk & Western railroad. It consists of 10 furnaces of the Belgian type. During 1908 only three of the 10 furnaces were in operation, making about 3300 lb. of spelter each per day. Three brands of spelter are produced at this plant: Bertha pure spelter, 99.98 per cent. pure; Old Dominion, containing from 0.2 to 0.5 per cent. lead, and Southern, containing about 1 per cent. lead.

Wisconsin. (By J. E. Kennedy.)—Mining in the Wisconsin district experienced a year of depression in 1908. But few new properties were opened and many concentrators were idle at all periods. Yet it is a significant fact that the production was the largest this district has ever made. The total shipment of zinc ore reported to Dec. 26 (57,472 tons), exceeded the smelter figure for 1907 by 4461 tons. The production of lead ore increased 100 per cent.; the total shipment reported to Dec. 26 (5393 tons) exceeded the total of 1907 by 2786 tons. The shipment of pyrites up to Dec. 26 (2536 tons) exceeded the total of 1907 by 2305 tons.

At the beginning of 1908 zinc-ore prices started upward from the low-water mark of 1907. By the end of January the base price for 60-per cent. zinc had advanced to $40 per ton. Many companies which had shut down were induced to resume by this rally in prices, but before these were well under way again a reaction in the market set in, the downward trend continuing until June, when the base price had landed back to $32 per ton. In spite of the low prices, shipments steadily increased. The Mineral Point Zinc Company re-entered the competitive

field for low-grade ores in April; the American Zinc Ore Separating Company began buying about the same time; and the Joplin Separator Works, at Galena, was also active. The increased competition for low-grade blende advanced the price for that ore. The market improved during July, August and September and the monthly average shipment during the last six months increased 60 per cent. over the first half of the year.

The metallic content of the zinc ore shipped was considerably greater than in 1907. During the first three months of 1908 there was little or no demand for the lower grade ore. The Mineral Point Zinc Company was practically out of the open market from December, 1907, to April, 1908, and other buyers of low and intermediate grades were restricted to limited purchases during this time. Aside from an increased tonnage from roasting and magnetic separating plants, the electrostatic separator, at Platteville, turned out a product assaying between 55 and 60 per cent. zinc.

While there are 18 different camps and shipping points in the district, the bulk of the output has come from Platteville, Benton, Hazel Green, Highland, Cuba City, Linden, Rewey, Harker, Livingston, Galena and Mineral Point. It has been the older camps, and, with few exceptions, the older mines which have been producing the ore. Benton and Cuba City developed exceptionally rich bodies of sheet ground. The individual production of many mines was materially increased by better methods of operation; labor, moreover, was plentiful and more efficient.

An important point was the completion and successful development of the electrostatic separator at Platteville by the American Zinc, Lead and Smelting Company, operating locally as the American Zinc Ore Separating Company. Blende concentrate assaying between 20 and 50 per cent. zinc is bought in the open market. The accompanying pyrite is separated from the blende and recovered as a product assaying between 42 and 48 per cent. sulphur. In the development of the process the original Huff machines were largely changed and improved upon. A constancy of current has been developed which assures a permanent success for the process. (The capacity of the plant, at present, is 70 tons per day.)

Nineteen properties were equipped with concentrators during 1908; eight of these plants were begun in 1907 and five were second-hand mills. Two new roasting and magnetic separating plants were built. Foundations were laid for a Sutton, Steele & Steele dry-process mill, at Platteville, but construction was deferred until the spring of 1909.

Comparatively little capital was invested in new enterprises. Shares of stock in undeveloped properties were unsalable. The event of the

year was the consolidation of the Empire, Acme, Royal and Mitchell Hollow properties, at Platteville, into the Wisconsin Zinc Company, subsidiary to the American Zinc, Lead and Smelting Company. The Picher Lead Company purchased the D. D. C. mine, at Rewey, to furnish raw concentrates to the Joplin Separator Works at Galena, a roasting and magnetic separating plant. The Mineral Point Zinc Company bought additional mineral lands at Highland and Benton. A few other sales of minor importance were made.

The average weekly output of zinc ore during 1908 slightly exceeded 1000 tons. During November and December the production had increased to 1500 tons weekly. With an open market for all grades of ore and the base price for 60-per cent. zinc ore above $40 per ton, the mines already opened up will yield in 1909, fully 2000 tons weekly.

Zinc Mining in Foreign Countries.

Toward the end of 1908 the convention among European zinc producers, under consideration all through the year, was consummated, to become effective Jan. 1, 1909. This convention appears to be the most severe and most scientific of any that has been organized in the zinc industry of Europe. Its fundamental idea is restriction of production to conform to the requirements for consumption at a certain price for spelter, viz., £19@23. If the unsold stock of spelter, to be reported monthly, exceeds a certain amount, there is to be a restriction of production according to schedule based on the excess of stock. Or if the price for spelter falls below £19 there is also to be a restriction of production. The regulation of production is to be in the hands of three commissioners. Bad faith on the part of any member of the convention, or failure to comply with the allotment, is to be punished by a fine. The members of the convention deposit with the commissioners good securities as a guarantee for the collection of the fine. The convention has fortified itself against the entrance of new interests in the smelting business by associating with itself some of the largest banking interests of the Continent, including the Deutsche Bank, which on their part undertake to refuse to assist in the financing new enterprises.

The convention comprises three groups of smelters, viz., (*a*) the German, including the Dutch and some of the French and Belgian producers; (*b*) the Franco-Belgian; and (*c*) the British. Among the important producers only Giesche's Erben remains outside, that famous concern adhering to its historic policy of independence. The German groups have incorporated as the "Zinkhüttenverhand," with a capital of 2,047,000 marks, of which Fritz Lob is chairman. Mr. Lob is director-general of the Hohenlohe interests; the organization of the convention

was largely due to him. The spelter of the German group is to be sold by the Metallgesellschaft, Frankfurt am Main; Beer, Sondheimer & Co., of Frankfurt am Main; and Aron Hirsch & Sohn, of Halberstadt. These concerns themselves have large zinc-smelting interests. Production of all the groups will be regulated according to the plan previously stated.

The agreement is to remain in force for two years, commencing Jan. 1, 1909. The output of the German group is limited to 255,700 tons for 1909 and 264,200 tons for 1910; the Franco-Belgian group to 174,500 tons for 1909 and 176,000 tons for 1910. The allotment for the English group has not yet been finally determined, but probably will be about 55,000 tons. The members of the several groups may arrange among themselves for changes in their allotments, providing that the total for the group does not exceed the quantity fixed by the convention.

Curtailment of output will take place whenever the average price of ordinary brands of spelter has been below £19 for an uninterrupted period of six months; or if the total stock of spelter shows an increase during such a period. If any member of the convention exceeds its allotment it will be subject to a penalty of 80 marks for every 1000 kg. of excess production. The convention may be dissolved after one month's notice whenever smelters not belonging to the convention, or new smelters, cause an increase of spelter production amounting in the aggregate to more than 2 per cent. of the estimated total production.

XVII. IMPORTATIONS OF ZINC ORE INTO BELGIUM.

Country.	1902	1903	1904	1905	1906	1907
Italy and Sardinia	74,740	71,674	66,538	65,069	71,840	69,490
France	25,522	27,268	32,288	33,769	33,627	23,807
Sweden and Norway	19,333	24,388	24,867	26,302	25,437	24,203
Germany	22,365	13,252	12,016	8,822	7,605	10,200
Spain and Portugal	52,993	66,566	74,762	56,025	61,408	62,299
Algeria and Tunis	26,511	32,595	37,483	51,430	42,905	42,754
Greece	361	1,486	4,896	6,385	4,919	6,773
Australia	5,288	1,114	18,274	30,551	38,974	54,807
England	9,017	9,200	6,447	10,011	4,821	2,815
America	26,227	31,133	21,806	8,914	21,553	12,615
Turkey	509	789	495	508	850	166
Japan	.....	.....	.....	2,626	11,335	16,961
Russia	.....	.....	.....	1,580	648	359
China	.....	.....	.....	169	16	642
Africa	.....	.....	.....	.....	.....	459
Other countries	2,146	4,416	4,448	1,059	184	.....
Totals	265,012	283,880	304,320	313,240	326,140	328,360
Belgium	5,750	5,355	3,050	3,265	2,630	2,235
Grand Totals	270,762	289,235	307,370	316,505	328,770	330,595

Canada. (By E. Jacobs.)—British Columbia in 1908 produced between 9000 and 10,000 tons of zinc ore. About 8000 tons were shipped from the Whitewater and Whitewater Deep mines, of which 3000 tons had been held over from 1907 and the remainder made in 1908. This

product averaged 44 to 45 per cent. zinc. The Ruth mine at Sandon, in the Slocan, sent to Kaslo nearly 1000 tons of zinc concentrate, the grade of which was raised to about 50 per cent. zinc at the Kootenay Ore Company's works and 250 tons of the product was shipped, leaving about 650 tons on hand. The Vancouver mine shipped 1036 tons of concentrate averaging 45 per cent. zinc and 40 oz. silver per ton. The remaining production was in smaller lots.

The works built at Nelson for the treatment of lead-zinc ores by the Snyder electrothermic process were in operation during the closing weeks of 1908. The capacity of the works is 10 tons of ore per day. It is claimed that metallurgically the process is successful, but that mechanical difficulties occasionally occur, the overcoming of which will be a matter of time and experience. At the Blue Bell mill the zinc-separation plant is not yet in full operation. It is expected to make a product of 40 to 45 per cent. zinc.

China.—The exports of zinc ore from Hankow, China, in 1907 amounted to 7355 tons, all of which came from the province of Hunan, and nearly all of which was shipped to Antwerp.

(By T. T. Read.)—China must have been at one time a considerable producer of zinc, for enormous quantities of brass have been used in coins and household and other articles. But apparently the production is now confined to zinc ore which is chiefly exported to Belgium. Practically all the production is confined to the province of Kuei-chou, but a small amount also comes from Fukien. The production seems to have increased enormously in 1908, the incomplete returns being 15,072 tons of ore; nearly double that for 1907. A very small amount of the metal is produced locally, about 100 tons having passed through the customs in 1907.

Great Britain.—The new works of the Central Zinc Company, at Seaton Carew, did not go into operation in 1908, but it is expected that they will be started in 1909. A new company, the Flintshire Silver-lead Mines and Spelter Works Company, is talking about building a

XVIII. VALUE OF ZINC ORE IN WALES.

Spelter, Foreign.	Blende, 50 Per cent.	Variations.		Spelter, Foreign.	Blende, 50 Per cent.	Variations.	
		Per Spelter price.	Per unit of zinc.			Per Spelter price.	Per unit of zinc.
£	£ s. d.	s. d.	s. d.	£	£ s. d.	s. d.	s. d.
17	3 6 6	5 3	3 0	22	5 0 9	6 9	4 2
18	3 11 0	7 3	3 3	23	5 7 6	7 0	4 5
19	3 19 0	7 6	3 6	24	5 14 6	7 6	4 8
20	4 6 6	7 6	3 9	25	6 2 0	7 6	4 10
21	4 14 0	6 9	3 11	26	6 10 6	8 6	5 2

smeltery at Greenfield, near Holywell station, in Wales. A plant for the manufacture of zinc oxide has been erected at Widnes, Lancashire. Zinc mining in Great Britain in 1908 showed but little change. In the Flintshire district, Wales, the mines were fairly prosperous. In Cumberland, where blende was formerly unworkable, some of the old workings are being cleaned out, and some output is being made. A correspondent in Wales contributes the following data as to the price received by his company for its blende ore, his letter being of Oct. 10, 1908:

"The rate for blende, delivered at the railway, is as follows, the terms being net cash on settlement of assay. The cost of freight to the smelter is approximately 17s. 6d. per 2240 lb. The scale of prices has been steadily falling during the last few years. Thus, on our contract for 1903-05, with spelter at £24, the scale price was £6 19s.; while in 1905-07 it was £6 4s. 6d. We consider the prices paid for English ores absurd, but we fancy that the Swansea smelters have formed a close ring and pool the ores. We are selling to the Continent at prices considerably above those paid by the Swansea buyers. Our blende assays somewhat above 50 per cent. zinc and produces a very good spelter."

XIX. GERMAN IMPORTS AND EXPORTS.
(In centners of 100 kg.)

	Imports.				Exports.			
	1905	1906	1907	1908	1905	1906	1907	1908
Spelter	268,406	370,359	284,591	326,223	623,233	633,947	622,379	689,026
Zinc sheets	544	808	1,171	1,171	189,817	172,979	214,759	186,756
Broken zinc	27,425	22,777	10,264	18,999	53,515	57,007	66,686	63,669
Zinc ore	1,265,773	1,790,360	1,847,026	1,998,403	389,727	426,055	348,632	394,502
Zinc dust (pouesiere)	.....	6,033	.....	.....	.....	40,443	.....	.....
Oxide of zinc	.....	52,310	70,492	50,483	.....	141,057	187,633	177,077
Lithophone	9,073	15,104	22,090	22,319	77,467	79,947	94,951	86,414

Mexico.—The production of zinc ore in Mexico was perhaps 15 per cent. larger than the imports into the United States from that country, but comparatively little zinc ore being exported from Mexico to Europe. Zinc ore is of widespread occurrence in Mexico, just as it is in the Rocky Mountain part of the United States, but at present a large part of the output is made by a few districts in the State of Chihuahua. In that State zinc ore is mined in the following camps: (1) Picachos, on the Kansas City, Mexico & Orient railroad, about 125 miles from the city of Chihuahua, where carbonate ore is the main product. (2) Santa Eulalia, about 16 miles from the city of Chihuahua, which produces carbonate ore exclusively. (3) San Isidro, on the Chihuahua & Pacific railroad, about 140 miles from the city of Chihuahua, where sulphide ore is produced. At San Isidro the Sutton-Steele process of dry concentration is

employed. (4) Almoloya, where carbonate and silicate ores are produced. (5) Parral and Santa Barbara, producing sulphide ore only. The zinc orebodies are always found associated with silver-lead. In

XX. AVERAGE PRICE OF SPELTER PER 1000 KG. IN SILESIA.

	1904	1905	1906	1907
First quarter	416 Mk.	468 Mk.	509 Mk.	511 Mk.
Second quarter	421	454	516	487
Third quarter	427	481	523	427
Fourth quarter	464	547	533	401
Average	432.00 Mk.	487.50 Mk.	521.50 Mk.	456.50 Mk.

many cases, the zinc ore is simply a by-product, although some mines, as at Picachos, are mined solely for zinc. It is claimed that there are, more or less exposed, in the State of Chihuahua, 1,000,000 tons of ore, as follows: Picachos, 50,000 tons; Santa Eulalia, 100,000; San Isidro, 200,000; Almoloya, 50,000; Parral and Santa Barbara, 60,000. The carbonate and silicate ores run from 32 to 45 per cent. zinc after hand sorting. The crude sulphide ores run from 10 to 35 per cent. zinc. The sulphide ores present hard and complex milling difficulties. The zinc ore resources of Mexico have been greatly overrated. Aside from the deposits of calamine, which probably will not be of long duration, the Mexican ore is chiefly mixed sulphides such as exist in the Rocky Mountain regions of the United States.

New South Wales.—The production of spelter in 1908 was 1065 tons against 984 in 1907. In addition thereto, zinc concentrate to the amount of 275,932 tons was exported against 236,251 tons in 1907. All of this came from Broken Hill. The Broken Hill Proprietary Company treated 276,703 tons of tailings, and produced 64,373 tons of zinc concentrate, averaging 15.61 oz. silver per ton, 10.23 per cent. lead and 42.44 per cent. zinc. The Sulphide Corporation made an output of 98,000 tons of concentrates and at its works at Cockle Creek produced 1013 tons of spelter. The concentrates of this company averaged 41 to 42 per cent. zinc, 11.3 to 12.5 per cent. lead and about 17.5 oz. silver per ton. The Minerals Separation Company produced 32,197 tons of concentrate averaging 45.2 per cent. zinc, 10.3 per cent. lead and 14.9 oz. silver per ton. The De Bavay company treated 74,187 tons of tailings from the North mine. Its plant proved to be capable of turning out 600 tons of concentrate per week. This company has made contracts with the Block 10 mine for the purchase of 368,000 tons of tailings, and also with the North mine to take all the tailings produced during the next 10 years. The output of concentrate by this company in 1908 was 22,590 tons, assaying 47.5 per cent. zinc, 7 per cent. lead and 6 oz. silver per ton. The

Zinc Corporation, using the Elmore process, treated 131,965 tons of tailings, yielding 45,707 tons of zinc concentrates, assaying 46.3 per cent. zinc, 7.3 per cent. lead and 16 oz. silver per ton, together with 2377 tons of lead concentrate, assaying 58 per cent. lead, 16.3 per cent. zinc, and 39 oz. silver per ton.

XXI. PRODUCTION OF ZINC IN NEW SOUTH WALES.
(In tons of 2240 lb.)

	1903	1904	1905	1906	1907	1908
Spelter	286	299	544	1,008	984	1,065
Zinc in ore exported	14,625	22,318	30,637	33,427	76,645	113,853

The experimental zinc-distillation furnace of the Broken Hill Proprietary Company, at Port Pirie, was put in operation May 16, 1908. This furnace has 144 retorts, 72 per side in three rows of 24 each. It is fired with gas from a Schmidt & Desgraz producer. The furnace is of the Rhenish type. The secondary air is supplied by a fan and is preheated to about 800 deg. C. in passing through a flue between those which convey away the hot waste gases. The waste gases pass through a steam boiler which generates sufficient steam to run the fan and producer. The furnace is expected to treat 5½ tons (2240 lb.) of ore per day. The directors of the company have authorized the building of nine more furnaces, at a further cost of about £81,000, about £19,000 having been already expended in the erection of the first furnace and experimental work. It is expected that the new furnaces will be producing spelter by the end of 1910. The production of the 10 furnaces is estimated at 8000 tons of spelter per annum, for all of which a market is available in Australia and the East. With spelter at £20 per ton, it is reckoned that a satisfactory return will be secured. In the recent experimental work clay was imported from Belgium, but suitable material has since been discovered in Australia.

The Zinc Corporation, Ltd., was organized several years ago to deal with the tailings at Broken Hill, of which it purchased several million tons. H. C. Hoover was largely instrumental in organizing the company and has been its technical director. It was the intention to employ the Potter flotation process, but this proved a failure and the company passed through a series of experimental vicissitudes which carried its fortunes to a low ebb. Under the orders of Mr. Hoover the Elmore vacuum process was finally installed, and from the beginning of its application in 1908 the indications have pointed to success. At the end of 1908 it could be pronounced that both the process and the Zinc Corporation are successes. This is a great, and well-deserved triumph, for Mr.

Hoover. In this connection the statement made by him at the annual meeting of the company, last July, is of peculiar interest. In substance it was as follows: "The ore at Broken Hill contains lead, zinc and silver. The mining companies there crush this ore to the fineness of sand and extract from it, by water concentration, about 60 to 70 per cent. of the lead and silver contents. The residues from this treatment, which are sent to the dumps, contain, therefore, practically the whole of the zinc and between 30 and 40 per cent. of the original lead and silver in the ore. We have already bought and paid for a million tons of these residue dumps, and we have the right to buy some millions of tons more. The residues we are treating at the present time are those from the Block 10 mine, and they average about 20 per cent. of zinc, about 8 oz. of silver and 5 or 6 per cent. of lead. We take the residues from these old dumps by our own railway and dump them into large bins which we have prepared underneath the track at our works. From here the material is carried by a belt conveyer to bins at the top of the mill. Thence they pass to Huntington mills and grinding pans. After this regrinding, the material passes to agitators, where it is mixed with oil and sulphuric acid, and is drawn from the agitators through the Elmore vacuum machines which deliver a mixed zinc, lead, and silver concentrate.

"The second stage of the treatment is the separation of the lead from the zinc in this mixed concentrate, making one in which zinc predominates and one in which lead is the principal content. The zinc smelters will pay only for lead in excess of 8 per cent., and then only half the value of the lead. Likewise, they will pay only 1s. 3d. an ounce for the silver over 5 oz. a ton, and then pay for the zinc content at the average price of zinc over the year of purchase, less the cost of shipment to Europe, smelting charges, loss in recovery, and their profits on smelting. Nor will the lead smelter pay for zinc in the lead concentrate —in fact, he penalizes us if we have over 12 per cent. zinc in our lead.

"The average contents of the mixed concentrate is about 43 per cent. zinc, 12 per cent. lead, and 16 oz. silver. From this we are now obtaining lead concentrates, at the rate of 230 tons per month, and by doing so we have increased the contents of the zinc concentrates from 43 per cent. to 47 per cent. zinc. On our present output, if we sold the mixed concentrate as it stands, we should realize about £3500 less than we do realize by first separating a portion of the lead. On the basis of our last week's output we are earning a net sum of £7000 per month above all expenses, out of which about £3000 should be set aside for cost of tailings worked up, or a net amount of about £4000 per month, or £48,000 per annum profit. I am in this taking metals at the bedrock prices I have mentioned (zinc, £19¼ and lead, £12½) and taking working expenses at their average for June.

This profit is, I believe, capable of very important increase from many directions, and it is the directions from which these increases may be anticipated that I wish to make clear. First, the working expenses are always high at the initial stages of a new plant. Although our plant is absolutely mechanical, and no human hands touch the material from the beginning to the end, it takes months to train men and organize them to the best advantage. In this business men have had to be trained from the ground up as this is entirely a new business in which no men of experience can be obtained. Moreover, we have been buying water, and we shall soon have our own supply for at least part of the year. Our expenses at the start were about 14s. per ton, and for the month of June were reduced to 12s. 3d. per ton. I believe it will be ultimately decreased to 8s. per ton, which would give an additional profit of over £2500 per month. Second, the mill is treating about 13,000 tons per month. I believe that by gradually strengthening the weaker links of the plant this can be increased by 1000 or 2000 tons a month. This would mean an increase in working profit by £500 to £1000 per month. Third, we are obtaining an extraction of over 88 per cent. of the original zinc contents of the residues. This shows an increase from about 72 per cent. the first month. We have obtained in systematic experimental work as high as 92 per cent. extraction of the zinc, so that we may make a minor improvement here, although we have not much room for improvement over present results on this line. Still, an additional 4 per cent. recovery would augment our profits by £500 per month. Fourth, the improvement in retreatment of the mixed lead and zinc concentrate is one of the most important matters with which we have to deal, and one where very important possibilities lie.[1]

"The lead extraction from the original residues at present average about 70 per cent. in the Elmore concentrates; but we get out of the mixed concentrates only a part of the lead in them, equal to an actual extraction as final salable lead of only 17 per cent. of the lead in the original material. In experimental work we averaged a recovery of 40 per cent. of the original lead, instead of 17 per cent., as above. We hope eventually to duplicate this in regular operation of the plant. If we do so, the increased profit from the extra lead and the further increased purity of the zinc would, on present tonnage, and at the metal prices I have given, amount to between £2500 and £3000 per month. We have, therefore, a possible increase of the present profit from reduction of expenses of £2500 per month, from increased zinc extraction £500, from increased tonnage £500, and from increased lead extraction £2500—or a total of £6000 per month, or £70,000 per annum, and this

[1] The expectations of Mr. Hoover as to increased profits have already been realized to an important extent. Editor.

without increased price of metals. To do this, or something approaching it, we shall need to lay out £7000 to £8000. To recapitulate, at the present moment we are earning a profit above the cost of the material we treat, and on present low prices of metals, of about £48,000 a year. If we succeed in these technical improvements we shall raise it to over £100,000 per annum. I cannot guarantee that we shall accomplish all this. We have had such results for short periods, and every week since we started has shown some improvement. One thing I can say, and that is, we will accomplish part of it."

Russia. (By I. I. Rogovin).—The zinc industry of Russia is insignificant. The metal is produced in only two regions, the bulk of the output being made in Poland, besides which there is an insignificant production in the Caucasus. The output of the latter district in 1907 was 23,595 poods. In 1908 it was about the same.

Spain.—Early in 1908 it was reported that the Sociedad Minera y Metallurgica de Penarroya was to begin construction of a plant at Penarroya for the roasting and distillation of zinc blende. The works are expected to treat about 5000 tons of ore annually, making between 1600 and 1700 tons of zinc.

Transvaal.—Zinc mining is developing in this colony, especially in the dolomite region at the head of the Marico river. The ore is blende, some of it of very high grade, certain shipments having assayed 60 per cent. zinc.

The Spelter Market in 1908.

New York.—The large accumulations carried over from 1907 exerted an influence which was felt throughout 1908. At the opening of 1908, weak holders had been fairly well eliminated and the existing stocks were in a large measure concentrated in strong hands. Production was at low ebb. The consuming community began to emerge from the demoralization incident and subsequent to the panic. The purchases of spelter for legitimate requirements had been practically nil for some months. The tendency everywhere was to work up the stocks on hand to the lowest point possible, which policy was facilitated by minimum requirements. Meanwhile the production has adjusted itself to this state of affairs. Unlike other base metals, the greater percentage of zinc ore is obtained from small properties, the operation of which can be stopped without serious sacrifice. The cost of idleness leaves a very small margin for losses incurred through sales of ore below cost, so that whenever spelter prices show a permanent decline, the production is automatically checked, depending upon the cost of producing in various mines.

At the opening of 1908, the reduced output was just about sufficient to supply current requirements. When it was found that stocks were

not increasing, speculative buying became an influence in the market and brought about an appreciation in the price, from 4.15c., St. Louis in early January, to 4.40@4.45, St. Louis at the end of that month. The movement made further progress in February, during which month there was an uninterrupted advance to 4.65@4.70c., St. Louis. Realization of speculative holdings caused a decline in March to about 4.50c., St. Louis, and around this level the market fluctuated within narrow limits until well into May.

During all these months the demand was never large enough to enable liquidation of the old stock. In consequence, some of the first-hand holders lost patience and in their efforts to force sales, depressed prices throughout June and July, bringing about a decline to 4.27½@4.30, St. Louis. They, however, accomplished their purpose by shifting a good portion of the stock to consumers, who wisely took advantage of this opportunity to replenish their empty yards at low prices. The revival in general business during the fall months was reflected in the spelter market through increased business not only from galvanizers, but also from the brass manufacturers, who then, for the first time in over a year and a half, became buyers of anything like normal quantities. As a result, an advance started in August, which continued slowly but steadily throughout the remaining months of the year. The rise would doubtless have been more rapid if it were not that the remaining surplus was gradually fed out with the current output. The market closed firm at 4.97½@5c., St. Louis.

XXII. AVERAGE MONTHLY PRICE OF SPELTER PER POUND IN ST. LOUIS.

Year.	Jan.	Feb.	Mar.	April.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts
1903....	4.688	4.681	5.174	5.375	5.469	5.537	5.507	5.550	5.514	5.350	4.886	4.556	5.191
1904....	4.673	4.717	4.841	5.038	4.853	4.596	4.723	4.716	4.896	5.033	5.363	5.720	4.931
1905....	6.032	5.989	5.917	5.667	5.284	5.040	5.247	5.556	5.737	5.934	5.984	6.374	5.730
1906....	6.337	5.924	6.056	5.931	5.846	5.948	5.856	5.878	6.056	6.070	6.225	6.443	6.048
1907....	6.582	6.664	6.687	6.535	6.201	6.269	5.922	5.551	5.086	5.280	4.775	4.104	5.812
1908....	4.363	4.638	4.527	4.495	4.458	4.393	4.338	4.556	4.619	4.651	4.909	4.987	4.578

XXIII. AVERAGE MONTHLY PRICE OF SPELTER PER POUND IN NEW YORK.

Year.	Jan.	Feb.	Mar.	April.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.
1900....	4.65	4.64	4.60	4.73	4.53	4.29	4.28	4.17	4.11	4.15	4.29	4.25	4.39
1901....	4.13	4.01	3.91	3.98	4.04	3.99	3.95	3.99	4.08	4.23	4.29	4.31	4.07
1902....	4.27	4.15	4.28	4.37	4.47	4.96	5.27	5.44	5.49	5.38	5.18	4.78	4.84
1903....	4.87	5.04	5.35	5.55	5.63	5.70	5.66	5.73	5.69	5.51	5.39	4.73	5.40
1904....	4.863	4.916	5.057	5.219	5.031	4.760	4.873	4.866	5.046	5.181	5.513	5.872	5.100
1905....	6.190	6.139	6.067	5.817	5.434	5.190	5.396	5.706	5.887	6.087	6.145	6.522	5.882
1906....	6.487	6.075	6.209	6.078	5.997	6.096	6.006	6.027	6.216	6.222	6.375	6.593	6.198
1907....	6.732	6.814	6.837	6.685	6.441	6.419	6.072	5.701	5.236	5.430	4.925	4.254	5.962
1908....	4.513	4.788	4.665	4.645	4.608	4.543	4.485	4.702	4.769	4.801	5.059	5.137	4.726

London.—Some revival in the demand from the galvanizers stimulated an upward movement in January, which culminated on the 20th, with £21⅜ for ordinary brands. The closing prices for the month were £20⅝ for ordinaries, and £21⅜ for specials. In February there was a recovery to £21½ for ordinaries, the month closing at £21⅛ for those brands and £22⅜ for specials. In March the fluctuations were slight, that month closing at £21⅛ for ordinaries and £21½ for specials.

Some large business in April led to an advance of about 7s. 6d., but subsequently there was a persistent decline, the month closing at £21 for ordinaries and £21⅛ for specials. The decline continued right through May, which closed at £19⅞ for ordinaries and £20⅛ for specials. June showed a continuation of the decline, and closed at £18⅜ for ordinaries and £19⅜ for specials.

XXIV. AVERAGE MONTHLY PRICE OF SPELTER IN LONDON.
(Pounds sterling per ton of 2240 lb. of good ordinary brands.)

Year.	Jan.	Feb.	Mar.	April.	May.	June.	July.	Aug.	Sept.	Oct.	Nov.	Dec.	Year.
	£	£	£	£	£	£	£	£	£	£	£	£	£
1905.....	25.063	24.594	23.825	23.813	23.594	23.875	23.938	24.675	26.375	28.225	28.500	28.719	25.433
1906.....	29.225	25.844	24.563	25.781	27.000	27.728	26.800	26.938	27.563	28.075	27.737	27.938	27.020
1907.....	27.125	25.938	26.094	25.900	25.563	25.469	23.850	21.969	21.050	21.781	21.438	20.075	23.771
1908.....	20.563	20.875	21.075	21.344	19.906	19.000	19.031	19.350	19.563	19.750	20.875	20.625	20.163

July opened with a stagnant market, but prices were further depressed by forced sales of arriving parcels which were marketed chiefly at £18. Subsequently, some speculation developed, which raised prices to £20, the month closing at £19 for ordinaries and £19⅜ for specials. There was but little change in August, which closed at £19¼ for ordinaries and £19¾ for specials. In September prices advanced to £19½, closing at £19¾ for ordinaries and £20⅜ for specials.

October experienced a steady trade throughout, but fluctuations were comprised within narrow limits, and the month closed at £19 15/16 for ordinaries and £20⅜ for specials. In November prices advanced steadily, closing at £21 7/8 for ordinaries and £21⅛ for specials. In December some speculative holdings pressed upon an unsupported market and the price gradually receded to £20⅝, but the month closed at £21 for ordinaries and £21¼ for specials.

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ZIRCONIUM.

By Charles Baskerville.

Zirconium is allied to titanium and is rather widely diffused in igneous rocks. Zirconium compounds are almost always present in nepheline syenites. Zirconia is extracted from zircon, which is mined for technical purposes in North Carolina, especially near Green river, Henderson county. Cyrtolite is abundant in Baringer hill, Llano county, Texas, and the deposits await development.

Zirconia possesses great radiating power and is very refractory. Cylinders of zirconia have been used in the Drummond light instead of lime and magnesia cylinders, and zirconia is in demand for producing refractory compounds suitable for incandescent illumination. With thorium and yttrium, zirconium enters into the manufacture of Nernst lamps, and filaments of zirconium are used in "Zirconium lamps." Zirconia yields with alizarin a color like one produced by chromium material. The use of zirconium nitrate as a food preservative has been patented. The market is well supplied with zirconium compounds at present.

Zircon occurs in square prisms and pyramids, also in irregular forms and grains. It is found in crystalline rocks, especially granular limestone, chloritic schists, gneiss, syenite; also in granite, iron-ore beds, and alluvial deposits. It is one of the earliest minerals to crystallize from the cooling magmas, and the first one among silicates. The luster of zircon is adamantine and the color is colorless, grayish, yellowish, or brown. Its harness is 7.5, and the specific gravity is from 4.2 to 4.86. It is brittle, and the streak is white. Zircon usually contains a small percentage of ferric oxide.

Zircons are especially abundant in Henderson county, N. C. Other specimen localities are: Maine, Vermont, Connecticut, New York, New Jersey, Pennsylvania, Colorado and California. Eudialyte and eucolite occur in Arkansas.

A REVIEW OF THE LITERATURE ON ORE DEPOSITS IN 1908.

By J. F. Kemp.

Since the period covered by my last review in The Mineral Industry, copper has indeed fallen in price, but if anything has risen in geological interest. The review for 1907 opened with a summary of recent work upon its methods of precipitation in the veins and rocks, and especially with the reactions bearing upon the peculiar native copper of Lake Superior. These exceptional deposits possess more than ordinary interest and upon the problem of the source and segregation of so much native metal one cannot well forbear dwelling with especial care. In June, 1908, Dr. A. C. Lane read before the Lake Superior Mining Institute[1] a paper which contains the completest summary yet given regarding the composition of the shallow and deep waters in the mines. There is a very striking increase of chlorine with depth, to such a degree in fact, that it is more than enough to satisfy the sodium present and appears in relatively large percentages of calcium chloride. This feature, Dr. Lane finds to be everywhere characteristic of deep-seated as contrasted with surface waters. Thus if we deal in atomic or molecular proportions, obtained by dividing the percentages by the atomic or molecular weights, as the case may be, we can easily see which molecule, the NaCl or the $CaCl_2$, is in the greater number in the waters. The molecules of sodium chloride decrease steadily with depth, and those of calcium chloride increase.

As applied to the copper and leaving out many parallel reactions of alteration, the general suggestion of Dr. Lane, fortified by the chemical experiments of Dr. Fernekes is as follows: The basalt flows contain naturally small percentages of copper. While still heated they probably absorbed or "sucked in" sea-waters charged with sodium chloride and certainly in later time atmospheric waters not provided with this compound until it was obtained during their migrations through the rocks. Relatively rich in sodium chloride these waters pass downward, extracting copper in solution as copper chloride. In reactions with the glassy base or with the original minerals of the volcanic rocks, sodium silicates were formed, copper was precipitated, and calcium chloride resulted. From widely extended areas descending circulations became concentrated along favor-

[1] As yet only available in advance reprints.

able lines and shoots of relative richness resulted. A directing influence has often been exerted upon the circulations by faults or slips. Many additional reactions enter, too numerous to be cited here, but the essential characteristics are those outlined. The hypothesis has the important feature of using chlorine as the copper solvent and of coinciding with experiments, yet some good observers familiar with Keweenaw Point are still upholders of uprising waters as the introducers of the copper and some are inclined to regard the deep-seated waters with their peculiar composition, as possibly magmatic in origin.

Dr. Lane also treats of waters in the iron mines and in a more limited way refers to the iron ores.

The veins at Butte Mont., have been among the most instructive of the many deposits worked for copper throughout the world. Secondary enrichment may be said to have had its demonstration in their careful study by S. F. Emmons, W. H. Weed and H. V. Winchell. The starting point of the processes has been a low-grade copper-bearing pyrite, and while perhaps not absolutely demonstrated, the feeling has been that copper might replace some of the iron in the pyrite molecule and be only freed in later alteration. Recently J. F. Simpson[1] has utilized the Campbell method of polishing surfaces of the opaque metallic minerals in the search for minute inclusions. Samples of ores were obtained down to 2000 ft., which were believed by R. H. Sales, geologist of the Amalgamated company, to be original in their character. When polished and magnified, they all revealed included copper minerals whose order of deposition was, 1, chalcopyrite; 2, enargite and bornite; 3, chalcocite. The exact order of enargite and bornite could not be made out, they were so closely associated. Wherever copper could be shown by analysis, copper minerals as distinct from pyrite were found with the microscope. Cupriferous pyrite, with copper replacing iron in the molecule FeS_2, seems therefore improbable. The corroboration of these results by the study of lean copper-bearing pyrites collected in other localities will be of much significance.

A number of valuable contributions were made during 1908 to the general chemistry and principles of ore deposition. In former volumes of THE MINERAL INDUSTRY, adsorption and precipitation by comminuted feldspar and similar topics have been reviewed. Dr. E. C. Sullivan has, however, added a new discovery within the past year which is both suggestive and illuminating. Dr. Sullivan has observed that, when a solution of ferric sulphate is passed through a Pasteur filter, 18 per cent. of the iron is held in the tube. Repeated passage of the same solution serves each time to reduce further the dissolved metal. The explanation suggested is that hydrolysis has produced a colloid form of iron oxide which is caught in

[1] "The relation of copper to pyrite in the lean copper ores of Butte, Mont.," *Econ. Geol.*, 1908, III, 628.

the porcelain. There was no demonstrable chemical effect upon the silicates of the filter. The application of these results to natural phenomena comes in the passage of solutions of heavy metals through wall rocks and the possible abstraction of the metals from solvents so as to yield impregnations. The justifiable inference is, that we have in this reaction a true cause of the enrichment of wall rocks outside of actual crevices in which ores have been deposited.

In *Bull.* 330 of the U. S. Geological Survey, entitled "The Data of Geochemistry," Dr. F. W. Clarke has brought together a vast amount of valuable matter in a very condensed and yet readable form. One is, of course, at once reminded of the early work of Gustav Bischoff and of the later volumes of Justus Roth, prepared along similar lines, and yet, somehow, without lacking in any respect the thoroughness of the older writers, Dr. Clarke has escaped the ponderousness of German scholarship and has prepared a book that is no less readable than valuable as a work of reference. Chapter XV is devoted to the metallic ores. After an introductory general section on the source of the metals and their concentration, each one is taken up in detail; its minerals are reviewed; its determinations in rocks are summarized and much solid basis of fact is afforded for all lines of reasoning upon the origin of ore deposits. Nothing seems to have escaped Dr. Clarke and references from all over the world are freely cited apparently up to the hour of going to press.

Until the past year literature upon the enormously important lead-silver mines of the Coeur d'Alene district, Idaho, has been limited to a very excellent paper by J. R. Finlay in the *Trans.* A. I. M. E. and to some valuable but brief notes by Waldemar Lindgren in *Professional Paper* 27 of the U. S. Geological Survey. There has been much need of a detailed study based on careful maps and structural work. Such complete records are now available in *Professional Paper* 62 of the Survey's reports, by F. L. Ransome and F. C. Calkins, entitled "The Geology and Ore Deposits of the Coeur d'Alene District, Idaho." The stratigraphical and structural geology is described by Mr. Calkins; the mining geology by Mr. Ransome. In the stratigraphical and structural portion we learn that the formations, except the recent auriferous gravels, belong in the Precambrian and consist of sandstones, shales and very subordinate limestones. They have a total of over 17,000 feet, and are divisible into six different stages as follows:—

Striped Peak shales and sandstones	1000+ ft.
Wallace sandstones, shales and limestones	4000 "
St. Regis, shales and sandstones	1000+ "
Revett white quartzite	1200 "
Burke shales and sandstone	2000 "
Prichard shales and sandstone	8000+ "
Total	17,200+ "

Later than all these a great intrusive mass of syenitic rocks or monzonites entered, followed by some diabase and a few basaltic dikes. The lead-silver ores are mainly in the Revett and Burke formations, but are not unknown in the Prichard, St. Regis and Wallace. The gold veins, which are of minor importance, are in the Prichard. It is a striking fact that lead-silver ores, which ordinarily favor limestones more than any other wall-rock, are here almost exclusively in quartzite.

This great series of shallow water sediments has been extensively folded and faulted. The faults are often of great size and extent, but the ore has not been deposited in the large ones. On the contrary, it favors relatively small displacements, which have at times sheeted the country rock, at times developed single fissures. Into these have come, presumably under pressure and while hot, solutions with ferrous carbonate and lead sulphide, which have filled open spaces and replaced the quartzite with siderite and galena. The sericitic filling between the quartz grains first falls a victim and later the quartz itself. There is some vein quartz also in the ore, and there are in some mines other metallic minerals than galena, as for instance stibnite and zinc blende. Copper minerals appear occasionally but are of small moment except in the one copper mine of the district, the Snowstorm. The gold-veins near Murray are the usual quartz-veins, and run parallel with the bedding. In the lead-silver mines the galena is now shown to be present over a vertical range of 2600 ft. but may be much more as the ore-shoots still continue in depth. Mr. Ransome thinks there may have been existent when the ore was deposited a lode of 4000 ft. of sediments now gone. There seems little doubt that the ore-bearing solutions emanated from the great, heated mass of the monzonite in depth and rising through the sediments formed the orebodies. The data given for the vertical range of deposition are timely and important. They will serve as a basis of reasoning upon this attractive and important phase of the subject. The monograph is an admirable instance of careful observation and clear presentation. Additional details regarding the gold-quartz veins in the Prichard slates near Murray, are given by H. S. Auerbach in an interesting paper in *Eng. and Min. Journ.* of July 12, 1908, p. 65. Scheelite is found in such quantity that it is saved and sold for tungstic oxide, some 15 tons having been produced in 1907.

In the early months of 1908 a very interesting discussion was started in the columns of *Economic Geology* upon the occurrence and cause of shoots and the localization of values in metalliferous deposits.[1] There is no more important feature of mineral veins than this, and if a series of records of occurrences can be accumulated, their interpretation cannot fail to be both

[1] The first paper opening the discussion was by J. D. Irving, *Econ. Geol.*, III, 143. It was followed by F. C. Smith, 224-229; R. H. Sales, 326-331; F. L. Ransome, 331-337; Max Boehmer, 337-340; H. V. Winchell, 425-428; H. Sjögren, 637-643.

interesting and suggestive. In the opening paper Professor Irving classifies shoots as follows:

- Shoots
  - Shoots of Occurrence
    - Due to available open spaces.
    - Due to chemical effect of wall-rock.
    - Due to impounding of mineral waters.
  - Shoots of Variation
    - Due to chemical effect of wall-rock.
    - Due to intersecting veins.
    - Due to causes not fully understood.

Shoots of occurrence embrace those variable bodies of ore, like the successive lenses of pyrites or magnetite, which occur in a linear series but are separated by pinches, wherein the walls of barren rock cut them off. Shoots of variation embrace the richer portions of a vein, which is mineralized throughout its entire extent. The general scheme presents the most obvious varieties of shoots; the subsequent discussion brings out many interesting details of particular cases. F. C. Smith emphasizes the great variability in causes and occurrences. Reno Sales cites the marked influence of the tight seams of clay in the fault-zones of crushed rock at Butte. The mineralization is sharply confined between them. F. L. Ransome brings out the case wherein mineralization in a vein may be later than a subsequent fault which crosses it and cuts it off with a tight gouge. An orebody might thus be apparently faulted yet have no prolongation beyond the break, even though its fault fissure continued. Max Boehmer cites a case from Leadville where a shoot was developed later than the fault which cuts it off. H. V. Winchell speaks of the following classes of shoots: 1. Original with the vein, or paragenetic; 2. Later than the vein, or postgenetic, including those formed by (*a*) Later internal movements, (*b*) Surface waters. Under the head of 2*b*, would come ordinary secondary enrichments which Irving had left out designedly. Hjalmar Sjögren sends some extremely interesting records of the decrease in value with depth of certain Swedish copper mines which had been worked for centuries. Other contributors have followed in 1909, and more may be expected, but even now the papers already available make very interesting reading.

In 1907 Waldemar Lindgren published a very interesting paper upon the relations of ore-deposition to physical conditions. Developing for ore-minerals somewhat the same ideas which had been applied to the minerals of the metamorphic rocks by F. Becke in Austria, i.e., their distribution in depth according to the pressure and heat required for their formation, Lindgren formulated a very interesting classification. The minerals were arranged in seven groups as follows and under each group the minerals were tabulated: 1. Persistent minerals; 2. Minerals of contact metamorphic deposits; 3. Minerals of deeper vein zones; 4. Minerals of middle and upper vein zones; 5. Minerals unstable in all vein zones; 6. Minerals in the zone of secondary sulphide enrichment; 7.

Minerals formed in oxidizing ground-water zones.[1] W. H. Emmons has taken up somewhat the same line of attack in 1908 and has carried it much further in a paper on the "Genetic Classification of Minerals."[2] Seven groups of minerals somewhat different from Lindgren's divisions are established as follows and under each very detailed tabulations of minerals are given. The seven groups are: 1. Cooling magmas; 2. Pegmatites; 3. Contact metamorphic zones; 4. Deep vein zones; 5. Moderate and shallow depths; 6. Sedimentary rocks far from intrusives; 7. Secondary enrichments; 8. Dynamic metamorphism. These associations are interesting and important. They make it possible to sum up in concrete form a vast amount of experience and they furnish topics of associations that may well be the subject of thoughtful reflection by observers in the field.

The subject of iron ores received much consideration during 1908. The wide discussion of the conservation of national resources directed attention especially to this subject and set in motion the gathering of estimates of reserves. A careful calculation of the future supplies in the United States has been prepared by C. W. Hayes of the U. S. Geological Survey and will be made public early in 1909. The culmination of all work of this kind will appear in 1910 in the volume upon the iron ore reserves of the world now in preparation by the authorities of the Eleventh International Geological Congress which will meet in the summer of 1910 in Stockholm. Meantime as indications of interest in the subject we find in both the National and the State Surveys papers that are forerunners. The New York Geological Survey has been carrying out a series of diamond drill borings to determine reserves of Clinton ore within the State's boundaries. Three main basins have been proved which contain some 600,000,000 tons of ore, considered by D. H. Newland to be of such richness, thickness and nearness to the surface as to be sometime utilized.[3] An earlier bulletin, No. 119, in the same series by D. H. Newland and J. F. Kemp reviews the iron ores in the Adirondack region. Both authors reach the conclusion that the largest deposits are basic segregations from the syenitic series of eruptive rocks, now known to form a goodly portion of the Adirondacks.

An extremely interesting and valuable paper has been prepared by C. K. Leith upon the iron ores of southwestern Utah, of which many reports have reached the technical press within recent years. Various views have been advanced regarding them, varying all the way from the early ones of J. S. Newberry that they were metamorphosed beds of limonite, to those of E. P. Jennings that they were intrusive dikes. Leith concludes that they were undoubtedly connected with the intrusive masses of andesite

[1] *Econ. Geol.*, II. 105.
[2] Ibid., III. 611.
[3] *Bull.* 123, N. Y. State Museum, by D. H. Newland and C. A. Hartnagel.

with which they were associated and from which they came in its cooling stages. They appear in contact zones of limestone, in fissures in the andesite, and in a breccia of overlying Cretaceous quartzite. An estimate of the quantity based on the superficial area exposed and a reasonable extension in depth gave 40,000,000 tons.[1]

In the East, A. C. Spencer has been investigating the "Magnetite Deposits of the Cornwall Type in Pennsylvania" and in *Bull.* 359 of the U. S. Geological Survey gives his conclusions. The huge deposits of earthy magnetite at Cornwall have been an object of much economic and scientific interest. Their association with dikes of diabase led J. P. Lesley and E. V. d'Invilliers to believe them old brown hematite beds, metamorphosed to magnetite by the dikes. Spencer considers them to be contact deposits developed in connection with the diabase intrusions. As the most probable method of origin he concludes that magmatic waters charged with iron introduced this metal into the shales and replaced them with ore.

In California iron ores have also received attention. Basil Prescott has described the magnetites of Shasta county. They all appear as contact deposits between intrusive diorite and its walls. The walls are limestone where the largest bodies are found, but smaller ones lie against granite, andesite and shale.[2] A different variety of iron ore, namely the brown hematites of the Great Valley in southeastern Pennsylvania, has been discussed by H. M. Chance who suggests a somewhat new explanation for them. The deposits specially treated form a belt in a series of slates lying on the old Cambrian quartzite and preceding the heavy blue limestones of this section. Earlier observers had referred them both to disseminated pyrite and to siderite. Dr. Chance considers them to be gossans of actual beds or bedlike deposits of pyrite and remarks the possibility of finding the pyrite in minable form below the weathered zone.[3] This conclusion has not gone unchallenged, since others still regard iron carbonate as the most likely source.

Contact deposits figure prominently again in *Bull.* 347 of the U. S. Geological Survey by Fred E. and Charles W. Wright upon the Ketchikan and Wrangell mining districts of Alaska. This type is specially developed upon Prince of Wales Island and presents the usual zones of lime-silicates at or near the borders of limestones and intrusive rocks. Besides these there are typical veins, breccia veins and impregnated bands or lodes in schists. The interesting deposits on the Kasaan peninsula of Prince of Wales Island are also taken up by Charles W. Wright in *Econ. Geol.*, III, 410.

Several important papers have dealt with ores in igneous rocks and

[1] *Bull.* 338, U. S. Geological Survey.
[2] *Econ. Geol.*, III, 465.
[3] *Trans.* A. I. M. E., 1908, pp 791-809; *Eng. and Min. Journ.*, LXXXVI, 408.

believed to be direct crystallizations from the magma. E. A. Ritter gives the details of one of more than ordinary interest at the Evergreen mine, in the northern part of Gilpin county, Colo. A dike of a peculiar rock like a quartz porphyry, but unusually high in soda, so that it contains minerals characterized by this oxide, carries chalcopyrite and bornite as orginal minerals. The metallic sulphides are included in the feldspar and quartz and certainly seem to antedate their hosts in time of formation. The paper appears in the *Trans.* A. I. M. E., Jan., 1908, p. 33. In the same volume on p. 311, J. B. Hastings describes a lake bottom near Antelope Springs, Park county, Colo., whose material seems to be derived from the neighboring granite, but which carries about 20c. to the ton in gold, so fine as to elude the pan. The gold is referred to the granite for its source. J. B. Harrison, the very active Director of Science and Agriculture in British Guiana, has also announced gold in the epidiorites and amphibolites of the Colony both being metamorphosed basic intrusives.[1] These two occurrences increase the list, now a fairly long one, of original gold in eruptive rocks.

In Canada the developments at Cobalt have naturally stimulated great interest in the discovery of other camps in the little explored districts to the north and west. The Montreal River region and a new camp called Gowganda have seen a rush of prospectors. Continued study has been given by Dr. W. G. Miller and his assistant Cyril Knight to the older camp so that we now have in Part II of the Report of the Ontario Bureau of Mines, for 1907, an excellent summary of our knowledge. The belief has been growing among observers that the silver and cobalt veins are end-products in the cooling of the diabase and that they were preceded by dikes of aplite. R. E. Hore has set this view forth in *Econ. Geol.*, III, 599. Convincing arguments for the Montreal River region are furnished by Alfred E. Barlow in a paper on the "Origin of the Silver of James Township, Montreal River Mining District" in the *Journ.* Can. Min. Inst., March, 1908. In connection with diabase intrusions of an acidic character and often having quartz, Dr. Barlow has found a series of dikes or veins, beginning with normal pegmatite, thence passing through pegmatite with more and more calcite, to veins with galena and chalcopyrite, and also into calcite veins with cobalt minerals and silver. Aside from the paper cited, the series of specimens is of more than ordinary interest. Regarding pegmatites in general, a very complete review of the literature and of the discussions of origin, has been prepared by J. B. Hastings and published in the *Trans.* A. I. M. E. for 1908, pp. 319–343. Mr. Hastings concludes that pegmatites are essentially coarse-grained silicious dikes, not involving great amounts of mineralizers at the outset; but that along their sides

[1] J. B. Harrison. "Geology of the Goldfields of the British Guiana." London, 1908, pp. 52. This work is a most valuable contribution to the local geology and petrography.

have afterward circulated pneumatolitic vapors which have contributed to the production of coarse crystallization. Finally highly heated silicious waters have brought in quartz and as the last stage various rarer elements have been introduced.

After an experience with the analyses of igneous rocks that is altogether unequalled, H. S. Washington has summed up the most important points of their composition in a very interesting paper on the "Distribution of the Elements in Igneous Rocks."[1] Not only are the more abundant ones treated, but the rare and unusual ones as well, so that many interesting and out-of-the-way generalizations are afforded.

A very curious fact in ore distribution is brought out by C. R. Keyes in a description of the Lake Valley mines of New Mexico, which forms the first paper in the volume of the A. I. M. E. just mentioned. These famous bodies of silver ore, which in their day furnished so many rare minerals, are in a series of 1045 ft. of sediments of which four-fifths are limestones. Yet the ore is limited to a single stratum of 25 ft. in thickness and of rather pure calcium carbonate. It is an interesting and striking case of selective precipitation by a special band of rock in a series. The ore is referred to a neighboring intrusive mass and its predominating haloid minerals are believed to be due to secondary reactions caused by descending waters containing salt derived from particles blown along in the dust of the desert.

A condensed and valuable summary of the deposits of the useful minerals in South Africa has been prepared and published by F. W. Voit in *Zeit. f. prakt. Geol.*, 1908, XVI, 137–145, and 191–217. After a brief review and tabular statement of the stratigraphy of South Africa, a scheme of classification of mineral deposits is given closely following the recent German textbooks of Beck and of Stelzner-Bergeat. The localities are then taken up under these heads. The most important single one is the gold of the Witwatersrand. Dr. Voit had already written of this elsewhere at length, but he summarizes his reasons for believing that the gold has been introduced in solution. It is interesting to find appearing in *Econ. Geol.* early in 1909 an article by J. W. Gregory, now of the University of Glasgow, which upholds with emphasis the origin of the gold by placer formation. Upon no great question of the deposition of the metals have the doctors differed more widely in diagnosis than over this one of the Transvaal banket.

In *Zeit. f. prakt. Geol.*, 1908, pp. 401–442, R. Delkeskamp gives a sketch of the progress of our knowledge regarding mineral springs. The various kinds of springs are classified and recent studies with regard to them are summarized. Students of ore-deposits will find in the paper not a little of interest. J. B. Hastings has discussed one phase of these subterranean

[1] *Trans.* A. I. M. E., 1908, pp. 809-839.

supplies, under the title "Volcanic Waters" in the *Trans.* A. I. M. E. for 1908, p. 345. Mr. Hastings' inquiry turns especially upon the waters imprisoned in sediments at the time of their deposition and the subsequent fate of these stores. In the possibility of primeval water or its elements, entering into the magmas in their origin as such, Mr. Hastings manifests less faith.

Two general books, the result of much reading and investigation and compiled with the aid of the Carnegie Institution, appeared during 1908. They are "Lead and Zinc in the United States" by W. R. Ingalls, and "A Treatise on Gold" by W. R. Crane. They cover the history of the mining industry in the metals mentioned, from the early settlement of the country and take up also occurrence and technical treatment. The U. S. Geological Survey has continued its excellent work in Alaska so that we have several bulletins bearing on the mineral resources of that country.[1] The general bulletin on "Contributions to Economic Geology in 1907" No. 340, gives a very interesting review of the work of the Survey in numerous lines. The papers are too numerous to mention in detail.

[1] *Bull.* 328, "Gold-placers of parts of the Seward Peninsula, Alaska," by Messrs. Collier, Hess, Smith and Brooks. *Bull.* 358, "Geology of the Seward Peninsula Tin Deposits," by Adolph Knopf. *Bull.* 337, "The Fairbanks and Rampart Quadrangles," by Messrs. Prindle, Hess and Covert.

NOTES ON THE PRACTICE IN MINING.

By Henry Lloyd Smyth.

The Courrières Disaster.

This terrible disaster[1], which involved a greater loss of life than any other in the history of mining, is described by M. Ch. E. Heurteau, in the *Annales des Mines* for October and November, 1907, numbers which were received in America early in 1908. This admirable paper relates with exceptional clearness the story of the disaster, and brings out several points in connection with it which are of general interest. Since the famous report of Mallard and Le Chatelier, in 1882, French engineers have been disposed to doubt whether the inflammation of coal dust alone, without fire-damp, was capable of propagating a severe and far-reaching explosion. The tragedy at Courrières has disposed of that doubt in the most impressive fashion, for this was a mine which had been entirely free from gas before the explosion and has remained so since. The paper of M. Heurteau makes it certain that no explosive agent except dust can possibly be considered as the cause of the disaster.

The large Courrières concession lies east of Lens in the Northern Basin. Eleven seams of coal in all are opened, of which the most important are the Cécile, Sainte-Barbe, Joséphine, and Marie, all occurring within a thickness of 250 ft. The other seams are less important, and are not always mined. The geological structure is extremely complicated. The main field of exploitation covers a broad, much faulted anticline, terminating to the south in an overturned syncline, in which the strata have not only been bent back over the anticline, but in consequence of a horizontal fault have also been shoved north across it. On the north side the anticline ends against an important fault, the Connétable.

The area affected by the explosion, covering nearly two miles east and west by one mile north and south, had been developed from three vertical shafts, Nos. 2, 3, and 4, No. 2 being the easternmost, and No. 4 the westernmost. Nos. 2 and 3 were hoisting shafts, while No. 4 was an air-shaft only. The general plan of development was to open the main levels 60 to 70 ft. apart. From each shaft on each level a cross-cut for a main haulage road was driven north and south across the territory

[1] *Annales des Mines*, Xième. Série, XII. 439.

through rock and coal seams, and from this the main gangways were turned off to the right and left in each seam as it was encountered. At the date of the disaster the principal working level was about 1100 ft. beneath the surface.

According to the general plan of ventilation the intake air passed down No. 3 shaft in the center of the ravaged district, and was drawn out at Nos. 2 and 4 on its borders. There was also a return ventilating current through No. 3 which drew air from a small area only in the neighborhood of the shaft.

The mine was free from gas and was worked with naked lights, except in openings in new ground where Wolf lamps were used. All new workings were tested for gas regularly every Sunday night. Gas had never been reported within the devastated area except on the 383-meter level in No. 4 shaft, where it was said to have been detected at various times in 1902, 1903, and 1905. On none of these occasions, however, was the report substantiated when an engineer was sent to verify it.

On the night of Tuesday, March 6, fire broke out south of No. 3 shaft in a raise on the Cécile vein from the 326-meter level to the 280-meter level. This was in the part of the mine ventilated by the split returning through No. 3 shaft, which was consequently filled with smoke. The fire was walled off from all active openings during the 7th, 8th, and 9th of March, but it was not possible to get behind it, to cut it off from an extensive area of old workings lying to the west and north.

The explosion occurred on March 10 in the early morning between 6:30 and 6:45. The effect was most disastrous at No. 3 shaft, which was wrecked and blocked with broken timber. Heurteau gives a vivid and very clear description of the way in which the work of rescue was undertaken and the general situation handled during the first two days. First, the down current of air was re-established through No. 3 shaft, and most determined efforts were made to get into the devastated territory by means of it. These efforts were frustrated by the mat of timber which filled the shaft below 170 m. Next, while only very slow progress was being made at No. 3, the devastated area was approached from the outside from the other working shafts, and numbers of men were rescued. Rescuing parties even succeeded in reaching No. 3 shaft from the No. 2 side, on the night of the 10th, and on the 11th, and explored its neighborhood on the 280-, 303- and 326-meter levels, as widely as the fallen ground and bad air permitted. At 1 a.m. on the 12th all hope was given up of saving any more lives and the air was reversed, No. 3 shaft was converted into an upcast, and systematic exploration of the devastated territory for repairs and the recovery of bodies was undertaken from No. 2. In this the Westphalian volunteers with the

Shamrock apparatus were especially helpful. On March 15, before much progress had been made with this program, fire was discovered in the north workings west of No. 2 on the 340-meter level, and from that time on every resource at hand was enlisted in subduing it, and exploration was dropped. Finally, on March 30, nearly three weeks after the disaster, 13 survivors, who had managed to keep themselves alive in the territory south of No. 3, appeared at No. 2 shaft. Search was then resumed and was pushed as rapidly as possible, with the result that another man came out alive at No. 4 shaft on April 4, twenty-five days after the disaster.

The authorities in charge have been severely and widely criticised for having given up the work of rescue too early, for having failed to avail themselves of the detailed knowledge of the mines possessed by the bosses and miners, and for having reversed the air current on the 12th. As a result the Government appointed a commission, the President of which was M. Carnot, Inspector General of Mines, to examine into the whole matter. The majority report of this commission, in which the two labor members did not concur, exonerates the authorities from blame, but it must be said that it is far from convincing.

The points of greatest interest in Heurteau's paper are those that relate to the tracing of the explosion back to its source, and to the explanation of its origin. In brief, the explosion is clearly proved to have originated at some point in a triangular area on the 326-meter level, to the north of No. 3 shaft. From this area it evidently spread outward in all directions. The direction of propagation was determined by the usual dynamic effects, namely, the overturning of timber, the blowing out of stoppings, doors, and regulators, the crushing in of cars, etc. These indications were reinforced by deposits of coke and dust on timber and other projecting surfaces. It was generally found (though there were exceptions) that a thin coating of coke was deposited on the lee side, while on the side toward the blast there was frequently dust partly coked, in the form of a prism with the sharp or bevelled edge facing the blast. Evidently these deposits were made by dust driven directly against the timber by the blast, and left there as it divided to pass the obstruction. Atkinson's conclusions as to the position of both coke crusts and dust prisms are therefore verified, though enough exceptions were noted to prevent them, taken alone, from being of decisive value in indicating the direction of propagation.

Within the focal area already referred to, the point of origin of the explosion was most probably a pair of headings which were being driven eastward on the Joséphine vein above the 326-meter level. In the south or main heading an explosion of extraordinary violence had

taken place which had been propagated outward through both headings into the main roads on the 326-meter level. Whether the explosion, which took place here, was the original explosion, or whether it was a secondary one resulting from a primary explosion elsewhere, is possibly open to question. It can only be said that nowhere else in the focal area was there any indication of an initial disturbance. It is therefore altogether likely that this south heading, known as the Lecoeuvre road, was the real point of origin.

As to the cause of the explosion in the Lecoeuvre road no certain conclusion can be reached. In spite of the fact that the north heading follows the Connétable fault, which elsewhere in the district has occasionally yielded pockets of gas, there is no apparent reason for believing that a minor gas explosion initiated the main explosion. This heading was regularly examined for gas and none had ever been found in it before the disaster, nor has any been found since. Perhaps the most plausible suggestion is that it resulted from an explosion of powder caused by picking out a missed shot. Of the four men at work in this heading one was found lying against the face partly covered by a fall of coal. Two others were lying on top of this coal. A pick and lamp were found just below a hole, which had not broken to the bottom, around which were pick marks. The partly dismembered body of the fourth man was lying 20 m. from the breast with the severed limbs some distance beyond. The coal on the sides was coked up to the face, and the bodies were burned, showing that the heading had been filled with flame. The return air pipe, which served to ventilate the face, was wrecked in extraordinary fashion. Two lengths were blown to small pieces. Other lengths were split open from the inside. These facts could perhaps be explained by supposing that, while one man was picking out the missed shot and the other two were working at his right, the fourth man was sitting on the air-pipe, 20 m. from the face, engaged in priming a cartridge. The explosion of the missed shot set off the powder in his hand, and ignited the cloud of dust so stirred up. The chief difficulty with this explanation is that all the powder issued to this contract on the morning of the 10th was found unexploded in the heading.

The investigation showed that the fire, which started on the 6th in the Cécile vein south of No. 3 shaft, and which at first was universally believed to have caused the explosion, had nothing whatever to do with it. Not only was there the clearest possible evidence that the point of origin was far removed from this spot, but it was found that the fire had been confined to a few sets of timber, and had not ignited or coked the coal. All told not more than 6 or 7 cu.m. of timber, say 3000 ft. b.m., had been

consumed. The imperfect combustion of this, extending over three days, could not possibly have yielded enough inflammable gas to have accounted for the explosion.

The general conclusions from M. Heurteau's study were these:

1. Beyond the point of origin the explosion was a dust explosion pure and simple, propagated by the progressive distillation and ignition of gas yielded by coal dust. The coke and dust in the devastated galleries was shown to have lost from one-third to one-half of its original volatile matter, the loss being greater in the crusts adhering to the sides than in the bottom dust.

2. The distribution of the explosion through the workings and its violence were extremely irregular. It can be said in general that it swept through the main haulage roads, the cross-cuts, and from them spread into the main gangways in the various veins. The wide openings suffered most. Many of the points at which the explosion ceased were those at which the floor and walls ceased to be dusty, where the gangway was lined with masonry, or where the opening passed into rock, or entered wet ground.

3. In its progress over the devastated area the force of the explosion exhibited a certain rhythm, almost dying out in places, and again starting up with great violence. The points of renewed activity were points at which fresh currents of air were encountered.

Coal Dust Explosions in America.

The terrible disasters in coal mines in 1907, particularly at the Monongah and the Darr mines in December, by which nearly 600 men lost their lives, thoroughly aroused engineers and operators in this country to the importance of dust as a chief if not the sole factor in such disasters. During the year 1908 many important papers on the subject of dust appeared in mining periodicals, which set forth interesting facts and opinions, and provoked much valuable discussion.

The Monongah Disaster.—It will be remembered that the mines affected were Nos. 6 and 8 of the Fairmont Coal Company, situated on the west bank of the west branch of the Monongahela river, near Fairmont, West Virginia. These mines were connected along one entry only. Shortly after 10 A. M., Dec. 6, part of a trip—15 loaded cars—broke away at the head of the slope at No. 6 mine and ran back into the mine. A few seconds later an explosion occurred in both mines, which did little damage in No. 6 slope, but was very violent at the No. 8 entries, wrecking the fan and the surface equipment in the neighborhood. That these two mines, which were thoroughly modern in lay-out and equipment, and had been designed to be models in every respect,

should be the scene of the greatest disaster in the history of mining in America, was very discouraging, and every effort was made to discover the cause of the explosion, and to find out how similar explosions might be avoided. The mine was visited by committees of mine inspectors from Pennsylvania and Ohio, as well as from West Virginia, and the company itself employed experts to make independent examinations. Also a great deal of testimony was taken by the coroner's jury in regard to the general operating conditions as well as the facts which bore directly on the cause of the explosion.

1. Cause of the Explosion. These investigations resulted in the usual difference of opinion as to the point at which the explosion originated and its cause. But practically all agreed that it started in No. 8 mine and not in No. 6, and that it was not caused by the runaway trip. This appears to have been merely an extraordinary coincidence, like the fire in the Cécile vein at Courrières. The representatives of the company were of the opinion that the primary cause was an accidental explosion of powder, while the Pennsylvania, Ohio, and most of the West Virginia inspectors believed that it resulted from a blown-out shot. But whatever the original cause all were agreed that the explosion was propagated by dust.

2. General Underground Conditions. The evidence showed that the ventilation of the mines was excellent, the air-ways well arranged, and the fans of ample capacity. Some gas was present; indeed, on the morning of the explosion it was reported in several places in No. 6 mine. Furthermore, the mines were very dry and dusty. It was the practice to sprinkle the main entries at irregular intervals, but little dust was removed from them, while the dust in the rooms was not dealt with at all. Both mines were worked with naked lights. The coal was undercut by electrically-operated chain machines.

There was abundant evidence of extreme recklessness in blasting. Black powder was taken into the mine in bulk and was used without effective restriction as to quantity of charge, and apparently holes were tamped with coal dust. Shooting from the solid was practised. Holes were found badly placed, drilled too deep, and evidently too heavily charged. In a word, there was clear proof of general ignorance and carelessness on the part of the men, and either of incredible ignorance or else lamentable shiftlessness and lack of supervision on the part of the management in this department of underground work.

The Darr Mine Disaster.—On Dec. 19, 1907, at 11:30 A. M., an explosion occurred at the Darr mine of the Pittsburgh Coal Company, situated on the Youghiogheny river, 42 miles south of Pittsburgh, through which 233 men lost their lives.

This disaster, following so closely on that at the Monongah mine, attracted great attention. Investigations were made by the bituminous mine inspectors of Pennsylvania, by E. J. Taylor, chief engineer of the Pittsburgh Coal Company, and by a committee of experts selected by the company. The reports of all these investigations have been published.

1. Cause of the Explosion. The investigations did not result in unanimity of opinion as to the point of origin of the explosion and its cause. The inspectors believed that it originated either in a blown-out shot or through ignition of gas. But whatever the original cause it swept the mine in consequence of the dust everywhere present. Mr. Taylor came to the conclusion that the cause of the explosion was a gas explosion brought about by men going with naked lights into a place known to be dangerous. In this opinion he was sustained by the outside experts called in by the company.

2. General Underground Conditions. From the reports it appears that the mine was moderately dusty and somewhat gaseous. The entries were sprinkled daily, but not the rooms. Safety lamps were used in advance headings and in pillar-drawing, but not in the rooms. The coal was under-cut by electric chain-machines and blasted with black powder.

According to the mine inspectors the general underground conditions were bad. Their report implies that the ventilation was insufficient; that safety explosives ought to have been used; that the dust should have been thoroughly wet and laid; that locked safety lamps should have been used; that competent shot firers should have been employed, and rigid dsicipline maintained. These are called well-known safeguards. If they had been employed the explosion would not have occurred.

The reports of Mr. Taylor and of the committee of experts as to underground conditions exonerate the company from all blame.

Coal Dust.

The danger from coal dust being universally admitted, how is this danger to be overcome? Among the large number of papers which during 1908 have dealt with one aspect or another of this question the most specific and important are the report of the Pennsylvania inspectors on the Monongah disaster, papers by Prof. H. M. Payne[1] of Morgantown and by Frank Haas[2] of the Fairmont Coal Company, the report of the foreign experts brought over by the Department of the Interior to investigate the condition of the coal mines of this country[3], and an

[1] *Eng. and Min. Journ.*, July 4, 1908.
[2] Ibid., Oct. 24, 1908.
[3] *Bull.* 369, U. S. Geol. Survey.

open letter from Mr. Page[1] to the coal-mine operators of West Virginia. These publications bring out clearly the main difficulties in the way of dealing with the dust problem—difficulties which are intensified in the case of gaseous mines by the complementary character of the dangers from gas and dust, for an air-current adequate to keep the faces swept clean of gas may increase the danger from dust by holding in suspension particles that otherwise would settle.

The principal points brought out in these papers may conveniently be considered under the following heads: (1) Why dust is dangerous. (2) Prevention or diminution of dust. (3) Wetting, dilution, and removal. (4) Ignition.

1. *Why Dust Is Dangerous.*—It is well known, other things being equal, that dust is dangerous in proportion to its fineness. While even coarse dust, when stirred up and heated by the blast of an explosion, will give off inflammable gas and so propagate an explosion, it is not capable of initiating it. Payne considers that the only dust which is initially dangerous is that which is fine enough to remain suspended in gentle air currents, and he draws the line at 100-mesh, calling all coarser material "slack," and restricting the name dust to the finer particles.

This fine dust Payne regards as having peculiar explosive properties, due partly to the fact that it gives off its occluded gas more readily than coarse, and also because it may have the property of condensing atmospheric oxygen on its surface, which property Wabner states is possessed by some coals. Bedson and McConnell[2] also have concluded that coal-dust long exposed to the air absorbs oxygen, and is thereby rendered more easily inflammable. Furthermore, the area of contact of fine dust with the air in proportion to its volume is greater than that of coarse dust. It would seem to be Payne's conception that a minute particle of such dust is like a grain of powder, inasmuch as a combustible substance and the oxygen to burn it are brought into exceedingly intimate physical (though not molecular) contact. In Payne's opinion such minute grains constitute a true explosive, capable of ignition through shock, compression, or heat.

This conception has been strongly combated by Haas, who contends that no matter how finely divided coal dust may be it is not explosive, but only the gases which are distilled from it.

2. *Prevention or Diminution of Dust.*—The fine dust is made or gets into the air-current in various ways. At the very outset the surface plant, as the Foreign Commissioners urge, ought to be so placed and designed as to prevent the fine dust rising from the screens at the tipple

[1] *Eng. and Min. Journ.*, Dec. 5, 1908.
[2] *Trans.*, Inst. Min. Eng., VII, 32-53.

from being taken up by the in-going air and carried back into the mine. Under-cutting, blasting, loading the coal, and its transportation underground are all unavoidable sources of dust. The last-named source particularly is of growing importance as operations extend and the speed of haulage increases.

There is an impression that machine mining makes more dust than under-cutting by hand, and that some types of machines make more dust than other types. Indeed, the Pennsylvania inspectors in their Monongah report have gone so far as to state their conviction that chain machines are a menace underground owing to the fine dust which they produce, and mainly, but not solely, on that ground they have recommended that their use be discontinued. It should be said that this impression does not appear to be sustained by the experiments of B. F. Jones[1] and C. E. Scott.[2] Mr. Jones has compared the percentage of dust produced in under-cutting by hand with that produced by pick machines and chain machines, respectively, in the Pittsburgh seam in Westmoreland county, Penn., weighing as dust all material finer than 40-mesh. He concluded that the chain machine made less fine cuttings than the puncher, and not much more than hand-mining. Mr. Scott at Fairmont found that the percentage of fines which would pass an 80-mesh screen was practically the same with the two mechanical cutters, a slight advantage, if any, resting with the puncher. It might be contended, however, that these experiments are not conclusive, or indeed are irrelevant, since neither took account of the finest dust which remains suspended in the air, and is undoubtedly the chief source of danger.

Measures tending somewhat to diminish the quantity of fine dust produced are wetting down the sides, floor, and face of the room before blasting, and keeping car-boxes tight, and the main roads and tracks clear of coarser coal, the further abrasion of which would produce dust. But such measures are very mildly palliative.

3. *Wetting, Dilution, and Removal.*—After dust has once formed (and its formation is unavoidable) how can it be dealt with? The increasing dryness of the air-current, as it passes underground, in consequence of its higher temperature and greater density (in deep mines), and its velocity are factors in producing fine dust, and among the chief factors in rendering it dangerous when made. Indeed, Mr. C. Scholz[3] has attempted to show that the anticyclones of winter with their cold and dry air are as significant in dust explosions as low barometric pressures, through drawing gas from the goaves, have been supposed to

[1] *Mines and Minerals*, March, 1908.
[2] Ibid., May, 1908.
[3] *Trans.*, A. I. M. E., Oct., 1908.

be in gas explosions. This is still another example of the complementary character of the dangers from gas and dust.

Wetting is the obvious remedy. But how shall water be applied? The Pennsylvania inspectors in their Monongah report have recommended that dusty mines be provided with a water system under sufficient pressure to lay the dust effectively, and particularly that the dust in rooms and headings be thoroughly wet down, using water equal to one-half the weight of the dust. This means a pipe system extended into every room, and D. Harrington has shown[1] that this is not only practicable, but is now practised with success in coal mines in Utah at a small operating cost. On the other hand Messrs. Payne, Haas, and Page have pointed out that it is impossible to moisten fine dust to the point of safety, or indeed to make it wet at all by the direct application of water, while merely wetting down the floors is clearly insufficient to meet the requirements of the case. Mr. Page furthermore points out the added insecurity to the roof which results from wetting, and reminds us that it is falls of roof after all which cause the majority of the accidents in coal mines.

Adding moisture directly to the air to the point of saturation or supersaturation would seem to be a promising direction for investigation as it permits that longer and more intimate contact which, as Mr. Haas has said, appears to be necessary for the wetting of coal dust. At all events it would prevent the air from robbing the coal of the moisture which it already possesses. Of the methods which have been tried or suggested—sprays at intervals along the main roads, or exhaust or live steam at the intake—the latter theoretically would seem to have most to commend it. But this also has been tried in Utah, and while according to Mr. Harrington[2] it is most effective in keeping the faces moist and in bringing down the suspended dust, it has serious drawbacks. The air heated by the steam and saturated with moisture is very unpleasant to work in, and indeed impossible to work in effectively; the atmosphere is thick and murky; and most important, its effect on the roof appears to be more harmful than water directly applied.

Coal dust may also be diluted with rock dust in the main roads. Experience at Courrières and experiments at the British station at Altofts[3] have indicated that the flame of dust explosions may be arrested in sufficiently long stretches of main road thus treated. Also it should not be forgotten that promising experiments have been made in both England and America[4] looking to the cementation of floor dust by sprinkling it with a deliquescent salt, such as calcium chloride.

[1] *Mines and Minerals*, Oct., 1908.
[2] Ibid., Oct., 1908, p. 102.
[3] Ibid., Dec., 1908, p. 235.
[4] Ibid., Dec., 1908, p. 216.

There is general agreement that so dangerous a substance as coal dust ought to be taken out of the mine. The Pennsylvania inspectors have recommended that, after thorough sprinkling, the dust in rooms and headings be loaded out of the mine. In this they are followed by the West Virginia inspectors. The foreign commissioners have recommended the removal of dust from the main roads. This is not so formidable an undertaking as might be supposed, if we may credit the experiments of H. M. Hall[1] as to the rate at which dust is deposited. On this point he says, "The common estimate of the quantity of dust deposited day by day far exceeds the facts; and it follows that, when once the roads have been made thoroughly clean, to keep them so involves no insuperable difficulty."

The fine dust from the sides and roof, behind timber, and in cracks is harder to deal with. It is altogether natural that attempts should be made to deal with it by means of the vacuum cleaner, and it is reported[2] that machines on this principle, operated by compressed air or electricity, are already doing good work at a cost below that of spraying in certain British collieries and at Courrières.

4. *Ignition of Dust.*—The experiments of the Prussian and Austrian Fire-damp Commissioners proved that suspended coal dust served to render small amounts of gas inflammable, even when present in quantities much below the danger limit, and later experiments have proved that dust alone without gas under certain circumstances may ignite and burn with explosive rapidity. No doubt there is danger that any flame may ignite gas in a dusty atmosphere. To ignite suspended dust alone without gas is more difficult, but may be accomplished as the Neunkirchen experiments proved by the long flame of blown-out or overcharged shots, if not in other ways. Indeed, Hughes states[3] that some coal dusts have been ignited by heating in a current of air at 140 deg. C.

The rapidly extending use of electricity underground for haulage, lighting, coal-cutting, etc., undeniably involves risk of firing gas or dust through the sparking or heating of motors, broken wires and lamps, defective bonding and insulation, etc. The Pennsylvania inspectors in their Monongah report made the radical recommendation that the use of electricity underground be reduced to a minimum. The foreign experts did not go so far, but would permit no live wire in that part of the mine in which 2 per cent. of gas had been found. And they recommended also that all working places in mines having gas in dan-

[1] *Mines and Minerals*, Dec., 1908, p. 223.
[2] Ibid., June, 1908, p. 534.
[3] "Text-book of Coal Mining," pp. 319, 320.

gerous quantities be examined for gas before any electric machine be operated in them.

The other source of flame are naked lights and blasting. The Pennsylvania inspectors have recommended that in mines producing gas locked safety lamps be exclusively used. The visiting experts are more specific in their language: "In any mine, where as much as 2 per cent. of gas can be detected by suitable methods, only locked safety lamps of an approved type should be used so long as such conditions exist or are likely to recur."

The main danger of ignition comes from blasting, and many suggestions and recommendations have been made looking to the reduction of that danger. Most of these relate to the prevention of blown-out shots. The Pennsylvania inspectors would prohibit by law shooting out of the solid and would blast down thick coals (such as the Pittsburgh seam at Fairmont) in two benches. The visiting experts on this point specifically recommend that the depth of the hole be less than the depth of the cut, and that over-charging be avoided. In the way of more general precautions they add a warning, which ought to be unnecessary, against tamping with coal dust, against successive firing in the same place before the dust has had a chance to settle and an examination can be made, and against blasting in a mine producing gas until a previous examination has shown the absence of 2 per cent. of gas within 60 ft. of the proposed shot. They also recommend the use of safety explosives, which they would call "permissible," and which should be so designated on a list to be issued by the Government. The use of such explosives has also been recommended by the Pennsylvania inspectors. There is evidently but one opinion in regard to the use of black powder in gaseous coal mines, and that is that it ought to be prohibited in this country as it is in other civilized countries.

The most important recommendations relate to improvement in methods of charging and blasting, and in the supervision of this work. The Pennsylvania inspectors recommended that it be turned over to a special organization of competent shot firers. They also recommended that in gaseous or dusty mines shots should be fired at night unless permission is given by the inspectors to shoot in the daytime. The visiting experts go much farther: "Believing that such will be one of the greatest advances which can be made in safeguarding the lives of the miners, we recommend the adoption of a system of electric shot firing in all mines, where practicable, by which all shots in the mine, or in each ventilation district of the mine, may be fired simultaneously at a time when all miners and other employees are out of the mine." These recommendations do not involve anything which is not entirely practi-

cable. Indeed, simultaneous firing by electricity throughout a mine is already practised at the mines of the Utah Fuel Company. Mr. D. Harrington[1] has described in detail the equipment and routine for doing this. The results, according to Mr. Harrington, are entirely satisfactory, and no serious disadvantages have developed during the five years in which the system has been in operation.

These reports and papers have brought out clearly the fact that dust is not only more dangerous than gas, but is much more difficult to deal with. The directions in which greater safety is to be attained are clearly these: The compulsory use of safety lamps in gaseous mines; the compulsory use of the less dangerous explosives; turning over the work of charging holes and blasting to a special department, and simultaneous firing by electricity. We may confidently expect that progress along these lines will be made for the reason, if for no other, that disasters like that at Monongah are very destructive to property as well as to life. If the economic loss resulting from the loss of life in disasters in coal mines could be shifted from the shoulders of individual families or of the community in general to small sections of the coal mining industry, it might confidently be expected, human nature being what it is, that the progress of improvement would be greatly accelerated.

Air-Hammer Drills.

Small drills for stoping and raising, especially air-hammer drills, continued to attract much attention during the year 1908, and important progress was made in their development. A great deal of interest was manifested in the stope drill competition held at Johannesburg in December, 1907. This has been described[2] by Professor Orr, one of the judges of the contest, and has been commented on during the year by several writers in technical journals, especially by E. M. Weston and G. E. Wolcott.[3]

It will be remembered that the competition was won by one of the two air-hammer drills which lasted to the end, namely the Gordon, a $1\frac{5}{8}$-in. machine with a transverse spool valve. During this contest (the holes being mainly horizontal and down holes) it employed hollow steel and a water-jet to clean the hole, and was mounted on a stoping bar. The competition was decided purely on results, that is to say, on footage drilled in a given time, irrespective of air-consumption, which was not measured, and of size of bit, which was left to the contestants, except that the minimum diameter of the finishing bit was fixed. But

[1] *Mines and Minerals*, Aug., 1908, p. 38.
[2] *Proc.*, Transvaal Inst. Mech. Eng., Jan. 11, 1908.
[3] *Eng. and Min. Journ.*, March 7, 1908; May 16, 1908.

even as regards air-consumption in Professor Orr's opinion the advantage lay with the Gordon, his impression being based on a comparison of cylinder volumes, and inferences as to the number of strokes per minute. The governing conditions of the contest, namely, the preponderance of down-holes and air-pressures of 50 and 60 lb., were selected in order to reproduce as closely as possible the usual underground conditions on the Rand.

Mr. Weston has pointed out that the effort to find or develop a small drill to take the place of hand-drilling in stopes is by no means new in South Africa. The reasons why all previous attempts have failed are, first, unintelligent design of the small drills, which ordinarily have been merely dwarf reproductions of piston machines of standard types, and in practice made with some parts too light; and next, the peculiar organization of underground labor. The white contractor or boss will not consent to enlarge his area of supervision to the extent necessary to make the substitution of the small drills for standard-size drills or hand-labor economical.

It is well known that the general operating conditions on the Rand are unfavorable to small drills and especially to air-hammer drills. The gold-bearing conglomerates are broken generally in underhand stopes with down-holes. The air-hammer drill must therefore use hollow steel for an air- or water-jet and must be mounted on a stoping bar to the partial sacrifice of one of its greatest advantages, its mobility. The general labor conditions also, as has just been said, are against it. And lastly, the absurdly low air-pressures—40 to 50 lb.—which are the rule in that district, are far below the pressure necessary for good work with the air-hammer drill. Altogether it seems questionable whether air-hammer drills will largely replace hand-drilling on the Rand until back-stoping is substituted for underhand stoping. Working the blocks of ground from the bottom up would have an important collateral advantage, as Mr. Denny has pointed out, namely, it would permit a more effective sorting out of waste rock underground, to the probable improvement of financial results.

The conditions on the Rand have been compared by Mr. Weston with those at Cripple Creek, and on the use of air-hammer drills in this district G. E. Wolcott also has given interesting particulars[1]. At Cripple Creek there are only two large companies which do not use air-hammer drills for back-stoping. The first drills of this type were employed for block-holing and for work in narrow stopes. Now they are made heavier, and are used with the stoping bar, but are still light enough to be handled by one man. The rose-bit is rapidly disappearing,

[1] *Eng. and Min. Journ.*, May 16, 1908; Feb. 15, 1908.

being supplanted by the chisel or bull-bit, and the cross-bit with straight or curved edge.

Mr. Wolcott repeats that the proper field for air-hammer drills is upper holes; therefore they are useless for drifting. They will not drill down-holes without a jet of water or air to keep the hole clean, and this requires hollow steel. The air-jet raises an insufferable cloud of dust, and therefore the water-jet alone would seem to be practicable. In comparison with the piston drill, Mr. Wolcott states that the air-hammer drill makes about double the footage per shift in drilling upper holes, but if the ground is wet or sticky its superiority is somewhat diminished.

A. Delmar has described[1] a competitive test under working conditions of an air-hammer and a piston drill, which clearly proved the superiority of the former for raising. That superiority he attributes to the smaller volume which it has to cut. For example, the air-hammer drill takes ⅞-in. grooved steel, and if the successive bits are reduced by ¼-in. the area of the bottom of the hole is not more than two-thirds that of a hole drilled with a 2¼-in. drill using 1-in. steel. Therefore the volume of rock cut is less than half the volume cut by the piston drill. The smaller holes are not only drilled faster, but they are in themselves more desirable because they avoid waste of powder. Generally speaking, holes of large diameter are unnecessary in stoping, which of course is true only of free-breaking ground.

In a specific comparison of the work of the two drills (an air-hammer and a 2¼-in. piston) in raising, Mr. Delmar states that the total labor cost was practically twice as great with the piston drill as with the air-hammer drill, not only because the latter was at work a larger proportion of the time, but also because it actually cut faster. The hammer drill finished its round of thirteen 4-ft. holes in less than six hours, while the piston drill could not complete the round in an eight-hour shift. The progress of the raise with the piston drill was 1 1/7 ft. a day, while with the hammer drill it was 1 4/7 ft. a day. The time taken in setting timber was the same in the two cases, which of course somewhat masked the real superiority of the air-hammer drill. The cost of repairs for the air-hammer drill was very low. The air-pressure in this competition was probably about 85 lb. at the drill.

As regards the general lines of development of air-hammer drills it may be said that the tendency during 1908 was to make them stronger, by increasing their weight, and by putting better material into them. For example, the drills now on the market are from 10 to 15 lb. heavier than those of two years ago. Experience has shown that the early

[1] *Min. and Sci. Press*, Feb. 1, 1908.

models were too light, and would not stand the wear and tear of work in hard rock.

Another serious defect of these drills, when they were first brought out, was that dust or gritty water easily found its way into the cylinder through the chuck, and rapidly cut out the moving parts, causing the valves to leak and the hammer to stick. Now great pains are taken to prevent the dust from getting inside, this end being attained in some of the late models by turning the exhaust against the falling stream of dust. This device renders washers or collars on the drill steel unnecessary.

It is not too much to say that the air-hammer drill has completely demonstrated its usefulness in its special field, and that within that field it constitutes one of the most important improvements made in many years in a department of mining in which improvement comes very slowly, namely, in underground work. Its special field is undoubtedly the putting in of upper holes drilled dry, and therefore it is especially adapted to raising and back-stoping. For small raises and back-stoping in narrow veins there can be no question of its superiority to the piston drill. In raising the speed is doubled and the costs cut practically in two. In stoping also the results are substantial although not so impressive. But in large raises and stopes in very hard rock or ore, experience in the Lake Superior country, at least, seems to show that these drills cannot hold their own with piston drills nor do they do as well in wet ground. They are best adapted to dry rock or ore, of medium hardness. The principal reason why they will not break ground in large raises as economically as piston drills seems to be the small size of the hole,—it will not take enough powder to do the work. In comparison with a 3¼-in. machine the number of holes necessary to break a given volume of hard ground is practically doubled, although it must be said that the speed of drilling also is nearly twice that of the large drill.

Shaft-Sinking.

In 1908 the literature of shaft-sinking was decidedly meager in comparison with that of 1907, in which not only several interesting short papers appeared, but also two translations into English of Riemers' "Shaft-Sinking Under Difficult Conditions." The interest which nearly all mining engineers feel in this subject is probably mainly divided between two of its aspects, namely, the overcoming of the difficulties arising from heavy flows of water or from running ground, and the methods by which the speed of sinking in rock may be increased without unduly increasing costs. Europe has led the way in the development

and standardization of methods for dealing with water and heavy ground, because it has in large measure the difficulties to be overcome. Many of the important coal basins and salt deposits of Germany, France, and Belgium as well as of Great Britain, are in part overlaid by thick blankets of water-bearing or soft Tertiary and Cretaceous strata, which have to be passed through in order that the valuable deposits beneath them may become available. So several processes have been devised and perfected for dealing with the variously modified conditions which these present,—the Kind-Chaudron for sinking in firm rock against heavy inflows of water; the German system of drop-shafts for sinking in easily dregeable clays and sands; the Poetsch or freezing process, applicable alike to watery or yielding ground, and to all depths within which stand-pipe can be put down; and the Triger or pneumatic process, which is limited by the tolerance of the human body to air-pressure to an extreme depth of about one hundred feet below water-level.

That the art of sinking against a heavy flow of water has not come to a standstill in Europe is proved by the successful application of a new method,—cementation,—to the sinking of an air-shaft through the Chalk at Liévin in the northern coal-basin of France, in the spring and summer of 1906. The description of this sinking by M. L. Morin[1] is an important contribution to the literature on this subject. An excellent précis in English of this paper is to be found in *Eng. and Min. Journ.*, Aug. 1, 1908, and its details therefore need not be repeated here. The conditions in brief were these: The water-level was 14 m. below the surface, and the water-bearing beds, which were of firm chalk, cut by minute fissures, striking in various directions and dipping at various angles, extended to a depth of 55 m. It was necessary therefore to pass 41 m. of water-bearing material. The shaft, which was of circular section, 6 m. in diameter, was sunk in the ordinary way and lined to a depth of 32 m., or 18 m. below water-level. At this point the inflow reached 1100 gal. a minute and was increasing at the rate of 360 gal. a minute for each meter of descent. Therefore it was clearly necessary to adopt some other method. After due consideration cementation was decided on in preference to freezing as being cheaper. It was carried out by a method suggested in 1905 by M. Duvivier, namely, by attaching a water-tight bottom to the shaft lining already in place in order to hold the water at the 32-meter level, and by drilling through it to the impervious basement a series of holes, which were then cemented. Into each hole, as soon as it was drilled, a thin grout of portland cement, consisting of a 5 per cent. mixture, was fed at the surface, and was carried into the rock by the slight pressure due to the head of the column

[1] *Annales des Mines*, Xième. Série, Vol. XII, p. 493

above the water-level. When the cracks were so filled that the column no longer settled, another hole was drilled, and the process repeated. In all five holes, unequally spaced on a circle four feet from the walls of the shaft, were drilled and cemented. After the cement had set, the bottom was taken off and sinking resumed, the inflow of water being cut down to 12 gal. a minute. The shaft was carried down without difficulty for an additional 20 m. but against increasing water, which finally reached 650 gal. a minute. At this point the bottom was again put on, and the ground again cemented, after which the impervious bed was reached without further trouble. A valuable feature of M. Morin's paper is his comparison of the cost of cementation with that of freezing, concerning which it is difficult for us here in America to get reliable figures. A firm of contractors estimated the cost of freezing to a depth of 98 m. at 2300 francs a meter (or about $140 a foot) exclusive of excavation and lining. The cost of cementation alone, exclusive of pumping, distributed over the 23 m. through which it was necessary, was $55 a foot.

Cementation has been employed in at least one case in America within the last few years, but none of the other European methods, except the pneumatic method, has taken root in this country (although the freezing process has been used in one or two instances), simply because none of our great mineral fields has presented this class of difficulties in any important degree. It is true that in America it has frequently been necessary to get down through moderate thicknesses of quicksand or gravel yielding great flows of water, either in the glacial sand-plains of the northwest or in the alluvial plains of rivers. The usual way of dealing with these conditions has been to sink a heavy timber drop-shaft, within and under which the ground was excavated, and to lower the water by pumping, a procedure always attended by the drawing in of the fine and loose material under the shoe, and frequently by the formation of a crater around the shaft, with the impairment of its alinement and the contraction of its section. In this way in one famous case in the Lake Superior region a depth of more than 150 ft. of fine sand was successfully passed, of which more than 100 ft. were below the original water-level.

The tendency in recent years has been to construct drop-shafts of reinforced concrete instead of timber, thus obtaining a stronger and more rigid structure, as well as one impervious to water. Of recent sinkings with lowered water-level, none is more interesting than that of the Woodward shaft, which was put down for the D. L. & W. R. R. in the flats of the Susquehanna river at Wilkesbarre in 1908.[1] At the site

[1] *Eng. News*, Sept. 24, 1908

of the shaft the water-level was some 15 ft. below the surface, and the depth to ledge about 79 ft. The upper 50 ft. consisted of sand and gravel, 35 ft. of which were water-bearing, below which was clay. The sinking in itself therefore presented no special difficulty.

The heavily reinforced concrete drop-shaft or caisson was of great size, namely, 48 ft. 6 in. by 14 ft. inside at the bottom, the rectangle being rounded at the corners. On the outside the walls were vertical, but on the inside they were drawn in in successive stages, the thickness of the side-wall being 7 ft. at the bottom and 2 ft. 8 in. at the top. Two transverse or dividing walls stiffened the structure, and cut it up into three compartments 20 ft. by 14 ft., 8 ft. 8 in. by 14 ft., and 16 ft. 4 in. by 14 ft. (at the bottom) for return air, pumps and ladder, etc., and hoisting respectively.

This shaft was sunk without serious difficulty to the ledge, although the inflow of water reached nearly 1400 gal. a minute before the gravel was passed. The rock was found to be badly fissured, and in order to make a tight joint it was eventually necessary to sink into it 20 ft. and to build up a concrete wall to a junction with the drop-shaft. Before this could be done a rise in the river raised the water-level in the alluvial plain, and channels were established between the clay walls and the shaft, resulting in a heavy flow under the shoe, which drowned the pumps. The water was finally fought down, with the usual troubles from sand cutting out the pumps, the rock section was excavated, and the joint made. The chief point of interest was the success with which the flow of water under the shoe was controlled, and indeed practically cut off by the use of pine blocks and wedges, enabling the final joint to be made without the use of compressed air.

In the Lake Superior region several concrete drop-shafts have recently been completed, including two in the Swanzy district, south of the Marquette range, as well as others in Minnesota. The Smith shaft at Swanzy was a heavily reinforced concrete structure, of rectangular section, 14 ft. 10 in. by 10 ft. 10 in. in the clear at the bottom, where the walls were about 3 ft. thick. The thickness was reduced at intervals so that at the top it was 18 in. This shaft was sunk through 61 ft. of water-bearing gravel and sand, and a tight joint made with the ledge, all under air-pressure. The working chamber was formed by a heavy timber deck, about 10 ft. above the shoe, built into the side walls as they were laid. The dirt was hoisted through a cylindrical well, 4 ft. in diameter, made of steel plate, built up in 4 ft. sections, with an air-lock at the top above water-level, in order to give the men a chance to escape in case of accident.

The Kidder shaft, also in the Swanzy district, is noteworthy from the fact that it is cylindrical through the sand, although the rock section below is as usual rectangular. The circular form is highly advantageous for a drop-shaft in sand because of its greater strength, and where the material is concrete, is at least as easy to construct as the rectangular shape. It has a good deal of waste room, however, especially where the compartments are side by side, as they are in most rectangular shafts. The Kidder shaft, however, has a long cage compartment alongside two skip compartments, and these three, together with narrow ladder and pipe compartments fit neatly inside a cylinder.

The sinking of the Kidder shaft to a depth of about 104 ft. in fine sand, nearly 90 ft. being below water-level, was effected in two stages. Down to about 85 ft. the water-level was not lowered; the ground was excavated inside the concrete drop-shaft by an orange-peel or clam-shell bucket operating inside a dredging-well, 10 ft. in diameter, constructed of steel plate, and built up in sections as the shaft was sunk. At the bottom the dreging-well was funneled out and attached by a bracket ring to the upper end of a steel caisson, about 10 ft. high, imbedded in the concrete, and forming the shoe of the drop-shaft. Below 85 ft. it was no longer possible to get the shaft down by this method. A steel deck was then attached to the top of the dredging-well, through and above which a 4-ft. hoisting-well was carried with an air-lock at the top. The remaining distance was sunk, and the joint with the rock made under compressed air. This method and equipment may be considered standard where the conditions are favorable, namely, when the sand or gravel to be passed through is uncemented and not too coarse to dredge, and when under the pressure of the beveled shoe it will readily cave into the shaft, without boiling up in the bottom.

H. Bousin has written[1] an interesting account of the manner in which the water was held at the ledge in a very wet shaft, permitting sinking in the rock below to be carried out. The general plan was to dam off the two end compartments with walls of puddle clay held in place with lagging, and to conduct the water to them by means of water-rings. Each of these compartments was supplied with an 18-in. Cornish pump, both together delivering about 2000 gal. a minute. In this way the water was completely cut off from the middle compartment, which was sunk without difficulty. After leaving a toe of rock under the end compartments the shaft was lengthened out to its full dimensions, and sunk as far as it had to go, a liberal sump of course being provided. The toes of rock under the two end compartments were then cut out one after the other.

[1] *Min. and Sci. Press*, Jan. 25, 1908.

This shaft is in the Caribou district in British Columbia. It passes through 91 ft. of gravel, and then into rock. The ultimate purpose is to drift from the shaft out under the Willow river in order to get at the gold-bearing gravel in its bed. Drifting under a flowing river and mining gravel in its bed is not an everyday feat, nor one which many engineers would lightly undertake, and it is to be regretted that Mr. Bousin has not furnished further details as to the character of the ground.

For rapid sinking in rock we have to go to South Africa, where in the development of the deep level mines of the Rand, speed has been of the first importance because of the heavy interest account running against the operation, due to the great initial investment. In South Africa the underlying principle of rapid sinking was learned early and thoroughly, namely, that it can be achieved only by disregarding the cost per foot and the pay-roll, and by being wasteful of labor.

E. M. Weston has given[1] an interesting account of some of the rapid sinkings, with valuable details of the conditions, equipment, and organization under and with which they were carried out. The Brackpan holds the record of 204 ft. sunk in a month of 31 days, but the Simmer & Jack West is a close second with 203 ft. The latter also has averaged 152 ft. a month for six months. These results were all attained with hand drilling. The best work with machines was 150 ft. a month. Mr. Weston concludes that for Rand conditions sinking by hand is both cheaper and more rapid than by machine.

No noteworthy articles appeared during 1908 on the sinking of rock shafts in this country, but in the spring of that year an interesting and important shaft was completed, namely, the "D" shaft of the Newport mine on the Gogebic range, from which soft iron ore is now being mined or developed at the greatest depth yet attained by any iron mine in the Lake Superior region—nearly 2200 ft. vertically below the surface—or about 400 ft. below sea-level. This shaft, being in a "going" mine, was not sunk altogether from the surface, but was attacked at intermediate points by raising on its line. It was started in December, 1905, and went into commission in May, 1908, within two weeks of the predicted date. It is not vertical, but inclined at 68 deg., following the general dip of the footwall. The only serious difficulty encountered in the sinking of this shaft, occurred at about 900 ft. from the surface. A section, nearly 200 ft. in hight, in quartzite, apparently very solid, suddenly came in with very little warning, wrecking the steel sets completely. This sudden and general movement of a considerable mass of the country rock, which was not at all heavy in

[1] *Eng. and Min. Journ.*, Feb. 29, 1908.

the ordinary sense, but on the contrary exceptionally firm, is believed to have resulted from residual strain due to folding, as in the well-known case of the Monson granite. Newport "D" shaft has five compartments side by side, four for hoisting and one for ladder and pipes. In one particular it differs from most of the shafts recently sunk in soft-ore mines on Lake Superior, viz., it has no cage compartment. In that region the tendency of late years has been to provide all hoisting shafts, whether vertical or inclined, with a cage compartment long enough to take a timber trolley loaded with 9 ft. caps laid flat. Such a cage permits the trolley, after being loaded in the timber yard, to reach the raise where the timber is to be used without rehandling the sticks—an important consideration in view of the large amount of timber which the mining of soft-ore requires. So long a cage of course involves a shaft nearly square in section.

INDEX TO THE LITERATURE OF THE ACTION OF NOXIOUS GASES ON VEGETATION.

By W. A. Hamor.

An attempt has been made to prepare a complete bibliography of the injurious effects of smoke and noxious gases on plant life. I am indebted to the treatises of Schroeder and Reuss, and Haselhoff and Lindau for a number of the German references. The "Bibliography of Injuries to Vegetation by Furnace Gases," prepared by Persifor Frazer (*Trans.* A. I. M. E., 1906, 399), although brief (45 titles), is of great value to all interested.

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PROGRESS IN ORE DRESSING AND COAL WASHING IN 1908.

By Robert H. Richards and Charles E. Locke.

During 1908 the Richards pulsating classifiers and jigs reached a stage of development which warranted their appearance in commercial work. There was the usual annual crop of bumping tables of the Wilfley type. The new Foust jig seems to be a step forward. The Blake-Morscher electrostatic machine was taken over by the people who control the Huff machine and the latter was installed on a commercial scale in a new mill in Wisconsin. At Broken Hill, the flotation processes seem to be working into shape. The problem of the concentration of the second class ores at Cobalt, Ontario, is being worked out. In the United States the large mills at Bingham, Utah, and Ely, Nevada, have reached the point of smooth running and are serving as forerunners of other mills of similar magnitude to be erected in Arizona. For the coal washing industry the United States testing plant at Denver, Colorado, is doing good work.

Crushing Machinery.

Immense Gates Breaker.—The Allis Chalmers Company has manufactured for the Casparis Stone Company at Columbus, O., an enormous Gates breaker. This machine is 25 ft. high, 14 ft. across the top shell and weighs about 400,000 lb. The minimum feed opening is 36 in. The capacity has not been determined.[1]

Holman Pneumatic Stamp.—This stamp has been used for many years in Cornwall. A good illustration of the cylinder is given in the original article. It shows the stamp stem connected to the piston and also the trunnions on the side of the cylinder to which the connecting rods are connected from the steam cylinder above.[2]

Tatman Grinding Mill.—This is used at the Blue Flag cyanide mill at Cripple Creek, Colo., for grinding material from a Dodge breaker so that it will pass a 16-mesh screen. It resembles a Chili mill in having a central shaft and radial arms carrying mullers which travel around on a ring shaped die, but it differs in that the die is trough shaped and

[1] *Eng. and Min. Journ.*, LXXXV (1908), p. 619.
[2] Edward Walker, *Eng. and Min. Journ.*, LXXXVI (1908), p. 213.

the mullers, of which there are seven, are rounded to fit the trough. The mill is 7 ft. in diameter and 4½ ft. high and weighs 18,000 lb. It reduces 50 tons of quartz from ½ in. to 20 mesh in 24 hours using 7 h.p. The chrome steel die ring is 6 ft. in diameter. The mullers with chrome steel tires are 22 in. in diameter and weigh 800 lb. each. The central shaft makes 100 r.p.m. The screen surface is 2 ft. high.[1]

Crushing and Grinding Practice.

The Mechanics of Ore Crushing.[2]—C. DeKalb found that in crushing glass spheres, 1 in. in diameter, in rolls set ¼ in. apart, with only a few exceptions in several hundred experiments, the sphere ruptured into six principal pieces, five being relatively large fragments with irregular fracture-planes or surfaces, and one being an elongated fragment the length of which almost equaled the entire distance between the points of contact of the opposing roll-faces with the original sphere. The large fragments could be refitted in the great majority of cases so as to reëstablish the sphere, which was then found to contain a hollow space having the form of an elongated ellipsoid terminating in small round openings at the points of contact with the roll-faces. The elongated fragment had come from this hollow space, which however it did not completely refill. A considerable quantity of small fragments and finely pulverized material had been produced in the interior of the sphere between the elongated fragment and the larger pieces. The volume of this interior ellipsoidal cavity, as determined by displacement of water, averaging about 10 per cent. of the whole, and the volume of the elongated fragment was about 4.5 per cent. of the entire sphere, leaving about 5.5 per cent. of relatively fine comminution as the result of a single passage of the sphere through the rolls. Exactly similar results in all details were obtained by crushing such spheres in jaw crushers. Selected Baltic pebbles were crushed under like conditions, yielding similar results, sometimes as perfect as in the case of the glass spheres, but original planes of weakness in the pebbles usually modified the forms of the fragments materially. Selected pebbles of granitoid rocks from glacial moraines gave results more nearly approximating those with the glass balls, except that the elongated fragment was more irregular in outline.

In similar experiments made by crushing with stamps, steel bars ¾ in. high were placed upon the die at either side of the spheres, which were 1 in. in diameter. As the stamp came to rest upon the steel bars the crushing effect was due solely to impact of the falling stamp upon the sphere. The stamp weighed 850 lb. and dropped 7¼ in. to the steel bars.

[1] F. L. Barker, *Mines and Minerals*, XXVIII (1908), p. 480.
[2] *Min. and Sci. Press*, XCVI (1908), p. 155.

In all cases the upper portion of the sphere was irregularly fractured, the lower portion was reduced to small particles, and in the center, in contact with the die, was found a quantity of finely comminuted material.

Other experiments in crushing granite, syenite and various other ores, with and without disseminated sulphides of iron, copper, and lead, brought out the following facts: Materials, which had been kiln-dried until they contained less than 2 per cent. moisture, yielded sizing curves which were practically coincident when similar crushing conditions had been maintained, irrespective of the mineral composition of the material. When reducing ores, in the normal way with rolls, intermediate screening out of material finer than 28-mesh (0.6096 mm.) between successive crushings did not affect the sizing curves of the total resultant product. The quantity of fine product rapidly increased with the quantity of water of imbibition in the ore up to about 7 per cent.

Comparative Crushing by Rolls and Ball-Mills.—To determine the nature of the crushing effect of these machines the following experiments were made[1]: In experiment I, matte was crushed in a ball-mill to pass a 6-mm. screen; in II, this mill crushed ordinary roasting sulphide; in III, a sulphide ore containing gypsum and pyrite was crushed by rolls to pass a slotted screen $\frac{3}{16}$x$\frac{3}{4}$ inch.

SIZING TEST OF CRUSHED MATERIAL.

Test.	On 6 Mesh.	Through 6 on 20 Mesh.	Through 20 on 40 Mesh.	Through 40 on 80 Mesh.	Through 80 on 120 Mesh.	Through 120 Mesh.
	%	%	%	%	%	%
I.....	10	15	18	12	13	22
II....	13	18	18	12	12	27
III...	25	24	28	3	4	16

Tube-Mills.—In this article E. B. Wilson[2] presents the main facts regarding the tube-mill and its work up to the present time. Beginning with its history and a discussion of the nature of slimes he goes on to describe the details of the mill, the variations in different makes dealing especially with the various forms of lining and their relative lives, the exact action of the pebbles, the nature of the pebbles to be used, the feeding and discharging, the proper speeds, water quantities and charges of pebbles, the power required, the cost of crushing and capacities.

Economics of Tube-Milling.[3]—A series of tests at the Colorado plant of the United States Reduction and Refining Company, at Colorado

[1] *Min. and Sci. Press*, XCVI (1908), p. 572.
[2] E. B. Wilson. *Mines and Minerals*, XXIX (1908), p. 8.
[3] *Mines and Minerals*, XXVIII (1908), p. 587. *Journ.*, Chem., Met. and Min. Soc., South Africa, IX (1908), p. 57.

Springs, on Stearns-Roger tube-mills, is described by H. W. Fox. The mills were of the trunnion type, 5 ft. in diameter and 23 ft. long, driven by gear and pinion. The mills were lined with 4-in. silex blocks making the inside dimensions 4 ft. 4 in. x 22 ft. 4 in. The total volume was 350 cu.ft. and each held a maximum pebble load of 33,600 lb. The pebble load was varied as were also the size and rate of the feed, and the amount of feed solution. The tests show that, for any given mill, there is a point of maximum efficiency of grinding with minimum consumption of power. Curves plotted from the results of sizing tests on the products obtained under various conditions show that, in the case in hand, this point comes with a load of 19,000 lb. of pebbles. Other curves show that the power increases with increased percentage of solution up to 36 per cent., and that beyond this point the percentage of solution does not affect the power consumption to any great extent. This point is considered to be that at which the pulp becomes sufficiently thin to allow the pebbles to roll. The ore in all these tests was tailings from chlorinating barrels which had been previously crushed to pass through a 12-mesh, No. 19 wire screen.

Grinding Efficiencies at Redjang Lebong, Sumatra.—At this mill sizing tests both wet and dry are made daily.[1] The following results were obtained on July 20, 1908, using a 16-mesh screen in the stamp mortar:

SIZING TESTS ON CRUSHED MATERIAL.

Size.	Stamp Product.			Tube-mill Product.		
	Wet.	Dry.	Efficiency Numbers.	Wet	Dry.	Efficiency Numbers.
On 100 mesh.....	45.37	43.02	21.5	1.84	0.92	0.5
Through 100 on 150 mesh.....	8.94	7.82	16.6	14.92	10.53	22.3
Through 150 on 200 mesh.....	4.37	5.54	16.1	8.85	10.89	32.0
Through 200 mesh.....	41.32	43.62	218.0	74.39	77.66	388.5
	100.00	100.00	272.2	100.00	100.00	443.3

The material through 200 mesh is still further graded by elutriation. The efficiency numbers are calculated according to the scheme of Pearce and Caldecott (*op. cit.*, Sept., 1906, p. 72), that is, the percentage (dry) is divided by the average size of the grain of that grade and the result divided by 100.

The mill contains seventy 150-lb. stamps and six Schmidt-Davidsen tube-mills 15 ft. 6 in. x 4 ft. 2 in. inside the iron shell. Only four mills are used. The mills are lined with a local eruptive rock and are fed with reef pebbles. Each mill consumes 47.5 h.p. The stamps require

[1] A. Yates. *Journ.*, Chem. Met. and Min. Soc., South Africa, IX (1908), p. 187.

183 h.p. and the duty per stamp per 24 hours is 4 tons through 16-mesh screen or 3.7 tons through 20-mesh or 3.0 tons through 35-mesh. Sizing tests under different conditions are as follows; the tube-mill products, in every case, received their first crushing in the 70 stamps, passing through mortar screens of the size stated:

SIZING TESTS OF MILL PRODUCTS.

Size.	Stamp Product.			Tube-mill Product.			
	16-mesh Screen.	20-mesh Screen.	35-mesh Screen.	16-mesh Screen 3-tube Mills.	16-mesh Screen 4-tube Mills.	20-mesh Screen 3-tube Mills.	20-mesh Screen 4-tube Mills.
On 100 mesh	45.7%	43.2%	24.8%	4.8%	1.1%	3.4%	1.3%
Through 100 on 150 mesh	8.2	7.9	9.3	23.8	7.2	13.8	7.1
Through 150 on 200 mesh	5.2	6.2	9.2	8.5	17.7	14.4	14.0
Through 200 mesh	41.1	42.7	52.7	62.9	74.0	68.3	77.5
Efficiency number	259	271	326	393	438	415	444

For comparing different crushing machines, or the same machine under different conditions, the author proposes a term "Relative Efficiency," which is obtained by multiplying the grading efficiency number by the tonnage and dividing the result by the horse-power. The following table shows the results of such a calculation.

TABLE OF CRUSHING EFFICIENCIES.

Number of Stamps.	Mesh of Stamp Screen.	Stamp Duty Tons.	Number of Tube Mills.	Tons per Day.	Horse Power.	Grinding Efficiency.	Relative Efficiency.	Relative Efficiency to Unity.
70	35	3.0	.....	210	183	326	373	1.00
70	20	3.7	.....	259	183	271	383	1.02
70	16	4.0	.....	280	183	259	396	1.06
70	20	3.7	3	259	325	415	330	0.88
70	20	3.7	4	259	373	444	308	0.82
70	16	4.0	3	280	325	393	341	0.91
70	16	4.0	4	280	373	438	328	0.87

Wet Concentration.

Separation of Slime.—The process of slime settlement described by H. G. Nichols and G. McM. Ross[1] first suggested itself in connection with the dewatering of tailings at a property in Central America. One of the principal causes that retard the settlement of slimes or finely divided material is held to be, in greater measure than has heretofore been appreciated, the accumulation of the slime itself in the process of settling. By avoiding such accumulation, free settlement of the solid particles is obtained. The principle involved is that of removing solid

[1] *Min. and Sci. Press*, XCVI (1908), pp. 563 and 697. *Trans.* I. M. M., XVII (1909), p. 293. *Eng. and Min. Journ.*, LXXXVI (1908), p. 854.

matter as fast as it reaches the bottom of the tank in which it is settling. The effect is that the solid matter settles very compacly, if not allowed to accumulate, and its removal prevents the thickening of the liquid above. Settlement becomes free when the retarding effect, due to an increase of density in the medium by the accumulation of the slime itself, is removed. Experimental results accord with the noted acceleration in the settlement of slimes in distilled or hot water, due, it is believed, to the reduced density of the medium, and they tend to show the marked effect on settlement produced by small variations in specific gravity.

Experiments were made in British Columbia on a quartz ore with argillaceous material, containing about 2 per cent. sulphides of iron, zinc and lead. Particular care was taken to separate the heavier and coarser portions from the pulp to be used, by crushing in a stamp battery, with high discharge, and by passing the pulp through a hydraulic classifier. The overflow from this classifier was used in making up the charges, and contained 86 per cent. of material which passed a 200-mesh I. M. M. standard dry screen. The experimental plant consisted of a pyramidal shaped tank, about 4x5 ft., connecting at the bottom through an 8-in. square aperture with a closed box in which a 10-in. belt was made to travel slowly. The pyramidal tank was provided with a trap door, just above its connection with the closed box, which could be removed from above as desired, thus allowing free passage between the two vessels.

The fine pulp was fed into the pyramidal tank while the closed box was filled with water to the level of the material in the tank. The conveying belt ran at a speed of 4 ft. per minute. The trap door was opened and it was found that the finest slime was carried out of the tank and discharged by the belt. The average discharge did not contain over 25 per cent. moisture, including the finest slime, while from a pulp containing 40 per cent. solid matter the initial discharge carried but 22.5 per cent. moisture. The belt showed no friction in running and the material remained on the belt and showed no tendency to choke in the bottom or around the conveyor. The process may be used continuously or intermittently and can be easily adapted to practical working conditions. Mr. Ross says that it is the best dewatering scheme available, as it is safe to figure that by its use a ton of ore can be crushed and concentrated with a loss of but 500 lb. of water. The original article contains experimental data, a discussion of the theory of slime settlement, and a description of the Howatson settling machine.

Investigation of Jigging.—The investigation of R. P. Jarvis[1] covered a hydraulic classifier, a Monroe pulsion jig, a Vezin laboratory jig, a

[1] *Bull.* A. I. M. E., Sept., 1908, p. 697.

5-sieve Harz jig and a Jarvis laboratory jig. This last was of the Harz type, had one sieve 12x8 in. and was specially built for testing purposes, being arranged to be run by eccentrics, by circular arc cams or by involute cams. The experiments were made on artificial mixtures of galena, sphalerite and quartz using combinations of either two or three minerals, ranging in size from 2 mm. down. In each test the hutch product was sized on a nest of sieves and each size analyzed. The piston speeds were also calculated. It is impossible to give a résumé of the tests in a short abstract. The author's conclusions are as follows:

(1) The pulsion-reaction is by far the most important one in the process of jigging. During this period, with sized grains of different specific gravities and with proper pulsion-velocity, the separation between them will be complete. The size limit is indicated by the hindered settling ratio. If the minerals are not sized, or lie above these ratios, the separation cannot be complete, but a definite arrangement will result. The positions of equilibrium will be attained when the above ratios of diameters are attained, after which further separation by pulsion is impossible.

(2) Suction due to the movement of water columns supplements gravity. Resisting the sum of these two forces is the resistance of the walls of the tube through which the grain must pass. The reaction, as a whole, must therefore be a resultant, the chief components of which are the force of the water columns, which are purely non-selective, acting with equal intensity upon all particles of the same shape and size, regardless of their specific gravities. Any advantage that the small, heavy grain would have over a large, light one would, of course, appear in the resultant tending to carry it to the hutch. The effect of the forces opposing the movement of the grain depends upon the character of the grain, and the conduit through which it is supposed to pass. Under any condition, the diameter of the grain cannot be greater than that of the conduit. If the channels are inclined, or crooked and zigzag (the condition obtaining on a jig-bed), the particles will more easily lodge against the sides of a tube large enough for them to pass through if the tube were vertical. The rapidly descending water currents, passing through these channels, easily carry the grains along. Thus suction, due only to the moving columns of water, constitutes a powerful impelling force to carry through the interstitial spaces those particles which, under the force of gravity alone, cannot move. Suction is, therefore, a necessary complement to pulsion in the jigging of all unsized material, and generally valuable in jigging under all conditions.

(3) From the observations under (2) it is clear what effect the bedding will have upon the result. Any part of the bedding or ore column

remaining fixed during the pulsion-cycle must be looked upon merely as a mass of very irregular tubes, of length somewhat greater than the thickness of such part, owing to their inclination, since they are mostly inclined. To that extent they are only an extension of the jig-sieve. The result of thickening or thinning the bed, or of increasing or decreasing the size ratio between bedding and feed, is evident. This assumes, of course, that the largest particle of feed is smaller than the sieve aperture. It is evident, too, that the shape of the bedding grain will have a marked effect. Grains that are more or less equi-dimensional, as galena, iron balls, etc., will form a more open bed than one of antimony ore, which breaks into long grains. Any part of the bedding free to pulsate is to be considered as part of the ore column, and is amenable to all the conditions applying to this reaction.

(4) The effect of very rapid acceleration, amounting to a shock or blow on the bottom of the jig bed, is an important factor. Its effect is to accelerate the work done by suction, and render a larger catch possible with a low mean piston velocity. The pulsation of the jig bed due to this force should be distinguished from that due to ordinary interstitial velocity; one sifts, the other separates.

(5) The results of many experiments, in which the piston speeds during the pulsion and suction were not the same, seemed to show that only by properly balancing the two are the best results attained. It has been generally noted that the eccentric, giving equal mean velocities, yields about as good results as any of the accelerated strokes. This observation applies only with the sizes used in the tests, 2 mm. downward, and it is not safe to speculate what the results would be for other sizes.

(6) While the use of the jig for the treatment of material sized between wide limits is possible, the advantages that follow when a more or less perfect sizing has preceded cannot be denied. It must be observed, that in the English system itself, when the hutch products of one jig are treated on another, we are treating sized material.

(7) The more general application of the English system, or the use of the jig instead of the hydraulic classifier for the treatment of unsized material, seems to be clearly indicated. This has been recognized in some quarters, but a wider use than it has hitherto been accorded appears to hold out favorable inducements. This seems to be a field eminently suited for the English methods of jigging—one that cannot be filled by the Continental system of jigging closely sized material.

(8) The arguments that have been advanced for the adoption of the English system on the ground that equal settling ratios many times larger than those obtainable under free-settling conditions exist on the jig bed, are not tenable.

The completeness of Mr. Jarvis' work serves to strengthen our knowledge of the principles of separation by jigging and corroborates in many respects my conclusions published in 1894. (*Trans.* A. I. M. E., XXIV, p. 409.) My latest determination gives the hindered settling ratio of quartz and galena as 6.9, using galena with a specific gravity of 7.5. The determination by Jarvis gave 5.8, using galena of 6.6 specific gravity.

Wilfley Table.—This paper by R. H. Richards[1] gives the results of 16 runs using artificial mixtures of chalcopyrite and quartz. Four runs were made on natural products: 2mm. to 0, 1 mm. to 0, 0.5 mm. to 0, and 0.25 mm. to 0. Six runs were made on sized products: 2 to 1.4 mm., 1.4 to 1 mm., 1 to 0.75 mm., 0.75 to 0.50 mm., 0.50 to 0.36 mm., and 0.36 to 0.28 mm. Six runs were made on products classified by the following rising velocities: 85, 55.4, 36.3, 23.6, 15.4, and 10 mm. per second. The concentrates, middlings and tailings from each run were sized on a nest of sieves and each sized product analyzed for copper.

The results show that both sized feed and classified feed give better results than natural feed in the percentage of copper that is easily and quickly saved in the concentrates. The tailings from the sized feed are decidedly better than those from the classified feed and the tailings from both these classes are better than those from natural feed. The losses come in the fine sizes of tailings both from natural feed and from classified feed. The theoretical diagram (see THE MINERAL INDUSTRY, XVI, p. 974) indicates that classified feed should have advantage over both natural feed and sized feed. This advantage is not realized owing to the imperfections of existing classifiers.

Concentration of Slimes.—For this work the greatest possible consistency should be given to the pulp, compatible with complete stratification on the tables.[2] Quartz pulp may go up to 25 or even 30 per cent. solids, while clayey or porphyritic ores should not exceed 15 to 18 per cent. A good slime classifier must have the slime particle brought to rest upon its surface and not be propelled across the surface. Just enough motion is necessary to keep the pulp "alive" or in quicksand condition.

There are three radical types of tables: (1) Conical round tables, (2) end-moving belts, represented by the Frue and Embrey vanners, (3) side-moving belts, represented by the Luhrig, Bilharz, McCoy and Roberts tables. Other tables are a combination of these types. Round slime tables take pulp of higher consistency than vanners to give same results, in one case 25 per cent. for the former and 15 per cent. for the latter. Side-moving belts take a position between round tables and

[1] *Bull.* A. I. M. E., Sept., 1908, p. 685.
[2] E. A. Sperry. *Western Chem. and Met.*, IV (1908), pp. 122, 191, 207, 223, 247. *Electrochem. and Met. Ind.*, VI (1908), p. 212.

vanners in this respect. Vanners have the disadvantage that they do not make a middling product.

In one case, operating a vanner and a round table in competition on a copper slime in which the finest sizes were much lower in value than the coarser, the round table concentrates were larger in weight, contained more of the total copper and assayed lower (19 per cent. copper) than the vanner concentrates (23 per cent. copper). The vanner tailings assayed higher in copper than those of the round table.

Capacities in tons per 24 hours are as follows: Round table, 2 to 3 tons per foot of radius; Luhrig table, 1.5 to 2 tons per foot of width; vanner, 1 to 1.5 tons per foot of width. Comparisons of efficiency are given in the accompanying table:

EFFICIENCIES OF SLIME CONCENTRATORS.

	Solid consistency.	Capacity per 24 hours.	Assay concentrates.	Assay tailings.
Round table	25%	10 tons	34% iron 6.5% silica	Trace of gold
Luhrig table	15%	6 tons	30% iron 18% silica	0.01 oz. gold
Vanner	15%	6 tons	33% iron 10% silica	0.02 oz. gold
Round table	25%	12 tons	47% lead	67% extraction
Vanner	14%	6 tons	55% lead	39% extraction

The proper amount of motion is between 200 and 300 in. per minute. For example, a vanner with 250 r.p.m. and a throw of $\frac{1}{2}$ to $\frac{5}{8}$ in. has 200 to 250 in. total motion. A Sperry buddle or round table has $\frac{3}{16}$ in. circle of motion, 300 r.p.m., or 225 in. total motion.

Crushed ore may be divided into four classes: (1) Coarse, down to 0.5 mm., for jigs. (2) Sands, from 0.5 to 0.1 mm., for tables of Wilfley type. (3) Fines, from 0.1 mm. to 0.05 mm., for vanners and slime tables. (4) Slimes, below 0.05 mm. or 200 mesh. The term "slime" is capable of three constructions: (*a*) The practical, which defines it as that portion of ore unfit for treatment on the usual forms of concentrators. This is about 100 mesh and finer. (*b*) The intermediate between practical and technical, which defines it as that portion of ore which requires considerable time to settle in water. This is about 200 mesh and finer. (*c*) The technical which defines it as that portion of ore which is so fine that the particles lose their individual integrity and, in combination with water, approach the condition of a hydrate.

Concentrating Machinery.

Vibracone Separator.—This is a dry fine screening apparatus in the form of an inverted cone.[1] Material fed by a revolving centrifugal dis-

[1] *Eng. and Min. Journ.*, LXXXV (1908), p. 902. *Can. Min. Journ.*, XXIX (1908), p. 53. *Iron Age*, LXXXI (1908), p. 844.

tributor passes down over the inside of the screen, which slopes at an angle of 45 deg., and the oversize is discharged at the apex of the cone while the undersize is caught in the hopper surrounding the screen. The screen does not revolve but is given a vertical vibration at the rate of 600 impacts per minute. The amplitude of the vibration may be regulated from 1/64 to 1/2 in. The whole apparatus is enclosed in a dust tight casing. The total hight is 8¾ feet and the diameter 8¼ feet.

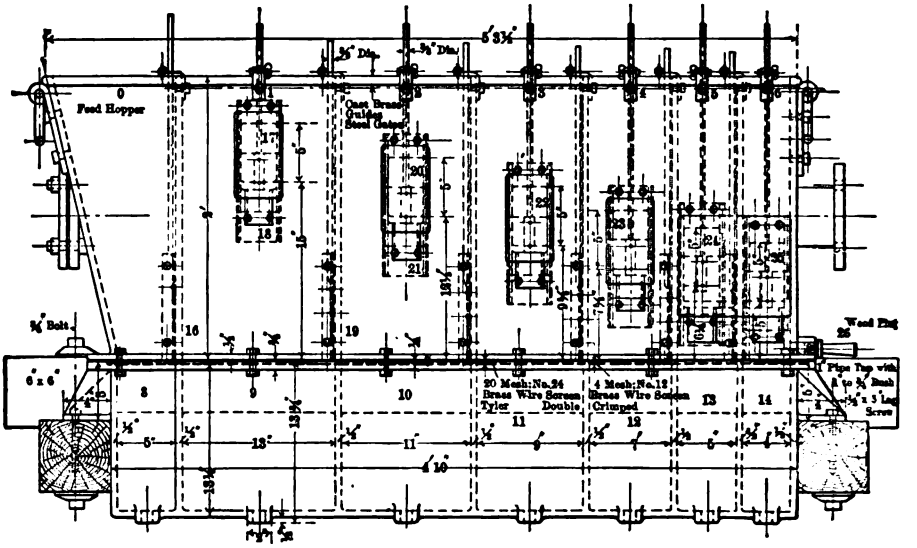
Dorr Classifier.—This is extensively used to separate sands from slimes for cyanide work; it consists of a long settling box having a sloping bottom which also acts as one end, and is extended some distance above the feed level.[1] The pulp to be treated is discharged across the center of the tank. The sand immediately settles and is dragged up the sloping bottom by means of a series of hoes resembling scrapers to which a raking motion is imparted; while the slime, kept in suspension by the movement of the scrapers, is discharged over the opposite end into a launder.

The advantages of this apparatus over cone classifiers are that it saves mill height and also delivers sands with a low moisture content for tube milling or cyaniding.

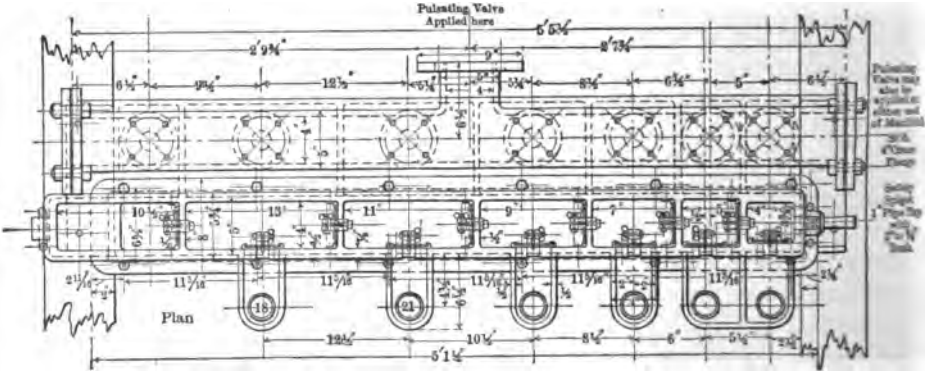
Richards Pulsator Classifier and Pulsator Jig.—The Richards pulsator classifier, while acting by hindered settling, differs widely from any of the devices heretofore described.[2] The figures show the usual inverted form of the pulsator classifier as made by the Denver Engineering Works Company. The classifier consists of a feed hopper (0) and six treatment pockets (1), (2), (3), (4), (5) and (6). The bottom of the feed hopper and of the six treatment pockets is a screen of two layers, one coarse mesh for strength, the other fine for size of hole. An intermittent rising current is admitted to the hutch or pressure boxes (8), (9), (10), (11), (12), (13) and (14) through the revolving cock (15). The ore is introduced at 0 and falls immediately to the screen level where it encounters the rising pulsating current of water. This serves to agitate the ore, get it in partial suspension and prevent clogging of the feed hopper. From the feed hopper (0) the ore passes below the gate (16) where it encounters and is subject to the action of the slowly rising, pulsating current of water in (1). This current further agitates the ore and raises to the discharge opening (17) the finest particles in the ore, whence they are discharged at (18). Through the gate (19) the ore passes from classifying compartment (1) on the screen to classifying compartment (2) where it encounters a more swiftly rising, pulsating current; there the next heavier grade of sands is lifted to the discharge

[1] J. V. N. Dorr. *Mines and Minerals*, XXVIII (1908), p. 541.
[2] Colorado Sci. Soc., IX (1906), p. 81. *Metallurgie*, V (1906), p. 728. *Jern Kontorets Annaler* (1906), p. 809. *Mines and Minerals*, XXIX (1908), p. 122. *Eng. and Min. Journ.*, LXXXVI (1908), p. 621. *Mining Science*, LVIII (1908), pp. 248 486.

opening (20) and discharged at (21). This process is repeated successively in pockets (3), (4), (5) and (6). In the latter compartment only the heaviest particles will have been left in the ore bed and these are discharged from (25). In cases where the ore contains particularly heavy particles, which the current in the last pocket is unable to


Feed Hopper
Open Bronze Gulley Steel Gates
20 Mesh No.24 Brass Wire Screen Tyler Double
4 Mesh No.12 Brass Wire Screen Crimped
Wood Plug
Pipe Tap with ¼ to ½ Bush
Screw

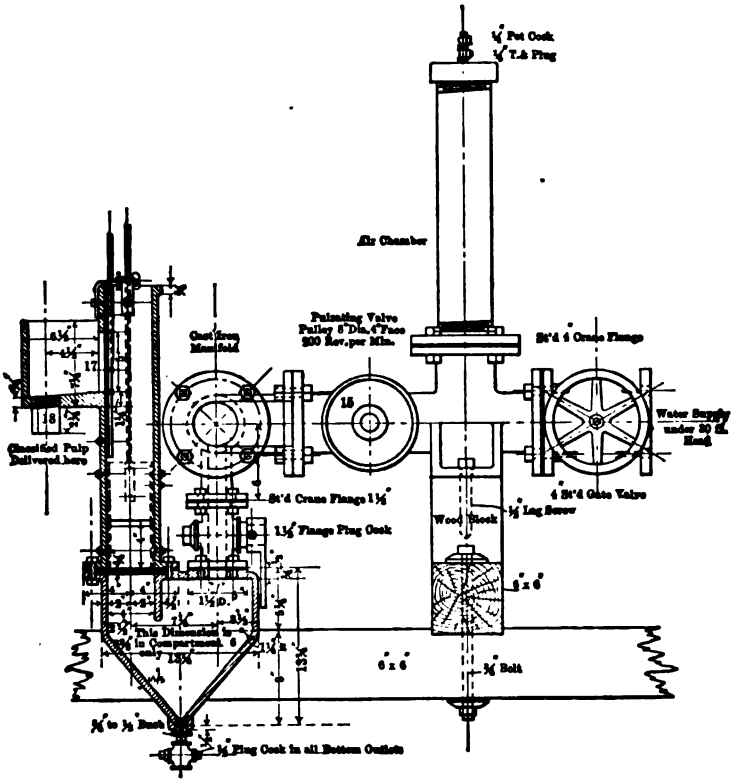
RICHARDS PULSATOR CLASSIFIER.—SIDE VIEW.



RICHARDS PULSATOR CLASSIFIER.—PLAN.

lift to the discharge (25), the plug (26) may be removed from its spigot. The discharge from this spigot may be made finished concentrates. To do this, however, requires constant attention and it is better to send the product directly to a jig for final cleaning up.

The openings (17), (20), (22), (23), (24) and (25) are the only discharge openings in the machine and (17) is considerably higher than (25). Hence the water in the machine tends to flow toward (25) washing the ore along the screen to the final discharge (25) or (26) as the case may be. Gates are provided for varying the hight of the discharge openings within wide limits. These discharge openings may in some cases require adjustment until they are almost in a horizontal line.



RICHARDS PULSATOR CLASSIFIER.—SECTION.

With heavy ores, 2.5 mm. and below in size, it is customary to have (17) 15 to 16 in. above the screen, and opening (25) about 5 in. above its screen, with the other openings on the diagonal line between. When treating ores of 1.5 to 1 mm. maximum grain and not heavily charged with mineral, (25) may be 10 in. more or less and (17) 15 in. more or less above the screen. By observing this adjustment the difficulty of keeping slimes out of the last discharges is greatly reduced. Properly

adjusted, this classifier will discharge all the slimes in the ore from the first two discharge openings, (17) and (20).

The sieve serves to support the bed of ore and has no part in the classification. Particles of free mineral of sufficient size to settle against the classifying current in the first one or two compartments, but finer than the mesh of screen may occur in the feed, in which event they will sink to the bottom of the hutch whence they may be drawn off as a rich concentrate through the spigots shown. This, however, will rarely occur and in general the hutch need not be drained oftener than once a day. It is considered wise to prevent the formation of hutch product as much as possible.

The number of overflow or discharge openings in this classifier is six. The first and second overflow products are slimes and the last overflow consists of a mixture of the heaviest ore particles with a true classified product. This leaves three perfectly classified products for table treatment. For these reasons the number of treatment pockets should never in any case be less than six. Any reduction of this number results in defeating the purpose for which the classifier is designed.

This classifier is built in several sizes, as shown in the accompanying table:

SIZES, CAPACITIES, ETC., RICHARDS PULSATOR CLASSIFIER.

Type.	Size Inches.	Number of Compartments.	Approximate Capacity in Tons per 24 Hours.*	Dimensions, Approximate.				
				Length Inches.	Height Inches.	Width Inches.	Pulley Face Inches.	Pulley Diameter Inches.
Direct	2	6	40	51	30	21	1½	8
Inverted	2	6	40	37	31	19	1½	8
Inverted	3	6	100	53	35	21	3	8
Inverted	4	6	175	70½	38	22½	4	8

* With pulp at a consistency of 3 parts water by volume to 1 of ore. Ore crushed to pass 2 mm.

The advantages of the pulsator classifier are: 1. A minimum dilution of slime. The slimes are diluted with the water rising in two pockets only. 2. The intermediate and coarse overflow products possess the advantages of hindered settling ratio and are at the same time free from fine mineral which would contaminate the tailings of the fine concentrators. 3. The machine has low water consumption.

The Richards pulsator classifier is also made in a form known as the direct in distinction from the inverted form just described. This classifier is made upon the same principle as the pulsator jig and is suited for low capacities, testing laboratories, etc. It is capable of quicker adjustment than is the inverted form but dilutes the slimes with all the rising water.

Richards Pulsator Jig.—It has been shown by experiment that, when treating fairly closely sized products, the removal of the suction of the plunger jig results in an enormous increase in the speed of jigging. Suction is of great value when a hutch product is desired as in the case of jigging an unsized product. It is a drag, however, in that it cuts down the capacity of the jig enormously. It has been by working along this line that the pulsator jig has been evolved and put in shape for practical operation. The pulsator jig is, in a sense, derived from the pulsator classifier. It is not claimed that this machine does away with the Harz jig. When, however, it is possible to treat sized products, the pulsator jig with four pockets, each 4 in. square, is capable of doing the work of three double four-compartment Harz jigs occupying nearly 70 times the floor space, using five times the water, and five times the power required for the pulsator. One square inch of screen area is moreover capable of doing the work of 200 sq.in. in the case of the Harz jig.

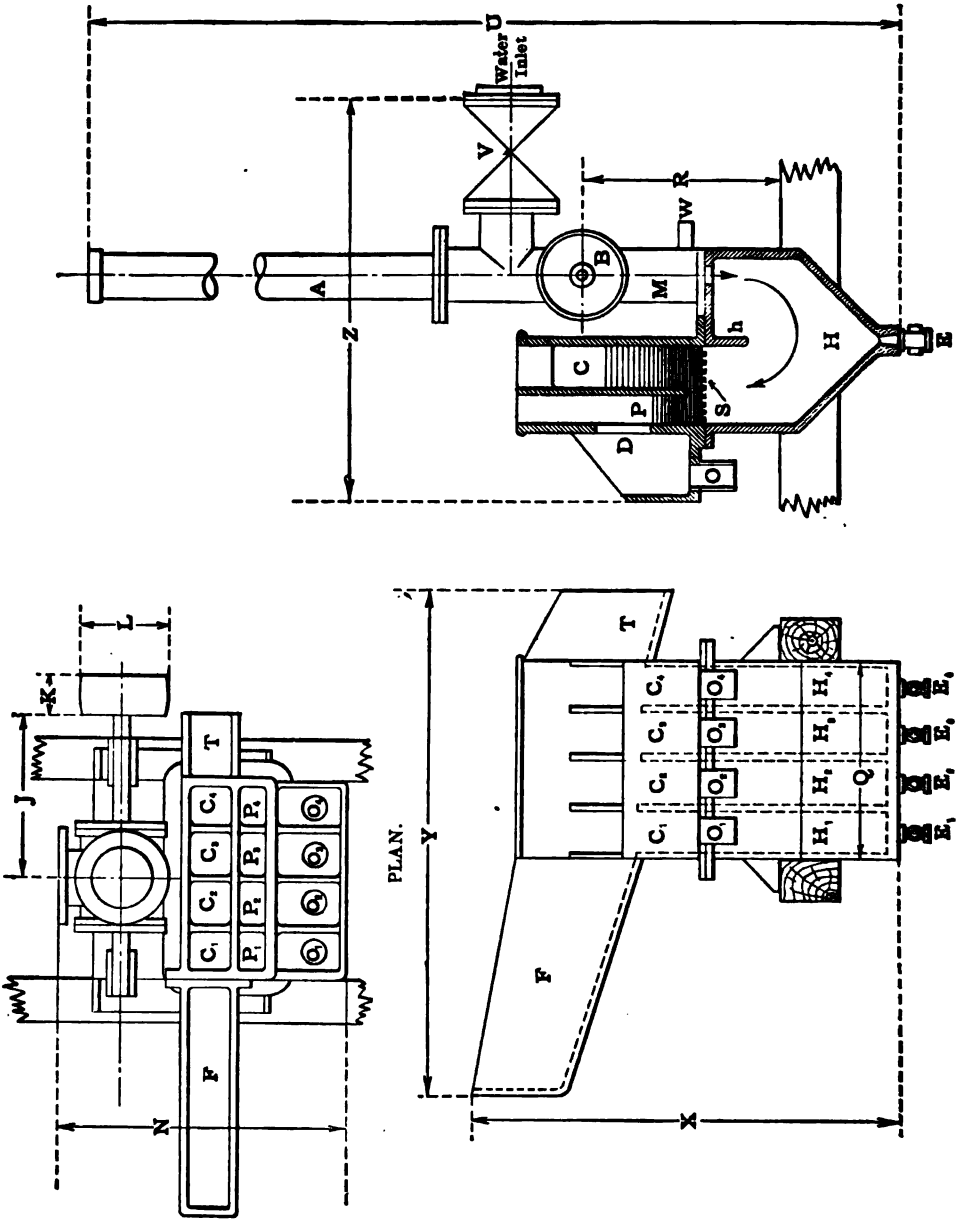
It is not claimed for the Richards pulsator jig that it can do cleaner work than the ordinary jig, but it is claimed that it will use much less water and occupy a marvelously small space compared to its capacity. There is a little crumbling of ore, which goes into the tailings, but when the tailings are to be reground this may all be recovered. The tailings may, in any case, be screened and this saved. In other jigs the particles remain such a long time in the jig before they are finally discharged as concentrates that their corners and edges are rubbed off to a considerable extent, forming a large amount of fine material that is likely to be lost in the tailings.

The sizes, capacities, etc., of the pulsator jig will be found in the following table. The letters J, K, L, N, etc., in the table refer to similarly marked dimensions shown in the figure.

The figure shows the jig in elevation, plan and section. H represents the hutch which is of the familiar form, with diving board *h*.

SIZES, CAPACITIES AND DIMENSIONS OF THE RICHARDS PULSATOR JIG.

Screen Surface Each Compartment		No. of Compartments.	Approximate Capacity in Tons per 24 Hours.	Approximate Gallons Hydraulic Water per Minute.	Dimensions, Approximate only—Inches										Size of Main Water Pipes, Inches
Wide, In.	Long, In.				J	K	L	N	O	R	U	X	Y	Z	
2	2	4	10	8	....	1½	8	18	10	....	58	26	30	22	2
2	2	6	13	10	....	1½	8	18	15	....	58	26	35	22	
3	3	4	40	30	12	3	8	22	14	20	82	34	34	28	2½
3	3	6	55	40	12	3	8	22	21	20	82	34	41	28	
4	4	4	90	70	14	4	8	26½	18½	25	103	42	50	34½	3
4	4	6	120	90	14	4	8	26½	27½	25	103	42	56	34½	
9	5	4	200	150	18	5	10	32	22½	35	112	48	54	45	4
9	5	6	275	200	18	5	10	32	33½	35	112	48	62	45	
14	6	4	400	300	21½	6	12	39½	26½	45	132	54	60	56	6
14	6	6	530	400	21½	6	12	39½	40½	45	132	54	74	56	



PLAN, ELEVATION AND SECTION OF RICHARDS PULSATOR JIG.

The screen is located at S, as in the Harz jig. The compartments C_1, C_2, C_3, C_4 are located above the screen and communicate with corresponding pockets P_1, P_2, P_3 and P_4. The dividing plate between P and C reaches nearly to the screen and acts as a seal or gate for the concentrates which rise in the pockets P and discharge through adjustable gates D into O and thence out to suitable launders.

In place of the usual plungers and eccentrics there is connected at the top of the hutch a manifold, M, by which water supplied through the valve, V, is distributed to each of the four compartments of the jig through four plug cocks connected to the branch flanges of the manifold. The water from the main supply pipe, which should be under a head of 30 ft. or more, passes through the gate valve, V, to the manifold, M, under control of the revolving pulsator valve, B. The valve, B, corresponds in function to the plungers in the ordinary jig, yet in its action is radically different in that it gives pulsations of an entirely different character and in one direction only.

The screen, S, is made up of two layers of brass, woven-wire cloth, the lower one being 4 mesh and the upper one 20 mesh. In consequence of the fine screen used in this jig practically no hutch product is made, all the concentrates being discharged in the pockets P_1, P_2, P_3, P_4 and out through D and O as explained below.

The material fed to the jig enters through the hopper, F, over the division plates between the compartments C_1, C_2, C_3, C_4, and out at the tailings gate, T. In passing through from F to T, the particles are subjected to the jigging action of the upward pulsating current of water through the screen, S, with the result that the heavy particles settle in the compartments C_1 to C_4 and the lighter gangue is carried over and out at the tailings gate, T. The heavy particles settle in C_1 to C_4 in the ratio of their specific gravity, that is, the heaviest mineral will be found in C_1, and the lightest mineral in C_4. The discharge of concentrates or heavy mineral is effected by extending the screen, and consequently the jigging action, across under the pockets, P (see sectional view). All the particles of both mineral and gangue in C are kept in agitation, the mineral being at the bottom and the gangue on top. The mineral at the bottom flows under the division into P by reason of the pressure due to the hight of the column of material in C. Eventually the hight of the material in P would become almost as great as in C, but for the gate at D, which allows the mineral from P to discharge. By adjusting the hight of the gate, D, the concentrates from P are discharged as fast as they accumulate in the bottom of C, the flow from C to P being maintained by the difference in head in C and P.

This very simple method of discharging the concentrates explains also why it is possible with the pulsator jig to feed lean or rich ore, or to stop the feed altogether and then start again without readjustment of the machine. The bed of material, which in C may be as deep as 10 in., will cease discharging concentrates or mineral as soon as the level of the material in C drops to the level of the gate, D. If gangue only is fed to the jig, then it passes over the bed and out at the tailings gate. If mineral and gangue are fed to the machine, the mineral accumulates in C, disturbs the balance between C and P until a sufficient amount of mineral is discharged at D to compensate for that which came into C. The gangue that came in with the feed is carried over and out the tailings gate.

In general, the feed to the pulsator jig should be screen sized to the same ratio of maximum and minimum particles as for jigs of the Harz type. At the present time the smallest sized material treated is 1 mm. and this is being done in a 4-in., 4-compartment jig having a capacity of 90 tons per 24 hours. The maximum size as yet treated is 12 mm. It seems highly possible that 1 in. material may be treated successfully in the larger sized jigs, i.e., 9- and 14-inch.

The depth of the bed carried on the ordinary Harz jig will probably average from 4 to 5 in. The bed carried on the pulsator jig will average 8 to 12 in. This being able to carry such a deep bed makes the performance of the machine less subject to fluctuation, with the result that having once been adjusted for a certain class of ore, this adjustment may remain unchanged while the ore fed to the jig may be lean or rich, or, in fact, may cease altogether, and yet the jig will operate perfectly under all these widely varying conditions. In fact, with some of the first jigs made it was possible to leave the jigs in operation for weeks at a time without changing a single adjustment. The ordinary jig has such a thin bed that the concentrates' discharge is decidedly non-regulative and very sensitive to any change in the richness or the quantity of the feed.

The author has conducted a test upon the single-pocket pulsator jig in the laboratory of the Massachusetts Institute of Technology. This test was made upon a Missouri lead ore with limestone gangue. The material treated was carefully sized through 4 mm. on 2 mm. The results obtained in this test are given in the following table.

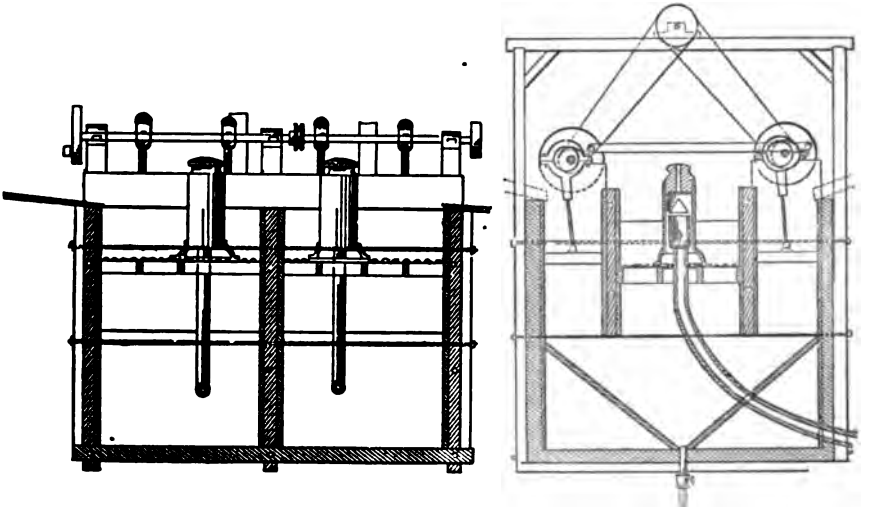
The first crude machine was installed in February, 1908, and in the short time that has elapsed since then the machine has been undergoing constant development. It is at present in successful operation in a number of important mills. Complete data with regard to actual tests in these mills are not, however, at present available.

PULSATOR JIG TEST ON MISSOURI LEAD ORE.

	Weight, kg.	Lead %	Weight Lead, kg.	Per cent. Total Lead
Feed	27.02	11.92	3.220	100.00
Concentrates	3.52	76.12	2.679	83.19
Tailings	23.50	2.30	0.541	16.81
Totals	27.02		3.220	100.00

Note.—All lead assays by permanganate method.

Henry Foust Jig.—This jig, used at the mill of the Virginia Mining Company, Baxter Springs, Kansas, for treating zinc ores, has four sieves each 4x5 ft. in size. For each sieve there are two plunger compartments one on each side instead of only one, as in the common form of jig. Each



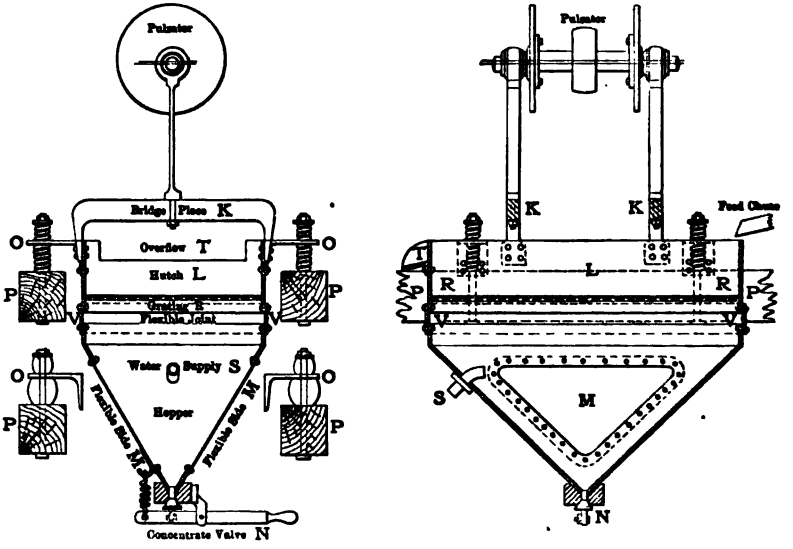
FOUST JIG, LONGITUDINAL AND CROSS SECTIONS.

sieve has a central discharge for coarse concentrates. To the author this appears to be a novel form of jig, which has the advantage of affording a more uniform pulsion all over the bed than the usual single plunger type.[1] The jig is shown in the accompanying figure.

Separation of Metallic Ores by Jigging.—The principle of a new jig designed for treating fine material is shown in the accompanying figure.[2] It consists of a screen box 0.6 to 0.75 m. long by 0.35 to 0.4 m. wide and a hutch beneath, all supported by four lugs, *O*, carried by springs on guide rods. The hutches have flexible sides, *M*, of sheet rubber or strong

[1] *Metallurgie*, V (1908), p. 363. *Eng. and Min. Journ.*, LXXXV (1908), p. 1089.
[2] Arthur Taylor. *Trans.* I. M. M., 1908. *Can. Min. Journ.*, XXIX (1908), p. 597.

canvas or are joined to jig box by flexible connection, *V*. Above the jig box are two bridge pieces, *K*, which support the connecting rods, double eccentrics, shaft, two heavy discs and pulley. Owing to the weight of the shaft with its discs and pulley and to the eccentricity of the eccentrics the revolution of the shaft causes a rapid reciprocating or vibratory motion of the connecting rods and hence of the sieve. Nuts serve to regulate the tension of the springs above and below the lugs, *O*. A bed of shot is used on the screen. Water is introduced into the hutch at *S*. For very fine ores the discs weigh 2.5 kg. and increase to 5 kg. for coarser material. For fine ores the speed is 1500 r.p.m. and the eccentricity may be adjusted up to 1 mm. For coarser ores the speed is 500 r.p.m. and the maximum eccentricity is 4 mm. A vibrator of the same principle may be applied to the ordinary movable-sieve jig or to the plunger of a fixed-sieve jig.



NEW JIG FOR FINE MATERIAL.

Note.—The flexible sides MM and flexible joint V are alternative, and are not employed together.

These jigs have been used at the Calamon mine, near Cordoba, Spain, for treating galena and blende in a gangue of limestone, slate and clay, in sizes under 14 mesh. In this mill eight sets of jigs of the ordinary type treat eight sizes of products, the coarsest of which is 12 mm. while the finest is material that has passed a 14-mesh screen and has settled through the spigot of a spitzkasten. The overflow of the spitzkasten is passed through a series of four large spitzkasten, each of which yields a spigot product to a set of jigs of the new type. These spigots yield

fine products; even in the coarsest, the whole of the galena will pass a 40-mesh sieve and a large proportion of it a 100-mesh, while in the finest product, the galena will all pass a 100-mesh sieve and 50 per cent. of it a 250-mesh. The galena concentrates of the new jigs assay 65 to 75 per cent. lead and 5 to 10 per cent. zinc; the blende concentrates contain 30 to 45 per cent. zinc and 5 to 10 per cent. lead; the tailings assay 2½ to 3 per cent. lead and 3 to 4 per cent. zinc. Nearly all the lead in the tailings is an impalpable powder, for when the tailings are sifted on a 250-mesh sieve the undersize assays as much as 12 per cent. lead and the oversize only 0.5 per cent. The four sets of new jigs (each containing four jigs in series) treat a total of 750 to 1000 kg. per hour.

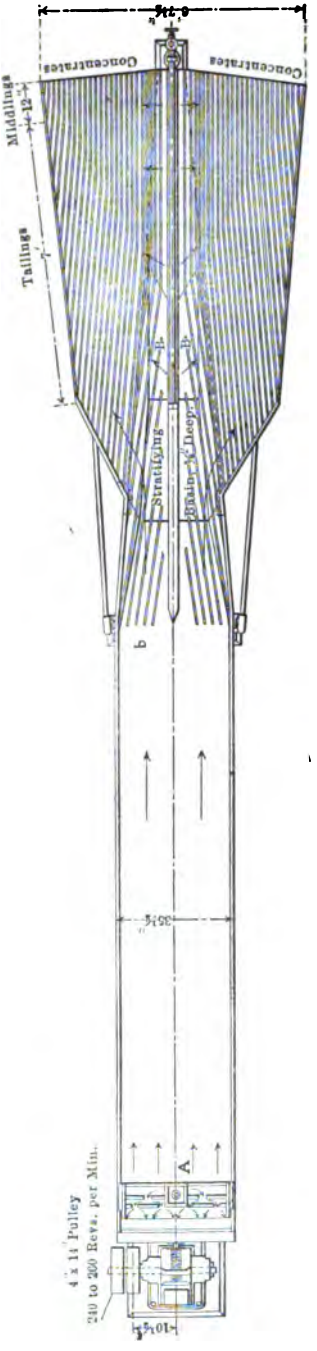
This appears to me to be an important extension in the field of jigs. In present practice the fine limit of jigging is about 1 mm. and the maximum speed is 400 r.p.m.

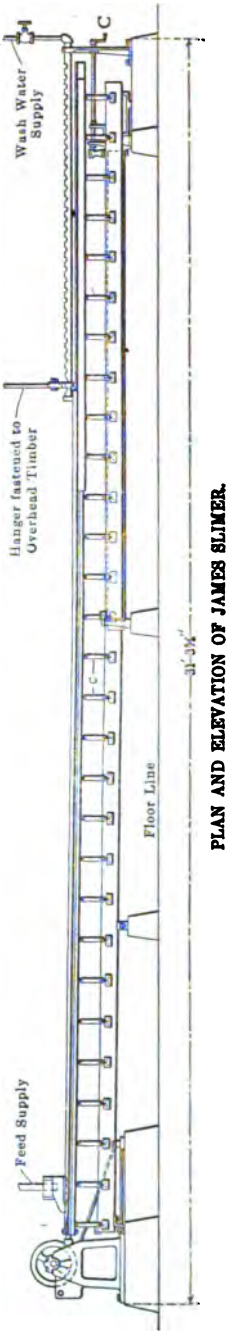
Hennig Concentrating Table.—This table, of the Wilfley type, has a canvas surface coated with asphalt and overlaid with riffles made of thin strips of sheet lead having a wavy form, a feed trough extending the whole length of the table, and a mounting on four ball and socket joints.[1]

James Slimer.—This table, shown in the figure, is of the bumping type.[2] Beginning at the feed end there comes first a longitudinal settling section 18 ft. long covered with a smooth, flat, and slightly inclined linoleum surface 3 ft. wide. Directly behind the settling section is the stratifying basin, ⅝ in. deep, 7 ft. long and gradually decreasing in depth and width toward the lower end. The bottom of this basin is provided with centrally diverging, parallel riffles ¼ in. deep and 2 in. apart. A central strip divides the basin into two parts. The two finishing sections, which are riffled, constitute the remainder of the table, one section being hinged to each side of the stratifying section and both transversely adjustable by one adjusting lever. Pulp fed at the head end is worked forward by the bump, aided by the water, and on the settling section the heavy material and slimes settle so that by the time the water reaches the stratifying basin it is clear enough to be allowed to flow off transversely. The remainder of the material continues on and spreads out into concentrates, middlings and tailings under the action of the central wash water trough.

Deister No. 3 Table.—This is a jerking table of the Wilfley type with a rectangular top.[3] This top is largely without riffles, particularly the first section, next to the feed box. The section at the tailings end of the table has very shallow riffles and also an increased downward slant.

[1] *Eng. and Min. Journ.*, LXXXVI (1908), pp. 134, 1198.
[2] M. T. Hoster, *Eng. and Min. Journ.*, LXXXVI (1908), p. 1149.
[3] *Eng. and Min. Journ.*, LXXXVI (1908), p. 610.





PLAN AND ELEVATION OF JAMES SLIMER.

The shortest dimension of the table is parallel to the direction of the bump.

Test of Deister Table and Suspended Vanner.—Following are average results of all tests at the Combination mill of the Goldfield Consolidated Mines Company during May and June, 1908[1]:

COMPARISON OF DEISTER TABLE AND VANNER.

	Deister Table.	Vanner.
Total solids in feed, per run. Tons	33.75	39.4
Total solids in 24 hours. Tons	8.90	10.006
Value of feed. Oz. gold per ton	0.97	0.97
Total value of feed. Oz. gold	32.736	39.40
Value of concentrates. Oz. gold per ton	14.27	17.20
Extraction by product. Per cent.	41.89	32.51
Extraction by assay of feed and tailings. Per cent	40.10	35.85
Flow to Callow tank. lb. per min	137.7	.....
Dry solids. lb. per min	16.36	.....
Water. lb. per min	115.5	.....
Overflow of clear water. Per cent	48.15	.....
Solids in discharge. Per cent	22.62	.....

Cement Tables.—These have come into use to supplant canvas tables in some of the mills at Guanajuato, Mexico.[2] At the Pinguico mill of the Guanajuato Development Company the ore, which has been through two Blake breakers, 40 stamps (2- to 8-mesh screens) and two Chili mills with 30-mesh, Tyler, rolled, slot screens, flows to two large spitzkasten. The spitzkasten spigots are concentrated on Wilfley tables whose middlings are recrushed in a Bryan mill and reconcentrated. The Wilfley tailings are separated into sand and slime, the former going to the cyanide plant and the latter to the cement table.

The overflow of the two large spitzkasten also goes to the cement table. This is 35 ft. long (in the direction of the flow) and is divided in its width into 29 sections each 6 ft. wide. The table was built on the hillside out of concrete covered with a thin layer of clean cement which was roughened into cross riffles $\frac{3}{8}$ in. deep and $1\frac{3}{4}$ in. wide. The table has a slope of $1\frac{1}{2}$ in. per foot and at distances of 8.5 and 10 ft. slats 1 in. high have been placed in order to help retard an unexpected flow of sand and slime. When a load of concentrates has accumulated on the table the flow is shut off from one section at a time and the concentrates hosed down to a sand pump lifting to a classifying cone. This cone sends stuff to Wilfley and Johnson tables. The tailings from the cement table go to cyanide plant.

Lake Superior Ore Dressing Practice.—Steam Stamps.—The tendency is towards the increased use of the compound head, because of its economy in steam.[3] Its use is not, however, increasing as fast as was at first antici-

[1] *Can. Min. Journ.*, XXIX (1908), p. 399.
[2] C. T. Rice, *Eng. and Min. Journ.*, LXXXVI (1908), pp. 951, 997.
[3] L. S. Austin. *Min. and Sci. Press*, XCVI (1908), p. 259.

pated. Its disadvantages are: More oil and waste needed, additional supplies for repairs, and the delays caused by repairs and break-downs. The labor cost is also higher than with the simple-head stamps. The Quincy has installed a 24-in. simple cylinder in place of the usual 20-in., has dispensed with rolls for regrinding the oversize from the trommels, and has increased the capacity of the head to 700 tons in 24 hours. The new installation at the Champion mill includes compound heads, and three have been added at the Centennial mill.

Rolls.—Rigid rolls are preferred; but at the Champion, spring rolls, arranged for a constantly parallel opening, are used. Where rigid rolls are employed a grating in the feed chute excludes any hard, unyielding object. The main driving pulley is fitted with an iron key, replacing the wooden ones formerly used, which sheared so readily. They are fed with nothing larger than $\frac{5}{8}$-in. material and are estimated to relieve the stamps to the extent of 200 tons daily in their duty of crushing the oversize, of $\frac{5}{16}$-in., from the trommels.

Recrushing Middlings.—The Chili mill has been adopted at the Calumet & Hecla new regrinding plant. The copper in this conglomerate rock is so closely knit that fine grinding gives the best results even when accompanied by some sliming. Considering the cheapness of the wearing surfaces in tube mills it is suggested that they might be used here with a saving for repairs and castings. They have never been tried.

Jigs.—Arranged as a series-jig the Woodbury jig has a large capacity, acts as a screening system, and separates out an overflow for classification. It is being installed at the Champion after being thoroughly tried in the Calumet & Hecla mill.

Classifiers.—The classifiers used are imperfect machines. The roll product is conducted to single compartment classifiers, having a pyramidal bottom, and using hydraulic water. The overflow goes to tables, the spigot discharge to jigs. The hutch product from these jigs goes to finishing jigs. The four-spigot Tamarack classifier is imperfect in that any given spigot discharge is contaminated with coarser and finer grains properly belonging in another product.

Tables.—The Wilfley table holds its own. It, or some equivalent table of the same type, is used for concentrating Evans table concentrates so as to raise the grade of the product. Schwartz has devised a roughened or grooved sheet lead covering for the Wilfley table. These grooves are shallow, 20 to the inch, and run parallel to the tailings and feed sides of the table. They assist in the elimination of the finer gangue, and in bringing the fine concentrates to the concentrates side of the table. The grooves, however, are disposed to fill up and coat over, and it is proposed to overcome the defect by copper-plating the surface.

At the Champion mill compound stamps are employed in connection with rolls, Woodbury jigs, large settling tanks with goose-neck discharge, and Deister tables.

Concentrator of the Steptoe Valley Smelting and Mining Company.—This mill, at McGill, Nev., has a total capacity of 4000 tons per 24 hours and is divided into three units of two sections each.[1] The plant handles the ores from the Cumberland Ely Copper Company's mines and from the mines of the Nevada Consolidated Copper Company in about equal proportions. The Cumberland Ely ore contains the minerals chalcocite, pyrite, chalcopyrite, melaconite, magnetite and limonite, as well as gold and silver values occurring in a quartz porphyry gangue. This crude ore will assay about as follows: 3.49 per cent. copper, 16 per cent. iron, of which 50 to 67 per cent. is in the form of pyrite and the remainder mostly as magnetite with a little limonite, 0.01 oz. gold and 0.2 oz. silver. The Nevada Consolidated ore is of a similar nature and occurrence with the following exceptions: In the crude state it will assay 2.3 per cent. copper and 4 per cent. in iron which is wholly in the form of pyrite, no magnetite or limonite being present.

The economic minerals, in each case, are evenly and finely disseminated throughout the gangue, which necessitates fine crushing. The problem is to save the copper, gold and silver values.

Each unit is composed of two similar sections. One section only will be described. The Cumberland Ely ores are screened at the mine while the Nevada Consolidated ores are screened at the concentrator. Ore from the mine goes to (1).

1. Shaking grizzly, 4 by 7 ft., with 2.125-in. spaces between the bars which have a slope of about 18.5 deg. and make 60 throws per min.; delivers oversize to (2) and undersize to (3).
2. Two No. "7.5" McCully gyratory breakers; deliver crushed ore to (3).
3. 18-in. belt conveyor. From (1), (2) and (5); delivers, via automatic distributor, to (6).
4. Ore bin, 30.5 ft. wide, 19.25 ft. deep, 189 ft. long. For ore from the Cumberland Ely mines; delivers, via cast iron gates, into a mounted hopper which carries an electrically driven Challenge feeder. The hopper and feeder travel the length of the bin and discharge to (5).
5. 18-in. Robins belt conveyor running at right angles to (3). From (4); delivers to (3).
6. Automatic sampler. From (3), delivers sample to assayer and rejects to (7).
7. Distributing bin, 11 ft. wide, 10 ft. deep, 32 ft. long, common to two sections. From (6); delivers, via automatic Challenge feeder making 7 r.p.m., to (8).
8. Trommel, 42 x 84 in. with 0.625-in. round holes and making 18 r.p.m. Driven by worm and wheel. From (7); delivers oversize to (9) and undersize to (10).
9. Allis-Chalmers "Style A" rolls, 15 x 36 in., making 81 r.p.m. and using Midvale steel shells having a life of about 1 year. From (8); deliver crushed ore to (12).
10. Trommel, 42 x 72 in., with 0.375-in. round holes and making 18 r.p.m. Driven by worm and wheel. From (8); delivers oversize to (11) and undersize to (12).
11. Allis-Chalmers "Style B" rolls, 14 x 36 in., making 98 r.p.m. and using Midvale steel shells having a life of about 1 year. From (10) and (13); deliver crushed ore to (12).
12. Two 20-in. elevators with 10-ply belts, speeds of 365 ft. per minute and malleable iron buckets, 6 x 7 x 18 in., set 20 in. apart. One is used and one held as a spare. From (9), (10), (11), (15), (17), (18) and (19); deliver to (13).
13. Trommel, 42 x 42 in., divided into two sections with 0.1875-in. and 0.3125-in. openings respectively, and making 18 r.p.m. Driven by worm and wheel. From (12); delivers, material larger than 0.3125-in. to (11), 0.3125 to 0.1875-in. material to (15) and material through 0.1875-in. to (14).
14. Eight trommels, 3 x 6 ft., with 2-mm. round holes and making 18 r.p.m. Driven by worm and wheel. From (13); deliver oversize to (19) and undersize to (20).
15. Three Traylor jigs with brass-cloth sieves, 20 x 30 in., of 5-mesh, No. 14 wire. The plungers make 170 strokes per min. From (13); deliver discharges to (40), hutch products, if rich enough to (40) and if not rich enough to (12) and tailings to (16).

[1] R. L. Herrick. *Mines and Minerals*, XXIX (1908), p. 167.

16. Dewatering screen, 3 x 3 ft., with 2-mm. round punched holes and making 18 r.p.m. From (15) and (19); delivers oversize to (17) and undersize to (18).
17. Allis-Chalmers "Style B" rolls, 14 x 36 in., making 114 r.p.m. and using Midvale steel shells having a life of about 1 year. From (16); deliver crushed ore to (12).
18. V-shaped settling tank, 3.5 ft. wide, 4 ft. deep, 7 ft. long. From (16); delivers settlings to (12) and overflow to waste.
19. Nine Traylor jigs with brass-cloth sieves of 8 mesh, No. 17 wire. From (14); deliver discharges to (40), hutch products, if rich enough to (40) but if not rich enough to (12) and tailings to (16).
20. Eight two-spigot Richards vortex classifiers. From (14); deliver the first spigots to (21), second spigots to (22) and overflows to (26).
21. Nine No. 5 Wilfley tables making 240 thrusts per min. From (20); deliver concentrates to (40), slimy head waters to (23) or waste depending on the ability of the tanks to settle the water, and tailings to (23) or waste.
22. Ten No. 5 Wilfley tables. From (20); deliver concentrates to (40), slimy head waters to (23) or waste depending on the ability of the tanks to settle the water, and tailings to (23) or waste.
23. Four 8-ft. Callow tanks. From (21) and (22); deliver spigots to (24) and overflows to waste.
24. Four 6-ft. Huntington mills crushing through slotted, punched screens with 20 meshes to the inch and making 65 r.p.m. Only two or three of these mills are in use at one time. From (23) and (27); deliver pulp to (25).
25. Two 18-in. elevators with 8-ply belts having speeds of 375 ft. per min. and malleable iron buckets, 5 x 6 x 16 in., set 2 ft. apart. Only one used, one being held as a spare. From (24); deliver to (26).
26. Two dewatering tanks. From (20) and (25); deliver pulp to (27) and overflows to (28).
27. Five double Callow screens with 30 meshes to the inch. From (26); deliver oversize to (24) and undersize to (28).
28. Four 4-compartment spitskasten. From (26) and (27); deliver the first and second spigots to (29), third spigots to (30), fourth spigots to (31) and overflows to (32).
29. Eight No. 5 Wilfley tables. From (28); deliver concentrates to (40), slimy head waters to (35) and tailings to (33).
30. Four No. 5 Wilfley tables. From (28); deliver concentrates to (40), slimy head waters to (35) and tailings to (33).
31. Eight No. 5 Wilfley tables. From (28); deliver concentrates to (40), slimy head waters to (35) and tailings to (33).
32. Ten 8-ft. Callow tanks. From (28); deliver spigots to (34) and overflows to waste.
33. Eight 8-ft. Callow tanks. From (29), (30) and (31); deliver spigots to (36) and overflows to waste.
34. Twenty No. 5 Wilfley tables. From (32); deliver concentrates to (40), slimy head waters to (35) and middlings and tailings to (35).
35. Eight 8-ft. Callow tanks. From (29), (30), (31) and (34); deliver spigots to (37) and overflows to waste.
36. Sixteen Allis-Chalmers 6-ft. Frue vanners with corrugated belts. Make 190 throws per min. From (33); deliver concentrates to (40) and tailings to waste.
37. Sixteen Allis-Chalmers 6-ft. Frue vanners with smooth belts. Make 190 throws per min. From (35); deliver concentrates to (40) and tailings to waste.
38. Eight 8-ft. Callow tanks. From (34); deliver spigots to (39) and overflows to waste.
39. Sixteen Allis-Chalmers 6-ft. Frue vanners with smooth belts. Make 190 throws per min. From (38); deliver concentrates to (40) and tailings to waste.
40. Two steel concentrates bins, 28 ft. diameter and 6 ft. deep with cocoa matting in the bottoms to act as filters. From (15), (19), (21), (22), (29), (30), (31), (34), (36), (37) and (39); deliver, via Blaisdell excavator, through centers into 10-ton hopper-bottomed steel cars which go to the smelter. The water goes to waste.

There are two tailings launders passing along the ends of the unit and each attached to the automatic sampler, so that different ores may be run in the unit at the same time. The automatic sampler delivers the sample to the assayer and rejects to waste.

Ultimately the mill is to have ten units. The location is a sloping hillside and the floors are arranged on terraces. The building is of structural steel and the walls are 1-in. boards overlaid with corrugated iron. The floors are of reinforced concrete and are supported on concrete piers. Electric power is used throughout. No reliable figures are yet available but it is safe to expect over 70 per cent. extraction.

Concentrating at Cobalt, Ontario.—There are five mills: Buffalo, Coniagas, Cobalt Central, McKinley-Darragh-Savage and the Muggley.[1] The fourth is an experimental plant not now in operation; the fifth is a custom mill. In the mines, after sorting out the high-grade ore, a residue is left consisting of a mixture of high-grade ore, rock and ore of low value. This residue goes to the mills.

The Buffalo mill receives the ore in (1).

[1] G. E. Sancton. *Journ.* Can. Min. Inst., XI (1908), p. 340. *Can. Min. Journ.*, XXIX (1908), pp. 458, 632. A. A. Cole, *Can. Min. Journ.*, XXIX (1908), p. 168; Ontario Bureau of Mines Report, XVI, Part II (1907), pp. 70, 75, 77, 97. *Mines and Minerals*, XXIX (1908), p. 200.

1. Bin. To (2).
2. Grizzly with 1-in. spaces. Oversize to (3); undersize to (6).
3. Blake breaker, 6 x 20 in., crushing to ¾ in. To (4).
4. Elevator. To (5).
5. Trommel with three screens ⅝, ½ and ¼-in. Over ⅝-in. to (8); through ⅝ on ½-in. to (9); through ½ and over ¼-in. to (9); through ¼-in. to (6).
6. Impact screen, 20-mesh. Oversize to (10); undersize to (7).
7. Cone settler. Settlings to (11).
8. Rolls, 20 x 20 in. To (4).
9. Two Harz jigs each with three compartments. Concentrates; tailings to (13).
10. Harz jig with three compartments. Products like (9).
11. Wilfley table. Concentrates; middlings returned to table; coarse tailings to (12).
12. Deister slime table. Concentrates; tailings.
13. One 6-ft. Chili mill with 20-mesh slotted screen. To (14).
14. Impact screen, 80-mesh. Oversize to (15); undersize to (16).
15. Four Deister tables. Concentrates; tailings.
16. Deister slime table. Concentrates; tailings.

The mill treats 40 to 50 tons in 24 hours. The tailings are stored for possible future treatment by cyanide.

The Cobalt Central mill receives the ore in (1).

1. Ore bin. To (2).
2. Blake breaker, 10 x 20 in., crushing to 1½ in. To (3).
3. Mill bin. By plunger feeder to (4).
4. Roughing rolls, 42 x 14 in. By elevator to (5).
5. Trommel, 2-mesh. Oversize to (6); undersize to (7).
6. Oversize bin. When this is full the feed from (3) is shut off and the oversize material is fed to (4).
7. No. 1 Centripact screen with 8-mesh holes. Oversize to (8); undersize to (14).
8. Two Harz jigs. Concentrates; tailings of one jig unwatered by screen to (9); tailings of other jig unwatered by screen to (10); excess water to (15).
9. Rolls, 10 x 32 in. To (11).
10. Rolls, 14 x 30 in. To (11).
11. Unwatering screen. Oversize to (12); undersize to (15).
12. Elevator with buckets 7 x 12 in. To (13).
13. No. 2 Centripact screen with 6-mesh holes. Oversize dewatered to (10); water to (15); undersize to (14).
14. No. 3 Centripact screen with 16-mesh holes. Oversize to (9); undersize to (15).
15. Two 20-in. hydraulic classifiers. Spigots to (17); overflow to (16).
16. Two Callow settlers. Spigots to (18); overflow to waste.
17. Four James tables. Concentrates; middlings to (19); tailings to waste.
18. Two James tables. Products like (17).
19. One James table. Concentrates; tailings to waste.

The mill treats 50 to 60 tons in 24 hours. About 70 per cent. of the total extraction is made on the jigs. The tailings of the James tables average not over 4 or 5 oz. silver per ton.

The Coniagas mill receives the ore in (1).

1. Bin. To (2).
2. Breaker, 10 x 16 in., crushing to 2 in. By elevator to (3).
3. Grizzly with 1-in. spaces. Oversize to (4); undersize to (5).
4. Breaker 7 x 13 in. crushing to 1½ in. To (5).
5. Bin. To (6).
6. No. 1 rolls, 10 x 30 in., crushing to ¾ in. By elevator to (7).
7. No. 1 trommel with ¾ and ⅜-in. holes. Over ¾-in. to (6); through ¾ on ⅜-in. to (10); through ⅜-in. to (8).
8. No. 2 trommel with 3-mm. screens. Oversize to (14); undersize to (9).
9. Classifier. Spigots to (16); overflow to (17).
10. Two Harz jigs. Concentrates; tailings to (11).
11. No. 2 rolls, 10 x 30 in., crushing to ¾ in. By elevator to (12).
12. No. 3 trommel with ¾ and ¼-in. holes. Over ¼-in. to (11); through ¾ on ¼-in. to (13); through ¼-in. to (8).
13. Ball-mill with 20-mesh screen. To (18).
14. Fine jigs. Concentrates; tailings to (15).
15. One 5-ft. Huntington mill with 20-mesh slotted screens. To (18).
16. Wilfley table. Concentrates; tailings to (15).
17. Frue vanner. Concentrates; tailings to (15).
18. Classifier. Spigot to (20); overflow to (19).
19. Callow settling tank. Settlings to (22); overflow to waste.
20. Four No. 2 Deister tables. Concentrates; middlings to (21); tailings to waste.
21. Wilfley table. Concentrates; tailings to waste.
22. Frue vanner. Concentrates; tailings to waste.

At the *Muggley mill* the ore is fed to (1).

1. No. 4 K Gates breaker. To (2).
2. Gates rolls crushing to ¾ in. To (3).
3. Screen with 1¼-in. holes. Oversize to (2); undersize to (4).

4. Jig with two compartments. Concentrates; tailings by belt elevator to (5).
5. Bins. By Challenge feeders to (6).
6. Twenty stamps, 1250 lb. each, with about 20-mesh screen. By elevator to (7).
7. Richards annular vortex classifiers. Spigots to (9); overflow to (8).
8. Two 8-ft. Callow tanks. Settlings to (10); overflow to waste.
9. Four Wilfley tables. Concentrates; middlings to (11); tailings to waste or re-treated if of sufficient value.
10. Frue vanners with corrugated belts. Concentrates; tailings to (11).
11. Eight-foot settling tanks. Settlings to (12); overflow to waste.
12. Eight-foot amalgamating pans followed by two 8-ft. settlers. Amalgam retorted; tailings to waste.

The writer states that the ores do not slime badly and a good extraction is obtained. The ore is hard to crush fine and therefore stamps are attractive on account of their simplicity and ability to do this work at a low cost. Cyaniding has yet to prove its adaptability to the treatment of tailings. There is little doubt that it will dissolve the silver but the time and cost of treatment may be prohibitive. The following is suggested as the best form of mill; Gates breaker, rolls, bull jig, stamps for recrushing jig tailings, tables, vanners, amalgamating pans.

[*Note.*—My experience with Cobalt ores has indicated that enough large nuggets of silver occur in the low-grade ores to warrant the use of jigs to keep this silver out of the stamp mill. My experience with steam stamps on Lake Superior native copper ores indicates that there is bound to be a loss by abrasion of silver nuggets in the stamp mill. This may be reduced by the application of the so-called Krause mortar discharge to stamps, and moreover if the mill tailings are subjected to cyanide treatment this abraded silver would be recovered.—R. H. Richards.]

Veta Colorada Mill, Parral, Mexico.—The ore is highly silicious and contains less than 1 per cent. lead, 1 per cent. zinc, a trace of copper and 2 to 3 per cent. iron pyrites. The value is in silver in the form of argentite and cerargyrite.[1] The ore goes to (1).

1. Bin holding 800 tons. To (2).
2. Two grizzlies with 2½-in. spaces. Oversize to (3); undersize to (4).
3. Two Blake breakers, 24 x 12 in. To (4).
4. Two grizzlies with 1½-in. spaces. Oversize to (5); undersize to (6).
5. Two Blake breakers 9 x 15 in. To (6).
6. Robins belt conveyor, 18 in. wide, 280 ft. long. To (7).
7. Sampling system and Blake-Denison weighing machine. To (8).
8. Stamp bins holding 1150 tons. By Challenge feeders to (9).
9. Ninety stamps, 1050 lb. each, 7-in. drop, 8 or 10-mesh screen, concrete mortar blocks, capacity 6½ tons per stamp per 24 hours. To (10).
10. Sixteen cone classifiers 14 x 16 in. Spigots to (12); overflow to (11).
11. Sixteen cone classifiers, 24 x 30 in. Spigots to (13); overflow to (16).
12. Eight Wilfley tables. Concentrates to smelter; tailings to (14).
13. Eight Card tables. Products like (12).
14. Eight Wheeler pans, 5 ft. diameter and 3 ft. deep, each grinding 25 tons in 24 hours. To (15).
15. Two bucket elevators (one in reserve). To (16).
16. Four cone classifiers. Spigots to (17); overflow to cyanide plant.
17. Five tube-mills 14 ft. long, 5 ft. diameter, chilled steel liners 1¼ in. thick. To (15).

San Diego Mill, Santa Barbara, Mexico.—The sulphide ores of this district carry about 250 grams of silver per metric ton, 2 to 8 grams of gold, 5 to 10 per cent. lead, 10 to 20 per cent. zinc, 6 to 7 per cent. iron, 0.5 to 3 per cent. copper and 30 to 40 per cent. silica.[2]

Lead ore is treated at the San Diego mill by breakers, picking belt,

[1] C. T. Rice. *Eng. and Min. Journ.*, LXXXVI (1908), p. 120.
[2] C. T. Rice. *Eng. and Min. Journ.*, LXXXVI (1908), p. 207.

rolls, trommels, classifiers, jigs, Card tables, Wilfley tables and Frue vanners.

Zinc ore is crushed by Blake breaker, hand picked to remove smelting ore and waste, and further crushed in Blake breaker and rolls to 1.5 mm. This ore, together with the zinc-iron middlings from the lead ore is treated on magnetic separators before being concentrated on jigs, tables and vanners. It contains 23 to 30 per cent. zinc, in the form of magnetic blende, 4 per cent. lead and 8 to 9 per cent. iron in the form of pyrite. It is dried (not roasted) in a Bartlett-Snow cylindrical drier and fed to two units of International magnetic separators with four separators in series in each unit. These remove the zinc and the residue goes to wet concentration which yields lead and iron concentrates and some zinc-iron middlings to go back to the magnetic treatment. The zinc extraction varies from 50 to 85 per cent. according to the grade of the ore. The concentrates assay 40 to 44 per cent. zinc. When making an extraction of 51 per cent. the first magnet saves 28 per cent., the second 14 per cent., the third 6 or 7 per cent., and the fourth 2 to 3 per cent.

Concentration in New South Wales.—At the Paramatta copper mines, at Moonta, the ore, containing bornite, chalcopyrite and pseudomorphic hematite, is passed over a grizzly and the oversize is hand picked into smelting ore, concentrating ore and waste.[1] The undersize is elevated to a trommel with $\frac{3}{4}$-in. holes. Over $\frac{3}{4}$-in. stuff is hand picked on a picking belt made of lengths of rope strapped with iron, into the above three classes. Concentrating ore together with stuff under $\frac{3}{4}$ in. goes to the mill.

At the Wallaroo mines the ore is chalcopyrite and iron pyrite and the country rock is metamorphic mica schist. At the Richman mill this ore is received in (1).

1. Storage bins. To (2).
2. Gates breaker crushing to 2 in. Product through automatic sampler to (3).
3. Trommel with $\frac{1}{2}$- and $\frac{1}{4}$-in. holes. Oversize to (4); undersize to (5).
4. Picking belt. Rich ore to smelter; waste to fill mine stopes; residue to bin and thence to (6).
5. Bin. Some rich ore picked out to smelter; residue to (6).
6. Blake breaker. To (7).
7. Grizzly. Oversize to (8); undersize to (9).
8. Rolls. To (9).
9. Trommel with $\frac{1}{4}$-in. holes. Oversize elevated to (8); undersize by wheel elevator to (10).
10. Slime separator. Spigot to (11); overflow to (12).
11. Hancock jig. 1st and 2d compartment concentrates to smelter; 3d returned to jig; 4th and 5th to leaching plant; overflow to (12).
12. Classifier with three spigots. Spigots to (13); overflow to (14).
13. Three Wilfley tables. Concentrates to smelter; tailings to (14).
14. Settling pits. Settlings to (15); overflow to water tank.
15. Grating to remove stones, etc., in oversize. Undersize to (16).
16. Jarrah rolls. To leaching plant.

The North Broken Hill mill treats the ore as follows:

1. Gates breaker. To (2).
2. Bin. By feeder to (3).

[1] F. W. Sewell. Austral. Inst. of Min. Eng., XII (1907), p. 105.

3. Rolls. To (4).
4. Trommel. Oversize to (5); undersize to (5).
5. Cone classifiers. Spigot to (6); overflow to (10).
6. Hancock jig. 1st and 2d compartments galena; 3d returned to jig; 4th and 5th to (7); tailings to dump.
7. Heberli mills. To (8).
8. Classifiers. Spigot to (9); overflow to (10).
9. May compound jig. 1st and 2d compartments galena; 3d, 4th and 5th returned to jig; tailings to dump.
10. Classifier with five spigots. First four spigots to (12); 5th to (13); overflow to (11).
11. Settling pit. Slimes to dump; overflow to water tank.
12. Four Wilfley tables. Concentrates galena; middlings returned to tables; tailings to dump.
13. Luhrig vanners. Concentrates galena; middlings returned to vanners; tailings to dump.

The Broken Hill Central concentrating mill uses wet concentration, dry magnetic and the Cattermole granulation process. The mine ore is crushed in a Gates breaker and transported by aerial tram to (1).

1. Blake breaker. To (2).
2. Rolls. To (3).
3. Trommel. Oversize to (2); undersize to (4).
4. May compound jig. 1st and 2d compartments galena; 3d and 4th to (5); 5th to dump.
5. Ball mill. To (6).
6. Classifiers. 1st spigot to (7); 2d to (8); overflow to (14).
7. May compound jig. 1st and 2d compartments, galena; 3d and 4th returned to jig; 5th to (9).
8. Wilfley tables. Concentrates galena; tailings to (9).
9. Settler. Settlings to (10); water to tanks.
10. Rotary drier. To (11).
11. Ball mill, crushing to 1 mm. To (12).
12. Trommel with 1- and ½-mm. holes. Dust to vanners. Sized product to (13).
13. Six Mechernich magnetic separators arranged in three sets with two in series in each set. Blende to smelter; quartz to (5); rhodonite to mine stopes.
14. Slime settlers. Settlings to (16); overflow to water tank to go to (2).
15. Tailings from old dump are crushed in ball mill and go to (16).
16. Mixing vats with oil and soap. To (17).
17. Separator vat. Tails to (18); oil and blende overflow to (19).
18. Settling pits. Settlings to waste; overflow to (20).
19. Settling boxes. Blende to smelter; overflow oil and water to (20).
20. Sump. Repumped to (16).

In the above magnetic process, 21,000 tons of middlings assaying 27 per cent. zinc, 8 per cent. lead and 9 oz. silver per ton yielded concentrates with 39 per cent. zinc, 12 per cent. lead and 13.4 oz. silver per ton. In the Cattermole process the powdered ore is agitated with oil in emulsion with water containing an alkaline emulsifying agent. Oil, soap and sulphuric acid are used in the bath.

Milling at Pierrefitte, France.[1]—The ore occurs as argentiferous galena and blende with a small amount of pyrite sometimes cupriferous, in a gangue of quartz and schistose matter. The mill treats 10 tons per hour, according to the following scheme:

PICKING SECTION.

1. Grizzly with 40-mm. spaces. Oversize to (2); undersize to (7).
2. Blake breaker with opening 24 in. wide, 200 r.p.m., crushing to 45 mm. To (3).
3. Bin. By percussion feeder to (4).
4. Trommel with 25-mm. round holes. Oversize to (5); undersize to (10).
5. Jeffrey picking belt 50 ft. long, 30 in. wide, speed 1 ft. per second. Rich stuff to smelter; residue to (6).
6. Bin holding 100 tons. By feeder to (11).
7. Bin holding 40 tons. By percussion feeder to (8).
8. Trommel like (4). Oversize to (9); undersize to (10).
9. Belt like (5). Products like (5).

JIGGING SECTION.

10. Trommels and spitzlutten followed by seven jigs. Concentrates to smelter, rich lead-zinc middlings to (13); poor middlings to (15); tailings to waste; fine sands and slimes to (16).
11. Rolls 950 x 320 mm., 41 r.p.m. To (12).
12. Trommels and spitzlutten followed by six jigs. Oversize of 12 mm. to (11); concentrates to smelter; rich lead-zinc middlings to (13); poor middlings to (15); tailings to waste; fine sands and slimes to (16).

[1] W. W. Van Ness. *Bull.* A. I. M. E., Nov., 1908, p. 935.

13. Rolls, 700 x 300 mm., 58 r.p.m., crush middlings ranging from 18 to 3 mm. and a No. 3 Heberli ball mill crushes middlings below 3 mm. To (14).
14. Trommels, spitzlutten and jigs. Products probably much like (10) except fine sands and slimes to (17).
15. Rolls like (13) and a No. 4 Heberli ball mill crush middlings above and below 3 mm. respectively. To (13).

SANDS AND SLIMES SECTION.

16. Five Ferraris tables and two double Luhrig vanners. Concentrates to smelter; rich middlings to (18); poor middlings to (19); tailings to waste.
17. Two Ferraris tables and three double Luhrig vanners. Products like (16).
18. Two Ferraris tables and two double Luhrig vanners. These make finished products and also rich middlings pumped back and poor middlings pumped to (19).
19. Two Ferraris tables and one double Luhrig vanner. These make products like (18).

The No. 3 ball-mill was charged with 60 steel balls of 125-mm. diameter and 30 balls of 100-mm. diameter. The No. 4 mill had 80 balls of 125-mm. and 40 balls of 100-mm. All trommels are conical and mostly of punched plate except the finest, which are of steel wire. Jigs are of Harz type except three coarse jigs which have a crank-arm mechanism. Jig middlings are taken away in cars and lifted by a balanced platform elevator. The Ferraris tables are fed from spitzlutten and the Luhrig vanners from spitzkasten. Water comes in by flume from a small stream. The mill requires 3 cu.ft. per second. Coarse tailings are delivered by iron launder to form dams behind which the slimes are discharged to prevent pollution of the river.

Electric power is used and is distributed as follows, in horsepower:

DISTRIBUTION OF POWER, PIERREFITTE MILL.

	Capacity of Motors.	Average Used.
Rolls, ball mills and picking belts	100	40 - 60
Rockbreaker	28	10 - 20
Four coarse jigs and five trommels	18	17 - 18
Remainder of jigs and trommels	18	16 - 18
Vanner plant (two motors)	36	28 - 34
Machine shop	5	4
Lighting	18	12

The mill is lighted by Cooper-Hewitt lamps which have proved very satisfactory. The fine mineral especially on tables is more sharply defined than in daylight. The extraction in the mill is well over 80 per cent. of the zinc and about 60 per cent. of the lead. The blende always contains about 4 per cent. lead. The lead concentrates assay about 67 per cent. lead and 28.05 oz. gold per ton. The zinc concentrates contain about 45 per cent. zinc. The milling costs about $0.44 per ton, not including depreciation. The labor required on the day shift is 36 men and 18 boys on the picking belts.

Milling in Hungary.—Government mills in this district seem to be as efficient as the average mill in other countries treating ore of the same character, i.e., containing gold, silver, galena, blende, antimony, pyrite and quartz. The rock surrounding the vein is a quartz-trachyte

covered with andesitic tuff. The miners are expected to pick out massive galena and blende which is sent direct to the smelters.[1]

The Cross mill is built on a level site, as are all the mills in the district, the building being of one story, long and narrow. In the mill are four sets of three batteries of California stamps, hand-fed, with no rock breakers. In front of each set of batteries are Rittinger amalgamators, of about 2 ft. diameter, which save from 6 to 8 per cent. of the valuable metals. No copper plates are used in the mills. The sand and slime are elevated by pumps to large spitzkasten which deliver the slimes to Bartsch revolving tables and the sands to Rittinger tables, both of which do good work. Light stamps are used, weighing only from 283 to 332 kg., to prevent sliming, the principal portion of the gold and silver being in the galena and blende. Heavy stamps have been tried by an English company unsuccessfully. In the Felsobanya mill are 75 stamps of 200 kg. each, and 72 wooden stamps weighing from 100 to 110 kg.; the mill treats from 30,000 to 33,000 tons per year. The cost of milling is about 80c., and of mining, including all charges, from $1.60 to $2.40 per ton.

The Silberau Mill.—The ore treated at this plant, near Ems, contains galena, blende, spathic iron and gangue.[2] The mill has three buildings, one for mine fines, one for blende middlings and one for poor lead ore. The mine fines building has two divisions for two different classes of ore. In each division the trommel series is 30, 18, 10, 5, 3, and 1½ mm. The products are hand picked or jigged on four-sieve jigs. Under 1½ mm. is classified and sent to four-sieve jigs and four Linkenbach fixed convex round tables. The blende middling building has two rolls, each 950x300 mm. The crushed material is sized on trommels with 6-, 4-, 3-, 2- and 1.5-mm. holes and treated on five-sieve jigs. Under 1½-mm. is classified and treated on jigs and four fixed convex Linkenbach round tables. Three Wetherill magnets clean the zinc concentrates by removing the spathic iron. The lead building has trommels with 6-, 3- and 1.5-mm. holes, four-sieve jigs, classifiers and three Linkenbach round tables.

Concentration of Broken Hill Tailings.—During the half year ending May 31, 1907, the Broken Hill Proprietary Company treated 116,240 tons of tailings by the Delprat process and obtained 25,353 tons of zinc concentrates which averaged 41.81 per cent. zinc. It has been found that for the Delprat process the coarse tailings must be well ground and in consequence the original installation of four grinding pans has been

[1] Edward Skewes. *Min. and Sci. Press*, XCVI (1908), p. 66.
[2] H. W. Linkenbach. *Glückauf*, XLIV (1908), p. 369.

increased to 15. This is expected to raise the percentage of zinc in the concentrates.[1]

The Zinc Corporation, Limited, erected a plant to use the Potter process followed by a magnetic concentration of the zinc concentrates. From Jan. 23 to March 2, 1907, this plant treated 3183 tons of tailings and yielded 493.3 tons of zinc concentrates with 40.1 per cent. zinc, 9.12 per cent. lead and 9.78 oz. silver per ton and 32.39 tons of lead concentrates with 49.9 per cent. lead and 21.4 oz. silver per ton. The capacity was 150 tons per day and the extraction of zinc 46.28 per cent. The results were unsatisfactory. Work with the Cattermole process, which uses acid and oil, between April 6 and May 4, 1907, yielded 1091 tons of zinc concentrates from 3970 tons of tailings. The zinc concentrates contained 44.9 per cent zinc, 9.6 per cent. lead and 10.1 oz. silver per ton. The zinc extraction was 73 per cent. These results were satisfactory, but owing to the delicacy of the process it was decided not to erect a large plant.

Experiments with an Elmore apparatus treating 30 to 40 tons daily gave an extraction of 89 per cent. of the zinc from the rich tailings of the Block 10 mine and 83 to 89 per cent. extraction from the tailings of the British mine. In the latter case the concentrates contained 44 to 45 per cent. zinc. The Elmore has the advantage over the Cattermole that it does not require so fine preliminary grinding and works at ordinary temperatures. The final result was that the installation of 16 sets of Elmore apparatus was begun, and plans have been made to double this if the large scale results are satisfactory.

Financial Side of the Elmore Process at Broken Hill.—The Broken Hill ore contains lead, zinc and silver.[2] Wet concentrates saves 60 to 70 per cent. of the lead and silver and puts practically all of the zinc into the tailings. These tailings are crushed in Huntington mills and grinding pans and treated by the Elmore vacuum process. The Elmore concentrate is divided into a zinc concentrate and a lead-silver concentrate. Zinc smelters pay for lead only when it is over 8 per cent. and then only half its value. They also pay for the silver above 5 oz. only and at the rate of 30c. per oz. They pay the average zinc price for the year after deducting smelting charges, freight, smelting losses, and profit. Lead smelters pay nothing for zinc but put a penalty on it if above 12 per cent.

The following conclusions are based on a value of $97.50 per ton for zinc and $60.50 per ton for lead. The Elmore concentrate averages 45 per cent. zinc, 12 per cent. lead and 16 oz. silver. The further treat-

[1] C. Göpner. *Metallurgie*, V (1908), p. 128.
[2] C. Göpner. *Metallurgie*, V (1908), p. 609.

ment of this material by heating and concentrating on Wilfley tables yields monthly 230 tons of lead concentrate and at the same time raises the zinc concentrate from 43 up to 47 per cent. zinc. These two products for a month sell for $17,500 more than would the Elmore concentrate without further treatment. This division of the Elmore concentrate is the key to the profit or loss. Based on the work of a certain week, which was better than the monthly average, there would be a surplus of $35,000 per month above the costs. From this is to be subtracted $15,000, which is the price or royalty paid for the tailings, leaving a net profit of $20,000 or $240,000 per year. Since the plant has been operating only a short time, Göpner makes the following estimates of increased profit in the future.

(1). A profit of $12,500 per month due to reduction of working costs in the plant, when it gets to running smoothly.

(2). A profit of $2500 to $5000 per month due to an increase of 1000 or 2000 tons capacity over the original 13,000 tons.

(3). A profit of $2500 per month due to increased extraction of the zinc from 88 up to 92 per cent.

(4) A profit of $12,500 to $15,000 per month due to increased extraction of the lead. At first 17 per cent. of the lead in original ore from the mine was saved by the Elmore process and re-treatment and they hope to double this figure.

The sum of all these is at least $30,000 per month without reckoning the possibility of higher prices for lead and zinc. The plant started in March, 1907.

Flotation Processes at Broken Hill.—The Minerals Separation Company's processes have their greatest development on the Central mine of the Sulphide Corporation.[1] There are four plants here with a total capacity of 2000 tons per day. One of these plants treats 500 tons per day of pure slimes, with an extraction of 87 per cent., producing concentrates assaying 36 per cent. zinc and 30 per cent. lead. Owing to the low price of lead the plant is now closed. The other three plants treat 500 tons per day each of tailings and slimes mixed as they come from the lead concentrators, and, with an extraction of 85 per cent., produce concentrates assaying 47 per cent. zinc and 10 per cent. lead. This process (erroneously called the Cattermole) was tried by the Zinc Corporation and thrown out as a failure.

Potter Process.—Since the consolidation of the Delprat and Potter interests, the Delprat process has become a thing of the past. The Broken Hill Proprietary is using the Potter process only. The plant handles 1200 tons daily and it is rumored that this requires two tons

[1] *Min. and Sci. Press*, XCVI (1908), p. 494; *Metallurgie*, V (1908), p. 297.

of oil. Enlargement of capacity is being considered. The tailings from the Wilfley tables, mixed with reground material from the dumps, are washed clear of slimes, and fed to Potter pans 9x9 ft. at the top, and 16 ft. deep, where they come in contact with strong acid solution. The extraction of zinc is said to be 90 per cent., in the form of concentrates assaying 40 per cent. zinc.

DeBavay Process.—This is in use on the Broken Hill North mine. The tailings from the lead mill are washed clear of slimes and sized to eliminate all material over 60-mesh size. This washed and screened material is delivered to an A. Z. Clark agitator, which takes 15 tons of material to the charge. After agitation with water, an acid wash is used which consists of a weak solution of sulphurous acid to which 5 or 10 lb. of strong sulphuric acid are added per ton of tailings. After complete agitation the tailings are transferred to a pressure-vat, where they are agitated under an air pressure of 60 lb. per sq.in., and an oil emulsion is added to aid subsequent flotation. From this vat the aerated and oiled pulp is forced by pressure to the head of the mill, where it is run into storage hoppers. From these hoppers it is fed in a thin stream upon inclined tables, where the oiled and aerated mineral floats off and the gangue sinks.

Elmore Vacuum Process.—The Pinnacles Elmore plant has been closed down by reason, according to the manager, of high costs and low extractions. The Zinc Corporation will install 16 machines handling, in all, 400 tons daily by the Elmore process. The tests made by this company with this process indicate results similar to those obtained by the Minerals Separation process on the Central mine.

Gillies Process.—This process on the Block 10 mine has been abandoned because of high costs and low extraction.

Concentrating Mesabi Iron Ore.—Formerly the great silicious ore bodies of the Canisteo and Holman mines on the Western Mesabi were thought to be worthless as they carried but 35 to 45 per cent. iron.[1] It was discovered, however, that the silica was not combined with the hematite but existed as taconite particles disseminated through the ore and a 250-ton experimental plant has been built to treat the ore. As the ore is soft, no crushing is required and it goes from cars to (1).

1. Bin. To (2).
2. Conical trommel with 2-in. round holes. Oversize to (3): undersize to (4).
3. Picking belt sloping 15 deg. and traveling 18 ft. per minute. Waste picked out; residue to (8).
4. Two log-washers 25 ft. long with steel centers formed by 4-in. and 6-in. angle irons with steel arms bolted to them Slope, 1 in.; speed, 9 r.p.m. Coarse ore to (5); fines to (6).
5. One concentric trommel with $\frac{5}{16}$-in. round holes and $\frac{3}{4}$ x $\frac{1}{2}$-in. slots. Water spray. Both oversizes to (8); undersize to (7).
6. Two settling tanks. Settlings to (7).
7. Two turbo (log) washers 18 ft. long, each containing two logs revolving in opposite directions. There is a false bottom of metal perforated with $\frac{1}{16}$-in. holes up through which water rises. Slope is 1 in. per foot. Concentrates to (8); tailings to waste.
8. Shipping bin.

[1] R. B. Brinsmade. *Min. Sci.*, LVIII (1908), p. 528. H. H. Stoek, *Mines and Minerals*, XXIX (1908), p. 97.

Ore containing 30 to 35 per cent. iron is concentrated to 60 per cent. iron. A plant is to be built having five units each treating 1000 tons in 10 hours.

Tin Dressing in Queensland.[1]—There are two types of ores in the Stannary Hills district. The first, free from sulphides, can be dressed up to 70 per cent. metallic tin by one continuous act of concentration. The second, a complex ore containing tin oxide with pyrite, arsenopyrite, blende, galena, etc., can be concentrated up to 30 or 40 per cent. tin oxide and then is roasted and finally concentrated up to 70 per cent. metallic tin.

At the Stannary Hills mill the ore is dumped from cars to (1).

1. Bin with 8-hours capacity. To (2).
2. Jaw breaker, 204 r.p.m., crushing 200 to 300 tons per 24 hours to 2½ in. By screw elevator to (3).
3. Cornish rolls crushing to ½ in. By distributing belt conveyor to (4).
4. Bins holding 120 tons. By shaking feeders to (5).
5. Thirty stamps weighing 1520 lb. each and making 104 drops of 6 in. per minute. Blanton cams and pulleys are used. Capacity 4 to 7 tons per stamp per 24 hours. Cast-iron guide bushings were tried but given up on account of rapid wear from grit. Screen originally 25-mesh but now 0.055-in. (12-mesh, 22 S.W.G. wire) which has increased mill capacity by 10 per cent. Sample of pulp taken automatically and amounts to ½ lb. per ton crushed. Pulp to (6).
6. Classifiers. Spigots to (7); overflow to (9).
7. Four Harts jigs with stream tin beds. The only concentrates are 3d and 4th hutches (30 to 40 per cent. tin oxide) to (15); tailings to (8).
8. Four Wilfley tables. Concentrates to (15); tailings to waste.
9. Spitzkasten. Spigots to (10); overflow to (12).
10. Five Wilfley tables. Concentrates to (15); middlings classified to (11); tailings to waste.
11. Two Wilfley tables. Concentrates to (15); tailings to waste.
12. Settling tanks. Settlings to (13); overflow to (29).
13. Three convex revolving slime tables. Concentrates to (15); middlings pumped to classifier and thence to (14); tailings to waste.
14. Convex revolving slime table. Products like (13).
15. Calciner pits of cement for settling. Material shoveled out and drained over flues of furnaces to (16); overflow to (29).
16. Merton furnace with capacity of 80 tons per six days. The concentrates to be treated amount to 50 to 80 tons in six days. A dead roast is necessary to change pyrite to ferric oxide. Product to (17).
17. Plunger sand pump. To (18).
18. Classifier. Spigot to (19); overflow to (24).
19. Three grinding pans. Cast-iron shoes last six to eight weeks and grind 140 tons. Krupp steel shoes last six months and grind 560 tons. To (20).
20. Three classifiers in series. Spigots to (21); overflow to (29).
21. Three pairs of Luhrig vanners. Concentrates 70 to 72 per cent. tin to (28); middlings 50 to 60 per cent. tin to (22); tailings 7 per cent. tin to (23).
22. Kieves 4 ft. diameter, 4 ft. deep. Concentrates to (28); tailings to (27).
23. Pump. To (24).
24. Settlers. Products to (25).
25. Ten Krupp vanners. Concentrates to (26); middlings to (23); tailings to waste.
26. Kieves 4 ft. diameter, 4 ft. deep. Concentrates to (28); tailings to (27).
27. Small Cornish buddle 12 ft. diameter. Concentrates to (26); tailings to (15).
28. Shaft drier. The dried concentrates are sacked for shipment.
29. All slime water passes to settling boxes and the slime settlings are shoveled out periodically and treated on Cornish buddles.

Wooden ties or boxes are placed in the tail race to catch the tin. The mill uses 9 to 10 tons of water in the stamps per ton of ore crushed and an equal amount on the concentrating machines.

Magnetic Concentration.

Ekman & Markman Magnetic Separator.—This machine is in use at Vintjern, Sweden, north of Falun, for treating magnetite ore containing 32 per cent. iron.[2] It yields concentrates with about 62 per cent.

[1] W. L. Cleland, Aust. Inst. Min. Eng., XII (1907), p. 154.
[2] G. Franke. *Glückauf*, XLIV (1908), p. 1457.

iron and tailings with only 2 per cent. iron, mostly as silicate. It consists of a short conical drum with horizontal axis and like a conical trommel except that the shell is made of alternate strips of soft iron and cement laid parallel to the axis and that the drum is supported on bearings beneath and revolved by a belt passing around it. Surrounding the drum over an arc of about 120 deg. are three fixed, horseshoe electromagnets whose poles are elongated, the long dimension corresponding in direction with the strips in the drum shell. The six poles are so arranged that there is a change of polarity between adjacent poles. Passing through the drum from end to end is a horizontal conveying belt studded with soft iron pegs. Ore is fed into the drum at the small end. The non-magnetic part passes through and is discharged at the large end while the magnetic particles are held against the inside of the shell by the magnets and carried upward by the revolution of the drum. Elimination of non-magnetic particles mechanically held occurs at each reversal of polarity. When the magnetic particles have reached the highest point of revolution they pass out of the magnetic field and are washed off by water jets upon the conveying belt. A second set of water jets plays upon the belt to wash the last of the non-magnetic particles over the sides of the belt. The magnetic particles are held by the magnetized pegs in the belt and pass over the end pulley of the belt where they are washed off by a third set of jets.

New Magnetic Separator.—The Hernadthaler Ungarische Eisenindustrie, A. G., of Budapest, has constructed an electromagnetic ore concentrator, capable of treating, both strongly and feebly magnetic minerals by the wet or dry process.[1] The apparatus was tried at the Krompach works, on a spathic iron ore containing iron 27.37 per cent., sulphur 1.511 per cent., and copper 0.911 per cent., with satisfactory results, the operation being performed in the following manner: The ore is fed through a hopper to the multipolar separation magnet, which is composed of an iron shaft, rotating at high speed, and provided with annular grooves and iron rings. The grooves contain the copper coils, and the rotary fields of magnetic force are set up in the annular spaces between the iron rings. The rotation of the shaft sets the ore also in rotation, the perfectly non-magnetic portions being expelled outward by the centrifugal force, the faintly magnetic portions being restrained by the influence of the poles and falling down between the dividing walls provided for that purpose, whilst the strongly magnetic particles are attracted to and carried round by the magnet poles until removed by a rotary detaching device and collected separately. The iron points of the detacher are mounted on two insulated metal plates, and act as magnetic

[1] *Min. Journ.*, LXXXIV (1908), p. 548.

inductors, whose magnetic potential is greater than that of the magnetic field, so that they remove the magnetic portions of the ore, releasing the latter, however, as soon as the points have rotated out of the magnetic field. In the wet process the mixture of ore and water is fed in a similar manner to the magnet shaft, the lower portion of which dips in a spitzkasten containing water, into which the non-magnetic parts of the ore subside, leaving the magnetic portions adhering to the magnets. With the above ore the output of the machine by the wet process amounted to 9.4 tons of material (size ¼ mm. and under) per hour, with four magnetic fields, and ¾ ton per hour (size 2½ to 1¼ mm.) by the dry process.

Trials were also made with wolfram-tin ore containing 19.62 per cent. of wolfram and 35.31 per cent. of tin, of which about 96 per cent. of the total of each metal was recovered in a separated condition. Very good results have also been obtained in separating zinc blende and spathic iron from heavy spar, crude and roasted spathic iron, lightly roasted pyrites and zinc blende.

Gröndal Magnetic Separators.—The Gröndal process is extensively used in Sweden and has been installed in plants in Norway, England, Spain and the United States. The following description of the Flogberget mill in Sweden is a good example.[1] Here the ore, containing 27 to 29 per cent. iron in the form of magnetite in a hard greenstone gangue, is brought to (1).

1. Jaw breaker, crushing to 8 cm. To (2).
2. Bin. By roller feeder and belt conveyor to (3).
3. Two Gröndal wet ball-mills 2 m. diameter and 2 m. long with hard steel balls up to 145-mm. diameter. Total charge of balls is 1 ton per mill. Each mill makes 28 r.p.m. and requires 45 h.p. The product is 0.3-mm. size. The mills crush 2.7 tons per hour and the wear of balls is about 1 kg. per ton of ore. To (4).
4. Two spitslutten with magnets to coagulate fine magnetic particles and cause them to sink to the spigot. Spigot to (5); overflow to waste.
5. Two double Gröndal No. 5 magnetic separators. These are of revolving drum type 600-mm. diameter with fixed electromagnets inside. Pulp is fed against the face of the drum and the magnetic particles carried up and around by the revolution of the drum to be washed off on the back side. The drum dips into a water tank so that fine magnetic particles may be attracted and lifted out of the water by the revolution of the drum. Magnetic concentrates to (9); middlings to (6); tailings overflow to waste.
6. Tube mill 4 m. long, 1.2 m. diameter with silex lining. It makes 25 r.p.m. and requires 40 h.p., grinding to 0.1 mm. By elevator to (7).
7. Two spitslutten like (4). Spigot to (8); overflow to waste.
8. Two double Gröndal No. 5 magnets like (5). Products like (5).
9. Four unwatering boxes of trapezoidal section. Magnetic concentrates with 8 per cent. moisture for briquetting plant; overflow water to be used again in the mill. These boxes are bumped 36 times per minute by a cam, for the purpose of settling the solid material and a box receives its full charge of one ton in about two hours.

The concentrates contain 67.9 per cent. iron and the tailings 6 per cent. 100 tons of ore yield 37 tons of concentrates. The power required is 40 h.p. for rock breaker, 1 for conveyer, 90 for two ball-mills, 40 for tube-mill, 2 for elevator, 10 for pump and 6 for unwatering boxes. In each 12-hour shift two men are required at the rock breaker and two for the rest of the mill. The plant cost $20,000 and the cost of concentration, not including amortization, is $0.41 per ton of raw ore.

[1] G. Franke, *Glückauf*, XLIV (1908), p. 1417.

Magnetic Separation in the District of Siegen.[1]—A new separator employed in this locality resembles the Wetherill except that instead of having cross belts for removing the magnetic particles it has a revolving drum surrounding the upper core. The axis of this drum coincides with the center of the core.

Five machines of another type (the drum type) are running in the Siegen district on roasted spathic iron ore. The drum diameter is 600 mm. and its length is 600 mm. The material is sized into stuff from 15 to 6 mm. and from 6 mm. to 0. This drum machine has a feed drum for bringing the ore into the magnetic field and a second drum containing iron pieces which become secondary magnets in the magnetic field and lift the magnetic particles. The magnet has 5000 turns and uses 5 amperes at 70 volts.

The mill process on roasted spathic iron at the Bruderbund mine at Eisern is as follows:

1. Elevator. To (2).
2. Grizzly with 120 mm. spaces. Oversize picked for iron and waste; undersize to (3).
3. Concentric conical trommel with 6-, 15- and 30-mm. holes. Over 30-mm. to (4); through 30- on 15-mm. to (6); through 15- on 6-mm. to (7); through 6-mm. to (9).
4. Picking table. Iron concentrates; waste; residue to (5).
5. Jaw breaker. To (1).
6. Rolls. To (1).
7. Magnetic separator. Iron concentrates; middlings to (8); tailings to waste.
8. Rolls. To (10).
9. Magnetic separator. Iron concentrates; middlings to (10); tailings to waste.
10. Elevator. To (11).
11. Trommel. Oversize to (8); undersize to (12).
12. Magnetic separator. Iron concentrates; middlings to (8); tailings to waste.

The ore contains about 30 per cent. iron and manganese; the concentrates contain 56 per cent. and the tailings 16 per cent.

Still another form of separator, called the tandem separator, has a feed conveying belt like the Wetherill only slightly inclined. The first separating magnet is placed about one-third of the distance from the feed to the discharge end of the belt. Here the magnetic particles are lifted off by a drum revolving around the upper cores and dropped on a cross belt. The second magnet is placed at the discharge end of the feed belt.

Coal Washing.

Coal Washing in Austria.[2]—Coal plants are usually made in units, each having a capacity of from 40 to 100 tons per hour. The common sizes of coal are: Over 120 mm., 120 to 70 mm., 70 to 40 mm., 40 to 25 mm., 25 to 15 mm., 15 to 8 mm., 8 or 10 mm. to 0. The upper limit of washing is 70 or sometimes 80 mm. For the first screening at 70 or 80 mm. the old bumping screens, fixed bar screens and Briart

[1] Horel. *Oest. Zeit. f. B. u. H.*, LVI (1908), p. 317. J. W. Bartsch. *Revue des Mines*, Series IV, Vol. XXIII (1908), p. 93.
[2] E. Ruland-Klein. *Oest. Zeit. f. B. u. H.*, LVI (1908), p. 365.

screens have been replaced by the roller screen, the moving bar screen, and the screen shaken by eccentrics. The moving bar screen has a series of punched channel irons lying beside one another and driven by four eccentrics so that while one half are moving in one direction the other half are moving in the opposite direction. The roller screens have a series of rollers which advance the oversize coal gently forward and at the same time allow the undersize to fall through the spaces between them. On account of its simplicity the screen shaken by eccentrics is commonly used in Westphalia for coal of over 70-mm. size. The head end of the sieve is driven by two eccentrics without connecting rods while the lower end is suspended by rods. The motion of the sieve at the head end is a circle lying in a vertical plane while at the lower end it is rectilinear. Two or more screens may be mounted one above the other in the same frame; sometimes a third eccentric is used at the lower end to impart a sidewise motion.

The oversize of screens is received on picking belts which generally have smooth surfaces 500 to 1400 mm. wide and moving at a speed of about 250 mm. per second. The old-style box conveyor is less used than formerly. It has the disadvantage that stone and slate are not so easily picked out from a box as from a smooth surface. The smooth conveyors may be made of smooth iron plates, of wire, or of rubber belt. For purposes of draining they may be perforated.

The undersize of the screens (below 60 to 80 mm.) is delivered to a hopper and thence fed to a bucket elevator usually inclined from the vertical. These vary in width of belt from 200 to 1800 mm. For draining purposes a wide elevator with slow speed (about 150 mm. per second) is used. The ordinary speed is 270 mm. per second. The normal capacity of an elevator 1000 mm. wide is about 70 tons per hour. For screening material under 70 or 80 mm., trommels or shaking screens are used, more commonly the latter, of which there are many varieties. Sieves having a rotary shaking motion are not favored. For high capacity and good screening the double side shaking suspended screen is to be recommended. It needs no support beneath and it works in balance without shaking the building. Where there is little room above for suspension a horizontal sieve supported from below is used. This is given an elliptical motion in a vertical plane. The Hempel screen is a flat screen whose frame is mounted directly on four eccentrics which give it a circular motion in a vertical plane. Several sizes of screen may be mounted one above another in the frame. Such a screen 1250 mm. wide and 3000 mm. long with 45-, 25-, 16- and 4-mm. holes has a capacity of 50 tons.

The coal sizes between 70, 40, 25, 15 and 8 mm. are treated on coarse jigs built with sieves 800, 1000, 1250, 1400 and 1600 mm. wide and having a capacity of about 5 tons per meter of width. The water consumption is about 600 liters per minute. The jigs are built of iron or of wood. For washing fine coal below 8 or 10 mm. jigs are used with a bed of feldspar, sometimes also quartz or spathic iron.

The unwatering of coarse coal, over 15 mm., is done on fixed screens; for coal between 15 and 8 mm. an eccentric screen is used; below 8 mm., a vibratory or bumping screen is coming into use. This last is an inclined screen to which are given about 250 reciprocating blows per minute perpendicular to the plane of the screen. Washed coal is dropped into cars through spiral conveyors to prevent breakage. Before loading on railroad cars the different sizes of coal may be run over a shaking screen to remove fines.

For watering large quantities of fine coal, drainage tanks may be used. These are arranged to be used in rotation. The coal and water are run into one tank until it is full of coal, the water overflowing to settling basins. Then the stream of coal and water is shifted to the next tank. The coal in the first tank is drained through perforated pipes until after 24 to 30 hours it contains 8 to 10 per cent. moisture. Shallow drainage tanks are more efficient and are now supplanting the deep tanks. There are several forms of settling tanks for the overflow water. In one case the tank is 20 to 25 m. long, divided in three compartments each 1600 to 2000 mm. wide. The largest particles of fine coal sludge settle in the first compartment and are removed by screw conveyor and elevator to large drainage tanks. Material settling in the second compartment is removed by screw conveyor and lifted by chain pump to three or four drainage tanks where moisture is reduced to 12 or 15 per cent. so that the coal may be burned under the boilers. The clear water overflowing from the third compartment is used again in the mill.

Roche-la-Molière Washery.—This washery, treating 80 tons per hour by the Coppée system, has two sections for two separate grades of coal.[1] The coal ranging from 50 mm. to 0, comes to (1).

1. Five bins holding 650 tons total. By roller feeders and two elevators to (2).
2. Two sets of shaking screens with 4-, 8-, 16- and 30-mm. sieves. These make five sizes to (3).
3. Jigs for each size. Feldspar beds are used for coal under 8-mm. size. Washed coal from jigs over 8 mm. to (4); washed coal under 8 mm. to (6); middlings from jigs over 8 mm. to (7); middlings under 8 mm. to (8). Slate to (9).
4. Screens to remove fine coal. Oversize to (5); undersize probably to (6).
5. Two bins each holding 40 tons. The coal before shipment passes over rinsing screens.
6. Two series of draining bins with total capacity of 1200 tons.
7. Draining bin. Drained material stored for future treatment.
8. Draining bin. Stored for future treatment.
9. Draining bin. To waste.

All slimy water goes to a series of spitzkasten whose spigot products

[1] *Compt. Rend.* de la Soc. Ind. Min. (1906), p. 175.

are washed over ½-mm. screens. The oversize of ½ mm. are mixed with the washed fine coal. The water is settled in large tanks.

Washery at Dawson, New Mexico.—In the new washery, which has a capacity of 2400 tons of coal in 10 hours[1], the scheme is as follows:

1. Primary screens and crushers. By belt conveyor to (2).
2. Two cylindrical steel bins 40 ft. diameter, 40 ft. high, each holding 1000 tons. By eight rocker gate feeders and two parallel belt conveyors to (3).
3. Two 6x12-ft. shaking screens with 1½-in. round holes. Oversize to (4); undersize to (5).
4. Toothed rolls crushing to 1½-in. To (5).
5. Weighing machine. By belt conveyor and distributor to (6).
6. Eight Stewart jigs with tanks of ½-in. steel plate lined with 4 in. of concrete. Coal to (10); refuse through hutch to (7).
7. Four No. 5 Luhrig elevators. To (8).
8. Two refuse trommels, 4x8 ft., with ½-in. holes. Oversize to refuse bin and thence to cars to dump; undersize to (9).
9. Four Luhrig fine-coal jigs of same general construction as the Stewart jigs. Coal to (10); refuse by two No. 5 Luhrig elevators to refuse bin mentioned in (8).
10. Two 4x12-ft. dewatering trommels. Oversize to (11); undersize to (12).
11. Two 60-in. disintegrators. To (13).
12. Settling tanks. Settlings recovered by two perforated bucket elevators to (13).
13. Belt conveyor and distributing conveyor to four circular steel bins 20 ft. diameter, 40 ft. high. By electric larries to coke ovens.

Anthracite from Culm Banks.—It is estimated that of the 216,000,000 tons in the Pennsylvania culm banks, nearly 26,000,000 tons have been recovered and sent to market.[2] A washery capable of treating 500 to 1000 tons per day can be erected for $25,000 or $35,000. The treatment is similar to that which raw coal receives in a washery except that there is not much of the large sizes (egg, stove and nut) to be cared for. The chief machinery used is the screen, the Emery picker, the roll, the crusher and the jig. Approximately 500 gallons of water are required per ton of washed coal. Washery products are pea coal (⅛ to ⅝ in.), buckwheat coal (⅝ to ⅜ in.), rice coal (⅜ to 3/16 in.), and barley (3/16 to 3/32 in.). The average percentage of the foregoing sizes is 23, 38, 24 and 15 per cent. respectively. The cost of washing figures out to be about 15c. per ton of washed coal.

For taking coal from the culm banks steam shovels are used in some cases. A shovel with a bucket holding 1 or 1.5 cu.yds. can deliver 80 tons per hour. In other cases the bank is flushed down by a fire hose using water under a pressure of from 10 to 20 lb. per sq.in., and guided to conveyors of the scraping type which deliver to the washery. These conveyors may be 500 or 600 ft. long and consist of a main permanent conveyor and branches which may be swung forward to keep up with the retreat of the bank. The waste from the washery, and stuff below 3/32 in. is flushed into the old mine workings and used for filling.

Bituminous Washery at Tyler, Pennsylvania.—One of the most northerly coke operations in the Pennsylvania bituminous field is located at Tyler, Clearfield County.[3] It is operated by the Powhatan Coal and Coke Company, of Buffalo. The seam is the Middle Kittanning and has

[1] *Mines and Minerals*, XXIX (1908), p. 91. *Eng. and Min. Journ.*, LXXXVI (1908), p. 182.
[2] Richard Lee. *Eng. and Min. Journ.*, LXXXV (1908), p. 720.
[3] Edward K. Judd. *Eng. and Min. Journ.*, LXXXV (1908), p. 457.

an average thickness of 3½ ft., counting 6 in. of bony and sulphurous coal at the top and 3 in. of bony coal at the bottom. Only 33 in. of the clear coal is broken down. The coal from the mine is dumped into (1).

1. Receiving bin. Delivers coal to (2).
2. Short conveyor. Delivers coal to (3).
3. Inclined screen with ½-in. round holes. Delivers undersize to (6) and oversize directly to (4).
4. Bradford breaker. This machine consists of a cylinder 11 ft. long and 9 ft. diameter made of steel plate punched with 1½-in. square holes. It is mounted by spiders on an axle and is turned by a spur gear at one end. The feed end is partially closed with an annular rim, but the discharge end is entirely open. To the inside are bolted cast-iron paddles 12 in. long, about 1½ in. thick and projecting inwardly about 6 in. They are set obliquely so as to urge the coal to the discharge end. Attached to the outside of the cylinder and concentric with it is a truncated cone of sheet steel, punched with ½x1½-in. slots. Lumps of hard and bony coal resist crushing in the Bradford breaker and are discharged from its open end. This material is used for fuel under the boilers. The undersize of both the 1½- and ½-in. screens goes to (6) and the undersize of the 1½-in. screen goes to (5).
5. Geared corrugated rolls set to ½ in. These rolls are 48 in. wide and 30 in. diameter. They are driven by a 40-h.p. motor and deliver crushed coal to (6).
6. From (3), (4) and (5). Elevator delivering ore by way of distributing conveyor belt to (7).
7. From (6). Two bins each having a capacity of 150 tons. To (8).
8. From (7). 36 Campbell tables having a total capacity of 150 tons per hour; coal by way of launders to (9) and slate to cars.
9. From (8). Rectangular draining pits delivering by clam shell buckets and conveyors; coal to (10).
10. From (9). Two bins delivering coal by way of larries to coking ovens.

The coal coming to the tables showed sulphur, 2.162, and ash, 9.04 per cent. The washed coal contained sulphur, 1.17, and ash, 5.17 per cent. The washery waste as a whole carried 12.69 per cent. sulphur.

Washery at Sydney Mines, Cape Breton.—The Nova Scotia Steel and Coal Company has a plant designed by Stein & Boericke which has an output of 500 tons per 10 hours.[1] All the screenings from the different tipples are brought to (1).

1. No. 1 bucket elevator. By Robins belt conveyor, 135 ft., to (2).
2. Modified Jeffrey rolls, 24-in. diameter and making 180 and 270 r.p.m. respectively in opposite directions. These originally had the movable roll held in position by a counterweight acting through toggles. This did not yield sufficiently when a large piece of iron came in the feed and so bolts have been added which shear off when the toggles do not yield. By No. 2 elevator to (3).
3. Shaking screen with ½-in. holes. Oversize to (4); undersize to (5).
4. Plain rolls 16 in. diameter, 80 r.p.m. To (5).
5. No. 3 elevator to (6).
6. Steel storage tank holding 1000 tons. By scraping conveyor and No. 4 elevator to (7).
7. Shaking screen inclined 15 deg. and run wet with ½-in. holes. Oversize to (8); undersize to (9).
8. Four coarse jigs with bed of ¾-in. feldspar 2 to 4 in. thick, 72 pulsations per min. Only three jigs used at a time, each treating six tons per hour. Coal to (10); slate to (12).
9. Four fine jigs with bed of ¾-in. feldspar, 135 pulsations per min.; ¼-in. sieve in one compartment and ½-in. on the other. Capacity 7½ tons per jig per hour. Coal to (10); slate to (12).
10. No. 6 elevator with perforated buckets. Coal to (11); drainings to (13).
11. Drainage and storage tower. Coal to one set of coke ovens or by chute and No. 5 elevator with perforated buckets to another storage tower for another set of ovens; drainings to (13).
12. No. 7 elevator with perforated buckets. Slate to bin in storage tower or flushed away; drainings to (13).
13. Sludge tank. Settled sludge drawn off periodically. Water repumped to be used over.

The water in circulation is about 2000 gallons per minute supplied by 12-in. centrifugal pump. The loss of water in the drained coal and slate and in flushing is about 25 per cent. by weight of the coal washed. Power is supplied by electricity from three induction motors of 75, 50, and 30 horse-power.

The slate contains about 25 per cent. good coal and is lifted by bucket elevator and rewashed on the fourth coarse jig, yielding an impure coal for boilers, and slate which is flushed away to the sea through a 6-in. cast-iron pipe 700 ft. long sloping 1 to 4 per cent. The salt cooling water

[1] C. L. Cantley. Can. Soc. Civ. Eng. (1908). *Can. Min. Journ.*, XXIX (1908), p. 525.

from the engine condenser is used for flushing. The slate contains about 13 per cent. impure coal.

AVERAGE ANALYSES OF WASHERY PRODUCTS.

	Moisture.	Volatile Matter.	Fixed Carbon.	Ash.	Sulphur.
Raw coal	5.65	27.56	56.43	16.19	2.18
Washed coal	13.42	29.54	59.53	4.53	1.50
Original slate	.....	21.80	37.10	41.10	4.06
Coal from slate	.....	30.59	51.61	17.18	2.79
Final slate	.....	19.10	24.40	50.50	9.07

Washing Tests on Coal.—The tests at the government testing plant at St. Louis were made with the following equipment: 18x24-in. Cornish toothed rolls crushing to 2 in., elevator, bumping screen, Stewart jig making 35 strokes per min. of 6 in. each, two sludge tanks and sludge elevators. A special center-plunger testing jig was installed in December, 1906. An apparatus for testing with heavy solutions was also used. Results of tests on 98 samples of coal are given.[1]

Coal Testing Plant.[2]—The government coal testing plant at Denver, Colorado, has the following equipment: One standard Stewart jig with a capacity of 20 to 30 tons per hour. One special jig. One sludge tank for washed coal with 12x16-in. perforated-bucket Luhrig elevator. Two 8x5-in. perforated bucket elevators for lifting refuse from each jig to a flight conveyor. One 16x12-in. gravity discharge bucket elevator and one 18-in. apron conveyor for raw coal. One 18x24-in. toothed roll crusher. One 24x30-in. corrugated roll disintegrator. One No. S-2 swing-hammer crusher. Delamater float and sink testing machine.

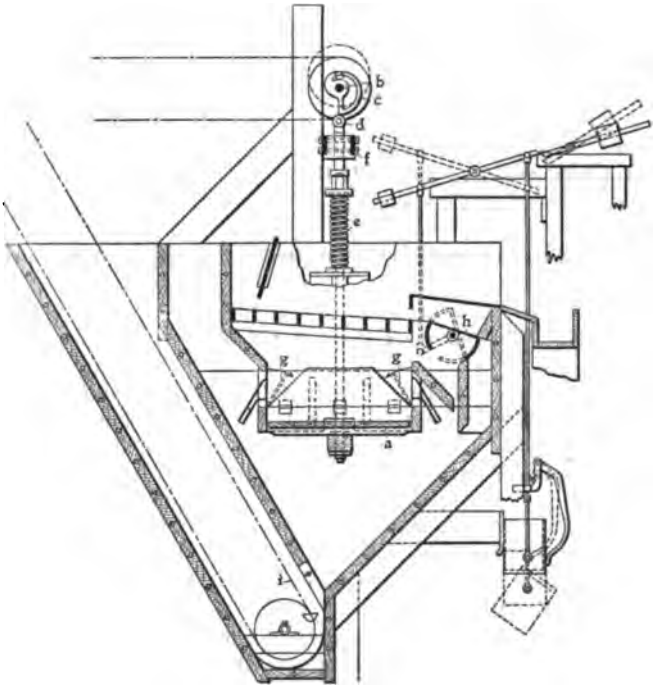
Raw coal is weighed, crushed by toothed rolls to 2½ in., conveyed and elevated to six storage bins, samples being taken on the way for analysis and for float and sink tests. From the bins it is drawn to the corrugated roll disintegrator and thence re-elevated to storage bin to be fed to the jigs.

The special jig is shown in the figure. It has a center plunger directly under the screen, thus ensuring uniform action all over the screen. Suction is avoided by flap valves in the walls above the piston. The screen is 4x5 ft. and the capacity is 15 to 25 tons per hour. The bed may be varied in thickness from 4 to 10 in. The driving mechanism is cam and spring and the number and length of the strokes may be quickly changed.

The Delamater float and sink apparatus has a rectangular tank with the bottom divided into two hopper shaped compartments. Each hopper

[1] G. R. Delamater. U. S. Geological Survey, *Bull.* No. 336 (1908), p. 8.
[2] G. R. Delamater. *Mines and Minerals*, XXVIII (1908), p. 401.

connects to a filter funnel to which suction may be applied. Inside the rectangular tank is a rectangular frame of such size that it will cover either one of the hoppers. In making a test the tank is filled with the heavy liquid and the frame is placed over one hopper which we will call *a* and the coal to be tested fed inside the frame. After stirring and allowing to stand a certain time the heavy stuff settles down to the filter



SPECIAL JIG FOR COAL TESTING.

beneath hopper *a*. Then the frame is carefully slid over to a position above the other hopper which we will call *b*, the heavy liquid is drained down through the hoppers, using suction if necessary, thus leaving heavy (sink) stuff on filter *a* and light (float) stuff on filter *b*. This material may then be washed with clear water and finally removed by lifting out the filter. This apparatus allows heavy solution tests to be made with rapidity.

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SAMPLING AND ASSAYING.

By F. F. Colcord.

Discussion of Sampling Methods.

Automatic Sampler.—C. S. Heathcote (*Journ.* Chamber of Mines of West. Australia, Mar. 31, 1908) describes an automatic sampler consisting of a bucket mounted on a revolving arm, which cuts through the stream of falling ore. The ore is screened, the oversize crushed and both portions united at the sampler spout. The sampler revolves at 325 r.p.m. and the bucket has a width equal to 1/112 of the circumference and is automatically tripped after cutting out the sample.

Sampling Ores from Cobalt.—W. K. McNeil (*Can. Min. Journ.*, Jan. 1, 1908) gives a few useful hints on the methods and apparatus employed in assaying ores from the Cobalt district. The ore in some cases is extremely rich and for this reason a plate glass quartering plate and a bucking board should be used on the final sample because of the greater facilities offered for thorough cleaning. In assaying, the scorification method is usually preferable to the fusion method.

Great Cobar Sampler.—An article (*Eng. and Min. Journ.*, LXXXV, pp. 950-952) on the Great Cobar smelting works in Australia gives a brief description of the sampling mill. The mill has a capacity of 50 tons per hour and is equipped with Simplex automatic samplers. The ore flow is to a No. 5 McCully gyratory crusher, then to a 64-in. sampler, and the cut to a No. 3 gyratory crusher. After this crusher is a 42-in. sampler, following which is a set of 24x14-in. belt-driven rolls, feeding to a 28-in. sampler. The sample is then passed through another set of rolls and sampler and the final sample, 1/1500 part of the whole, is discharged on the ground floor of the mill and there quartered down by hand.

Sampling at Great Cobar.—J. A. Church (*Eng. and Min. Journ.*, LXXXVI, pp. 113-115) analyzes the operations of the Great Cobar sampling mill just described. The mill is of the usual type, designed for large capacity and fairly constant grade of ore. The writer assumes that the ore is of not over 6-in. size and that probably one-half is small enough to pass through the initial crusher without further reduction in size. The fines consequently are partially separated from the coarse in passing through the crusher, while on the discharge spout the coarse

bounds ahead of the fines. The crushed ore is elevated to a bin and discharged down a spout, with apparently a 58-deg. slope, to the sampler. Here again the coarse precedes the fines down the spout as would be the case if the fall were in free air. Considering that this separation takes place at every step in the operation, what effect does it have on the accuracy of the work?

Theory of Sampling.—Very few articles have shown the results of sampling and resampling, but of the few is one by Paul Johnson (*Eng. and Min. Journ.*, April 12, 1902). Referring to this article, Mr. Church attributes the differences between the samplings to the delayed sampling of the rich fines apart from the coarse ore that belonged to them. The accuracy of sampling depends upon the incessant repetition of individual acts and the assumption that these acts are counterbalancing. The writer believes that the accuracy can be increased if the retardation of the ore is controlled so that the separation of particles belonging together does not take place and that equal quantities of the ore be delivered to the sampler in equal spaces of time. This result may be obtained by mechanical feed to the various pieces of apparatus instead of by gravity and is a change that need not increase the cost of the operation. The author concludes: (1) That particles of ore thrown together into the first (or any other except the last) crusher, become separated (without the millman's cognizance) and subsequently are sampled apart from each other. (2) The results of this separation are not known but are probably adverse to good sampling. (3) The introduction of controlled feed would correct the haphazard effects of gravitation, which cannot produce regular action from the heterogeneous mixture of sizes and densities of ore as it is mined and crushed. The solution of the problem lies in the introduction of shaking feed spouts.

Mixing Before Sampling.—L. S. Austin (*Eng. and Min. Journ.*, LXXXVI, pp. 238) agrees in the main with the ideas expressed in the foregoing article. He suggests revolving drums having within them staggered blades, in place of shaking spouts, since the drum acts as a mixer as well as a feeder. The conclusions drawn from Johnson's tables are not necessarily correct since many other causes may have been contributory. In the finishing methods the personal equation has to be considered as well as the conscientious cleaning of all parts of the mill. The point the writer wishes to bring out is that at the Great Cobar works only every fifth car is sampled and this raises the question whether the fifth car is representative of the other four.

Requirements of Mechanical Sampling.—Philip Argall (*Eng. and Min. Journ.*, LXXXVI, p. 291) in discussing Mr. Church's article agrees that the points raised regarding the use of long spouts, high veloc-

ity ore currents and excessive distance of fall are well taken but believes that the alleged ill-effect of retardation of the ore even in the extreme cases quoted is not detrimental to accurate sampling. A thorough mixing of the ore and a continuous feed to the samplers are, however, essential to good sampling. This desideratum is best obtained with the conical mixer of the late H. A. Vezin. The chief defects of the Great Cobar mill are the fifth car selection for sampling and the intermittent feed to the samplers, also the "tower" form of sampler like the mill in question, is not so suitable as one of less height with two elevators instead of one, which form would offer the same facilities for cleaning and would cost less to build and maintain. The conditions necessary for accurate sampling by machines as outlined by Mr. Argall are: (1) The first cut must be of sufficient size as to form an accurate sample of the maximum cubes. (2) The sample should be cut from a vertical, free-falling stream with a frequency of from 25 to 50 cuts per min. depending upon the character of the ore. (3) The sample after each cut should be crushed finer, thoroughly mixed and fed in a continuous stream to the next sampler, and so on.

Modern Mechanical Sampling.—E. P. Mathewson (*Eng. and Min. Journ.*, LXXXVI, pp. 338) comments on Mr. Church's article and contends that sampling in modern mills is as accurate as the best assaying. In modern mills short spouts lead to the cutting devices and shaking feed is used for the crushers and rolls, also the cutting devices are time

A COMPARISON OF SAMPLING.

Lot Numbers.	Per cent. copper.			Oz. silver.			Oz. gold.		
A B & C	A	B	C	A	B	C	A	B	C
2263 547	5.10	4.91	4.80	1.90	1.80	1.90	0.020	0.026	0.015
2464 548	3.42	3.52	3.70	1.50	1.70	1.80	0.018	0.026	0.010
2465 549	3.75	3.69	3.71	1.30	1.30	1.40	0.015	0.015	0.007
2466 550	6.13	6.03	5.86	2.10	2.00	2.20	0.015	0.023	0.015
2490 554	4.42	4.22	4.07	1.40	1.40	1.50	0.012	0.020	0.007
2491 552	4.02	4.57	4.48	1.30	1.40	1.60	0.012	0.015	0.007
6231 5730	4.08	4.03	4.01	1.70	1.60	1.40	0.02	0.02	0.025
118	3.54	3.38	.....	1.30	1.30	.....	0.007	0.007	.....
2118 38	7.03	6.95	.....	6.9	7.8	.....	0.015	0.012	.....
2119 37	7.44	7.01	.....	6.6	6.9	.....	0.015	0.015	.....
2121 40	9.15	9.32	.....	10.7	10.10	.....	0.022	0.03	.....
2122 45	15.11	14.48	.....	53.20	53.20	.....	0.010	0.010	.....
2124 46	6.91	5.98	.....	18.6	17.70	.....	0.010	0.012	.....
Total of assays	80.10	78.30	30.63	108.50	108.20	11.80	0.191	0.230	0.086
Mean	6.16	6.03	4.38	8.35	8.32	0.169	0.015	0.018	0.012
Difference	.....	0.13	.....	.....	0.03	.....	0.003	.....	.....
Per cent. difference	2.11	.....	.....	0.36	.....	.....	.....	16.67	.....

COMPARISON OF ASSAYERS.

Total of assays	.....	31.27	30.63	.....	11.20	11.80	.....	0.144	0.086
Mean	.....	4.47	4.38	.....	1.60	1.69	.....	0.0206	0.0123
Difference	.....	0.09	.....	.....	0.09	.....	.....	.....	0.0083
Per cent. difference	.....	2.01	.....	.....	.....	5.33	.....	40.29	0.0083

cutting machines, i.e., they are so proportioned and speeded as to take the whole of the stream of ore, for a given period of time, every so often. The contained table shows a comparison of the sampling of a number of 50-ton lots, sampled in two mills of similar design, A and B. A comparison of assayer's work is shown in B and C.

Sampling at Balaklala.—J. A. Church (*Eng. and Min. Journ.*, LXXXVI, pp. 431-433), in reply to the discussion of his previous paper, states that he is of the opinion that controlled feed to each separate piece of apparatus may not be necessary, but that a crusher and its sampler may be considered as a unit. He points to the new Balaklala home ore mill which has shaking feed to the first sampler and each subsequent pair of rolls, also to the Balaklala custom mill which has shaking feed to each of the first two samplers and each of the last two pairs of rolls. The frequency of cutting suggested by Mr. Argall seems too great. Regarding Mr. Mathewson's table, three out of the 13 lots showed a difference in the copper assay of more than 10 per cent., which seems to be entirely too large to be considered as satisfactory sampling. A further consideration of the Great Cobar mill shows that the rolls after the second and third cuts are so large that they must of necessity be running idle most of the time as well as the sampler. Prof. Austin's contention that the great number of cuts ensures by repetition the accuracy of the work is not borne out by the tables previously mentioned and must in any case depend upon whether the retardation is synchronous with the revolution or half revolution of the sampler.

Feeding to Sampling Machine.—T. R. Woodbridge (*Eng. and Min. Journ.*, LXXXVI, pp. 917-918) adds to the foregoing discussion by submitting data tending to prove that the effect of retardation in sampling is overdrawn. The use of feeding devices is evidently in extensive use but not for the purpose of controlling retardation of the ore but to obtain a more even feed to the crushers. The author has even considered the advisability of perforating the feed trays in order to pass the fines directly to the following sampler and thus increase the capacity of the rolls as well as to prevent the formation of the troublesome "pancakes." The essence of accurate sampling lies in the relation between the largest particle and the weight of the sample. (D. W. Brunton, *Trans.* A. I. M. E., XXV.) It is also well, in investigating sampling differences, to observe the working down of the sample after leaving the mechanical samplers. Referring again to the lack of effect of retardation, nine small lots of ore which had been sampled during a period of three months, were combined into one lot and sampled. The individual lots had been handled by different men and at different times but the proper relation between the size of the largest particle and the size

of the sample was in all cases maintained and proper care was used throughout. The table herein shows the results and demonstrates the accuracy of mechanical samplers.

INDIVIDUAL LOTS AS ORIGINALLY SAMPLED.

Gross weight, lb.	Dry weight, lb.	Assay and Analysis.							
		Gold oz.	Silver, oz.	Lead, per cent.	Insol., per cent.	Iron, per cent.	Sulphur per cent.	Zinc, per cent.	Copper, per cent.
3,095	3,079	0.00	3.8	53.0	28.0	1.0	9.0	0.0	0.0
1,387	1,345	0.89	88.7	26.0	32.0	11.0	0.0	0.0	0.0
1,410	1,400	1.44	312.0	4.0	22.0	34.0	21.0	2.0	0.0
31,420	30,634	0.01	8.0	14.0	48.0	6.0	2.0	10.0	0.0
20,460	19,928	0.01	7.8	13.0	49.0	7.0	2.0	10.0	0.0
8,550	8,414	0.15	7.3	37.0	15.0	6.0	4.0	2.0	0.0
1,310	1,294	1.39	19.0	19.0	43.0	6.0	2.0	2.0	1.25
1,800	1,778	0.04	28.3	6.0	79.0	4.0	2.0	0.0	1.35
12,580	12,378	1.37	21.0	48.0	27.0	4.0	9.0	0.0	1.67
82,012	80,250	0.295	17.0	22.8	40.8	6.3	3.8	6.5	0.31

AS SAMPLED TOGETHER.

First Assayer.....		0.34	16.8	22.4	41.0	5.6	3.0	6.4	.....
Second Assayer.....		0.31	16.9	23.0	41.3	6.6	3.3	6.7	.....
81,540	80,154	0.325	16.85	22.7	41.15	6.1	3.15	6.55	.....

The term "assaying limits" refers not to careless work or incorrect methods but to the mechanical condition of the pulp, the delicacy of the balances and to the personal equation of the assayer.

Speed of Sampling Machines.—J. A. Church (*Eng. and Min. Journ.*, LXXXVI, pp. 951-953, 1019) reprints Johnson's tables and believes that the generally excellent results shown are due to the compact arrangement of the mill which tended to prevent acceleration. Taking Brunton's tables, it is seen that the quantity taken for a cut decreases rapidly with the size of the particles and that consequently the first cut is very important and in most cases should not be less than 20 per cent. The well-known error of automatic samplers caused by the selective action of the sides of the scoop as it enters and leaves the falling ore stream, together with the accurate results obtained by Johnson, lead to the advocation of large cuts at slow speed. A very instructive table of comparative speeds of various sampling machines and a sketch of Johnson's mill are contained in the article.

Sampling of Dumps.—H. S. Munroe (*Sch. Mines Quart.*, XXIX, pp. 233-237) outlined an excellent method of sampling mine dumps. The sampling of mine dumps involves many difficulties, such as the variations in value of different parts of the dump, the difference in size of the particles of ore and usually poor crushing facilities to handle the

large samples that should be taken; in fact, power-operated machinery should be used if possible. In sampling the face of the dump, the samples should be taken at stated intervals, say at corners of 25 ft. squares, or at equal intervals on contour lines. A still better method is to sink shafts through the dump. In any method the weighted average of the results should be used. The shaft method is preferable since a true distribution of values from top to bottom is obtained as well as the contours of the original surface which greatly facilitates the estimation of the volume of the dump. The author describes a very clever form of shaft for this work. It consists of a lined shaft made by driving sheet piling of corrugated iron, 5x1 ft., and bracing the piling from within by elliptical rings, three to every 5 ft., suspending the rings one from another by means of hooked rods.

Sampling Lake Iron Ores.—L. S. Austin (*Proceedings,* Lake Superior Min. Inst., 1908, pp. 225-233) advocates the use of automatic samplers for sampling ores. This can be economically done at the loading point or at the lake shipping points; if at the latter points, stock piles, in connection with lighter but more lasting docks than those at present in use, are installed. In connection with the foregoing article was printed the standard method adopted for sampling cargoes of ore at the lower lake ports. Briefly, the method calls for the sampling of the surface in each hatch by trowel, shortly after the unloading is started, and repeating the operation at such intervals of time as to represent four distinct stages of the unloading. In the case of dangerous conditions in the hold, samples may be taken from the cars, provided that not more than ¼ of the sample is so taken. Each trowelful should be of the same size and taken out at equal intervals over the surface and the true proportion of coarse and fines maintained. The sample thus obtained is worked down with the usual precautions. The moisture sample is taken when the vessel is half unloaded by taking portions at equal intervals along lines midway between the side and center of the boat on both sides of the center line of the boat.

Sampling Silver Ores at Copper Cliff.—A. A. Cole (*Proceedings,* Can. Min. Inst., Mar., 1908) describes the method in vogue at Copper Cliff, Ont., for the sampling of the silver ores from Cobalt, Ont. The great difficulty in sampling these ores is the occurrence of large amounts of metallics, either native silver or silver-arsenical alloys. The ore is weighed when received and crushed with a Buchanan jaw crusher. A small shovelful of the crushed ore representing each sack is taken for a moisture sample. The combined moisture samples are coned and quartered to 100 lb., crushed and worked down to 4 samples of about 11 lb. and these are dried at 80 deg. C. for about 20 hours. This weight

and moisture are used only for checking purposes. The crushed ore, if wet, is dried on steam drying plates, fed to an Allis-Chalmers ball-mill and ground through a 20-mesh screen. The ore as it passes through the screen is automatically sampled with a 27-in. Snyder sampler, which cuts out a 1/10 portion. The sampler is chain driven and makes 25 r.p.m. with one cut for every pound of ore. Fifty per cent. of the milled ore is finer than 100 mesh and 80 per cent. is finer than 50 mesh. The rejected portion and the sample are weighed and their combined weight, less moisture, is used as the basis of settlement. The moisture sample consists of one tube samplerfull from each pailful of the sample. This moisture sample is cut down to 3 portions of 6.6 lb. each and these dried. The main sample is shoveled over twice, coned and quartered. The two halves obtained are treated as independent samples and each coned and quartered to 100 lb., and from this point cut down with a Jones riffler to 20 lb. The opposite half of the final 20 lb. is sealed in case of future need. The two samples are thoroughly dried, ground in a Sturtevant disc grinder, assisted sometimes by pebble or drug mills, until the fines pass a 100-mesh screen. The fines and metallics are weighed and sampled separately, the fines having been mixed by one to three hours' rotation in a pebble-mill.

The metallics remaining in the ball-mill after the completion of the run are removed, weighed and melted and the resulting bullion, speiss and slag are weighed, sampled and accounted for separately. The time required for sampling a 30-ton lot is three days and the method ought to give very accurate results.

Sampling Spelter.—E. W. Buskett (*Eng. and Min. Journ.*, LXXXV, p. 812) describes the sampling and assaying of spelter. Two holes are drilled in each slab about 4 in. from each end. The drill is run at low speed and feed to produce thin cuttings. The drillings are broken up, mixed and quartered down to a 0.5 lb. sample. Detail methods are given for the determination of iron, lead and cadmium.

Discussion of Assaying Methods.

Borax as a Flux.—J. E. Clennell (*Journ.* Chem. Met. and Min. Soc. of South Africa, April, 1908) strongly advocates the use of a small amount of borax in crucible assays. Numerous experiments were performed and the conclusions seem well founded. Besides the other beneficial effects of borax, it was found that of 297 assays, with and without borax, the average was 0.016 dwt. gold higher with the borax flux.

Temperature in Cupelling Silver.—C. H. Fulton, O. A. Anderson, I. E. Goodner and I. D. Ossa (*West. Chem. and Met.*, IV, pp. 31-54) made an extensive investigation of the actual temperatures of cupella-

tion and certain other interesting phenomena in the cupellation of silver. A brief review could not do the work justice on account of its extended character.

Assaying in Cyanide Plant.—C. H. Jay (*West. Chem. and Met.*, IV, pp. 157-164) has described the routine work in the laboratory of a cyanide mill. The various methods used are clearly given in detail.

Assay of Cyanide Solutions.—W. H. Barton (*West. Chem. and Met.*, IV, pp. 67-68) describes a modification of the Chiddey Method of assaying cyanide solutions that overcomes a good many objections to this well known method. The modification consists of dropping into the solution a small piece of aluminum foil, just after the bulk of the lead sponge is formed, which seems to keep the sponge from disintegrating.

Belgian Assay for Lead.—J. Lövy (*Revue des Mines*, May, 1908) finds that the Belgian process of fire-assaying for lead (melting in an iron crucible) is superior to fusions in a clay crucible with iron nails. Results by the Belgian method are from 98 to 100 per cent. of the amount shown by the wet assay, while other fire methods yield only 92 to 95 per cent. Furthermore, the resulting lead button may be cupelled for silver, with results equal to those obtained by scorification. These results were obtained only with low-grade ores practically free from zinc.

Assay of Tellurides.—G. Borrowman (*Journ.* Am. Chem. Soc., XXX, pp. 1023-1027) made some investigations on the losses in assaying telluride gold ores. The ore used in the investigations was silicious and contained about 10.5 per cent. of tellurium; it was ground to pass a 120-mesh screen and was then further ground in an agate mortar until it all passed through bolting cloth. The fluxes used were sieved through a 40-mesh screen and thoroughly mixed with the ore. The fusions were made in a muffle at a temperature of about 1200 deg. C. The tellurium was dissolved out of some of the ore with nitric acid and the residue assayed. The results on the treated ore did not vary from results obtained on the untreated ore, showing that tellurium does not cause any gold loss in the slag. The slagging of tellurium is not increased by using larger amounts of litharge provided an excess is used at the start. It was also found that the temperature of fusion made practically no difference in the results. Lead buttons obtained from crucible fusions were cupelled at different temperatures but the gold recovery was apparently independent of the cupellation temperature.

The writer believes that the irregularities in assaying attributed to tellurium are overdrawn and that the presence of tellurium in the silver bead does not necessarily imply a gold loss in cupellation, although

of course it is not permissible in silver work. Some of the previously reported differences in assaying telluride ores are more likely due to a lack of homogeneity of the sample than to the presence of tellurium. In silver work a preliminary treatment with nitric acid and subsequent precipitation of the silver as chloride, adding the latter to the residue, is recommended.

Cupellation Loss of Silver.—A. H. Hook (*Eng. and Min. Journ.*, LXXXV, pp. 661) quotes figures obtained in daily work showing that the silver loss in cupelling varies inversely with the amount of silver present, other conditions being the same. It was also found that an excess of litharge increases the slagging loss in the crucible method.

Assay of Tin Ores.—Georg Hohegan (*Eng. and Min. Journ.*, LXXXV, p. 422) assays tin ores by dissolving out the other metals that may be present with aqua regia. When antimony is present the residue after the acid treatment is boiled with tartaric and then with hydrochloric acid. Tungsten is removed with dilute ammonia. The washed residue is dried, calcined and fused with potassium cyanide for 45 min. in a hot fire. If metallic tin is present the ore is first treated with nitric acid and then with the hydrochloric acid.

Assay Practice in British Columbia.—C. S. Baker (*Trans.* Can. Min. Inst., 1908) has written some notes on the practice of assaying in British Columbia. The various methods used in this district for the assay and analysis of ores and metallurgical products are described in detail. The crucible method is favored for silver in ores, the combination method for silver, the scorification method for gold in blister copper, and the crucible method for lead in ores.

Assay of Telluride Ores.—S. W. Smith (*Bulletin* 43, I. M. M.) has made a valuable contribution to the literature on the behavior of tellurium in assaying. Previous writers have been in substantial agreement regarding the use of a basic slag and a slow fusion at moderate temperatures as the main essentials in assaying telluride ores. The work done by the author was quite extensive and may be briefly summarized as follows:

Pot fusion.—(1) Tellurium reacts with litharge at moderate temperatures (700 to 900 deg. C.), before the formation of silicates has taken place, in accordance with the equation: $PbO+Te=Pb_2O+TeO$. (2) During the formation of silicates there is a partial reversal of this reaction, although the greater part of the tellurium which has once been oxidized dissolves to a red glass in the slag. (3) When litharge has reacted sufficiently to form silicates there is no longer any oxidizing effect on the tellurium. (4) There is no marked selective action by lead for either gold or silver during pot fusions of tellurides.

Scorification.—(1) Scorification of lead buttons from pot fusions tends to concentrate tellurium in the button, lead being more easily oxidizable than tellurium, especially at high temperatures. (2) When conducted at moderate temperatures some tellurium may be removed by the litharge before the formation of silicates. (3) The removal of tellurium from a lead button is more readily effected by fusing or "soaking" it under an ample amount of litharge at moderate temperatures (700 to 900 deg. C.).

Cupellation.—(1) The presence of tellurium in a lead button carrying gold and silver is accompanied during cupellation by a weakening of the surface tension of the molten metal. Cupel absorption is due to this effect of tellurium in assisting the general tendency of the film of litharge in the same direction, the result being that the molten metals tend to "wet" the sides of the cupel. If the amount of lead is insufficient, small portions of the metal become detached and either pass into the cupel or leave minute beads. (2) If tellurium, being less easily oxidizable than lead, especially at high temperatures, remain in sufficient amount towards the end of cupellation, complete absorption may occur. (3) Complete absorption and sub-division are entirely the result of insufficient lead. (4) Bone-ash cupels show a greater tendency to become "wetted," giving rise to sub-division, than patent cupels. (5) Silver protects gold by acting as a diluent during the cupellation of buttons containing tellurium. (6) Absorption loss due to the presence of tellurium is a function of the temperature of cupellation. (7) During cupellation tellurium is oxidized to the dioxide, TeO_2.

Assay of Silver Bullion.—F. T. C. Hughes (Inst. M. and M., ***Bull.***, 41) describes the very interesting method of assaying silver bullion at the Indian mint. The assay piece is dissolved in nitric acid, the silver precipitated with hydrochloric acid and weighed as silver chloride. The work at the mint consists of determinations on fine silver, Mexican dollars, Chinese "Sycee" silver and various coins. In most cases granulated samples are taken of the melt as well as punchings from the coins. The assay piece weighs 18.821 grains, which weight of pure silver weighs 25 grains as chloride. The 25-grain weight is the unit for weighing purposes and the other weights are decimal parts of it. The procedure consists of carefully adjusting the assay pieces to weight, check weighing them and placing them in bottles, adding 5 c.c. of 1.25 nitric acid to each, assisting solution by warming on a hot plate. The nitrous fumes are removed after solution is complete, by blowing, the bottles removed from the hot plate, adding 150 c.c. of cold distilled water and then 5 c.c. of 1.075 hydrochloric acid. The bottles are stop-

pered, allowed to stand a few minutes and then shaken by hand, washing down the sides and the stopper by means of a twist of the hand. The bottles are nearly filled with a rapid inflow of water and settled for an hour. The supernatent liquor is siphoned off to within one inch of the bottom, when the bottles are refilled with water, shaken and settled over night. In the morning, the stoppers are removed and the bottles inverted in a trough of water over small Wedgewood cups standing in white porcelain saucers. When the chloride has completely left the bottles, the bottles are removed and the cups taken from the trough. The water is decanted from the cups, the saucers examined for any float, and the cups then dried in hot air chambers, first at just below 100 deg. C., and then at 180 deg. C. The weighing of the chloride is rapidly done to prevent any absorption of moisture. Checks of both pure and standard silver are run with each batch of assays as a test of the accuracy of the work but not for correction purposes. The results are accurate but the time and labor items are large, which, however, is not so important in India.

Gold, tin, antimony, bismuth, mercury and lead interfere slightly and certain changes in the method are necessary when these metals are present. The silver is recovered from the chloride by melting with 70 per cent. of chalk and 4 per cent. of powdered charcoal in a crucible, using a cover of salt.

Assay of Battery Chips, Etc.—L. J. Wilmoth (*Journ.* Chem. Met. and Min. Soc. S. Africa, VIII, pp. 229-232) describes some experiments in the assaying of unusual material. Much has been written on the assaying of ordinary ores or products, but little regarding by-products, which very often present especial difficulties in their assaying. The particular experiments were made on battery chips and screens. These products present sampling difficulties to which are added equally unsatisfactory means of finding the assay values of the samples. As a preliminary experiment, a pure copper carbonate ore was mixed with a gold ore and fluxed with soda, borax, litharge, charcoal and enough sulphur to matte the copper. A nail was added to take up any excess sulphur. The resulting lead button collected practically all of the gold but not the silver. This method was then applied to the battery chips, of which a large proportion was metallic iron, with the object of fluxing the iron as a matte. The following charge gave excellent results: Chips, 1 a.t.; soda, 1.5 a.t.; fused borax, 1.5 a.t.; sulphur, 0.5 a.t.; silica, 0.5 a.t.; charcoal, 1 gram; one iron nail. The nail takes care of any excess sulphur, keeps the lead malleable and at the same time yields a matte free from lead. After a quick, hot fusion, the charge is washed down with a small quantity of litharge and charcoal and must remain quiet in the furnace for a full 20 min. after fusion.

The battery screens sample, consisting of small clippings, is carefully oxidized in roasting dishes in the muffle until all the iron has been oxidized. The completely oxidized sample is cooled, pulverized and assayed by any of the usual methods, or by the above-mentioned matte method. It was found that the matte method tended to yield higher gold results than other methods. In the case of silver, a re-treatment of the matte was necessary since it retained considerable amounts of silver, although gold free.

Assaying at San Francisco Mint.—J. W. Pack (*Min. and Sci. Press,* XCVI, p. 324) describes the method for the preparation of proof gold and silver which he developed and uses at the San Francisco mint. Twelve oz. of gold cornets (if below 990 fine a preliminary refining is necessary) are dissolved in 36 oz. of aqua regia without heating, until violent action ceases, when heat is applied sufficient to complete the solution. The silver chloride present is settled out by allowing the solution to stand one or two days, when the solution is decanted and the silver chloride left behind, washed and filtered. The filtrate is evaporated to one-half its volume adding nitric and hydrochloric acids if crystallization starts. The nitric acid holds up the copper and the hydrochloric acid accelerates the reaction with the aluminum. The solution is diluted so that each 20 liquid oz. contains 1 oz. gold; the dilute solution causes the gold to precipitate in a spongy mass, in which condition it is easy to wash. The solution is allowed to flow at the rate of 90 drops per min. into a jar containing 2 or 3 strips of 0.25-in. aluminum. The reaction is completed in an hour, the aluminum removed, and the solution settled. The precipitated gold is washed till acid free. Any remaining pieces of aluminum are dissolved with hydrochloric acid and the gold again washed till free from acid and copper, dried and melted in a glazed clay crucible.

For proof silver, fine silver is dissolved in nitric acid and the solution diluted to contain 1 oz. of silver in each 20 liquid oz. of solution. After standing in the sun for three or four days, the clear solution is siphoned off and the silver precipitated with hydrochloric acid, agitating the solution during its addition. The silver chloride is settled, the supernatant liquor decanted and the precipitate washed with hot water several times. The silver chloride is placed in an evaporating dish, covered with water and small amounts of nitric and hydrochloric acid added. Strips of aluminum are introduced and the reaction allowed to proceed. When the reaction is completed, the aluminum is removed, the silver washed with hot water, dilute hydrochloric acid added to dissolve any remaining aluminum and the silver again washed till acid and copper free. The silver is dried and melted in a fire clay crucible.

Assaying Apparatus.

Design of Assay Furnaces.—D. Clark (*Aust. Min. Standard,* July 22, 1908) describes the assay furnaces at Melbourne University. The construction of crucible and coke and coal-fired muffle furnaces is illustrated with drawings and several novel ideas are shown in their design.

Portable Assay Furnace.—E. W. Buskett (*Eng. and Min. Journ.,* LXXXV, p. 1150) designed a portable coal-fired assay furnace made of sheet iron and lined with fire-brick. He also designed a gasolene-fired furnace, the sheet iron shell of which is in the form of a cylinder 21 in. long and 20 in. diameter. The axis of the cylinder is horizontal and the cylinder rests on a cradle composed of two nearly flattened wagon tires. The lining of the furnace is composed of 90 parts of old muffles or burnt fire-clay (20 mesh) and 10 parts portland cement mixed with water to a stiff consistency and is tamped into place around wooden cores for the combustion chamber, flue outlet, etc. When thoroughly dry the cores are cut or burnt out. The back of the furnace is cast in a separate mould and then put in place. Drying cracks between shell and lining are filled with a grouting of plaster paris.

The Assay Balance.—A. Austin and S. Hunter (*Min. and Sci. Press,* Aug. 15, 1908) have written of the theory underlying the construction of balances and have also described the usual analytical and assay button balances. Several useful hints are given regarding the installation, care and operation of balances and weights.

AUSTRALASIA.

In the following tables the production of minerals and metals in each of the Australian States and New Zealand is separately itemized. In the tables relating to foreign commerce, however, the states are not separately treated, the combined statistics of the Commonwealth now being officially reported.

MINERAL PRODUCTION OF NEW SOUTH WALES. (a)

(In metric tons or dollars; £1=$5). (b)

Year.	Alunite.	Antimony and Ore.	Bismuth Ore.	Chrome Ore.	Coal.	Coke.	Cobalt Ore.	Copper Ore.
1896	1,394	134	42	3,914	3,972,060	26,774	.....	15
1897	736	172	3	3,433	4,453,729	65,229	.....	169
1898	2,988	83	29	2,145	4,781,551	83,538	119	181
1899	935	332	16	5,327	4,670,580	98,074	193	445
1900	1,946	252	11	3,338	5,595,879	128,238	145	867
1901	3,196	90	21	2,523	6,063,921	130,944	112	655
1902	3,702	58	10	508	6,037,083	128,902	35	3,190
1903	2,524	13	23	1,982	6,456,523	163,161	155	1,750
1904	376	111	41	404	6,116,126	173,742	6	2,470
1905	2,745	394	56	53	6,738,252	166,568	*Nil.*	457
1906	1,886	2,490	25	15	7,748,384	189,038	*Nil.*	(g)
1907	2,021	1,780	17	30	8,796,451	258,683	*Nil.*	(g)
1908	1,099	119	9	*Nil.*	9,293,377	288,413	*Nil.*	(g)

Year.	Copper Matte, Ingot and Regulus.	Diamonds. *Karats.*	Gold. (b)	Lead, Argentiferous. (f)		Lead. Pig. (f)	Molybdenite.	Opal.
				Ore.	Metal. (e)			
1896	4,464	8,000	$5,222,971	271,641	19,886	24	..	$225,000
1897	6,458	9,189	5,373,596	275,249	18,395	32	..	375,000
1898	5,577	16,493	5,847,680	394,676	10,270	1,745	..	400,000
1899	5,574	25,874	7,399,075	431,126	20,614	(d)4,896	..	675,000
1900	6,243	9,828	5,211,097	426,480	19,400	(d)6,807	..	400,000
1901	6,184	9,322	3,587,040	406,560	17,191	(d)3,304	..	600,000
1902	5,560	11,995	3,333,064	371,496	15,660	(d)4,685	16	700,000
1903	8,094	12,239	5,255,421	335,870	18,779	(d)8,561	31	500,000
1904	6,654	14,296	5,576,966	373,362	30,212	(d)5,977	26	285,000
1905	7,899	6,354	5,669,099	420,266	28,244	214	20	295,000
1906	9,911	2,827	5,249,762	377,890	22,573	60	34	282,500
1907	10,260	2,539	5,112,852	441,024	(h)	20,084	22	295,000
1908	9,215	2,205	4,646,451	354,468	(h)	15,174	9	209,000

Year.	Platinum. Kg.	Shale Oil.	Silver—Kg. (f)	Tin.		Tungsten Ore.	Zinc. (c) (f)
				Ore.	Block.		
1896	75.8	32,348	6,307	98	1,147	.....	.....
1897	61.2	34,635	4,666	14	799	.....	29,303
1898	88.9	30,164	16,580	1	630	.....	39,564
1899	19.8	37,307	21,525	5	749	.....	50,677
1900	15.6	23,229	24,080	15	1,067	.....	20,594
1901	12.1	55,650	13,950	11	659	.....	642
1902	11.6	63,886	33,195	23	502	.....	1,281
1903	16.5	35,332	34,195	556	949	9	21,066
1904	16.6	38,477	34,890	586	1,084	106	58,523
1905	12.4	38,838	12,987	726	817	228	105,189
1906	6.4	32,965	8,865	(h)	1,098	245	105,325
1907	8.6	48,068	63,573	(h)	1,945	409	241,015
1908	4.2	47,044	77,490	(h)	1,822	247	281,147

(a) From the Annual Report of the Department of Mines, New South Wales. (b) Where gold is reported £1=$4.866. (c) Spelter and concentrate. (d) Includes minor quantities of lead carbonate and chloride, the product of the leaching plant at Broken Hill. (e) Includes a small quantity of silver-sulphide. (f) Exported. (g) Included with metal. (h) Included with ore.

MINERAL PRODUCTION OF QUEENSLAND. (a)

(In metric tons or dollars; £1=$5.)

Year.	Bismuth Ore.	Coal.	Copper.	Gold (b)	Lead.	Manganese Ore.
1895	60	328,237	441	$13,056,414	369	361
1896	.....	377,332	589	13,235,842	628	305
1897	1	364,142	293	16,699,477	391	402
1898	8	414,461	63	19,016,763	252	68
1899	2	501,913	164	19,571,662	57	747
1900	8	505,252	286	20,002,290	207	77
1901	20	548,104	3,110	12,367,276	570	221
1902	1	509,579	3,845	13,238,500	271	4,674
1903	11	515,950	4,998	13,818,653	3,856	1,341
1904	20	520,232	4,440	13,210,869	2,079	843
1905	15	537,795	7,337	12,249,157	2,464	1,541
1906	7	610,480	10,238	11,257,316	2,854	1,131
1907	6	694,204	12,959	9,641,789	5,240	1,134
1908	22	707,473	14,932	9,613,051	7,207	1,403

Year.	Molybdenite.	Opal.	Silver. Kg.	Tin Ore.	Tungsten Ore.
1895	.....	$163,750	6,999	2,146	25
1896	.....	116,500	8,657	1,579	3
1897	.....	51,250	7,280	1,222	13
1898	.....	33,235	3,235	1,041	79
1899	.....	45,000	4,531	1,222	263
1900	.....	37,500	3,514	1,133	193
1901	.....	37,000	17,777	1,633	73
1902	(c) 42	35,000	21,813	2,118	56
1903	(c) 24	36,500	19,972	3,766	209
1904	(c) 22	17,750	20,370	3,986	1,564
1905	64	15,000	18,716	4,008	1,434
1906	108	15,000	24,257	4,900	785
1907	68	15,000	28,662	5,222	627
1908	89	12,500	36,200	4,903	426

(a) From *Annual Reports of the Under Secretary of Mines*, Queensland. (b) Where gold values are reported £1=4.866. (c) Includes bismuth and tungsten

MINERAL PRODUCTION OF SOUTH AUSTRALIA. (a)
(In metric tons or dollars; £1=$5.) (b)

Year.	Copper.		Gold. (b)	Iron Ore.	Lead.	Limestone.	Phosphate Rock.	Salt.	Other Metals and Minerals.
	Ore.	Metal.							
1896.....	354	4,176	$99,827	.....	45	.....	.....	.....	$3,775
1897.....	554	4,267	189,871	.....	74	.....	.....	.....	14,340
1898.....	545	4,847	51,949	.....	321	.....	.....	.....	2,800
1899.....	2,938	5,584	75,822	.....	370	.....	.....	.....	6,786
1900.....	2,405	4,964	70,522	.....	389	.....	.....	.....	2,055
1901.....	1,896	6,844	80,839	.....	69	.....	.....	.....	11,095
1902.....	2,620	6,956	121,056	.....	2,210	.....	.....	.....	3,710
1903.....	7,182	6,594	139,411	86,291	732	.....	1,016	40,640	500
1904.....	3,100	6,378	369,938	47,434	.....	44,135	3,048	40,640	990
1905.....	2,604	6,653	223,121	85 335	53	45,210	5,080	33,020	6,305
1906.....	535	8,339	131,382	76,430	51	32,451	5,944	55,880	11,045
1907.....	.....	8,058	99,948	85,954	.....	31,598	8,128	76,200	12,500

(a) From *Review of Mining Operation* by Hon. L. O'Loughlin, Adelaide, 1908. (b) Where gold is reported, £1=$4.866.

MINERAL PRODUCTION OF TASMANIA. (a)
(In metric tons or dollars; £1=$5). (b)

Year.	Coal.	Copper ore.	Blister copper.	Gold. (b)	Iron Ore.	Lead-Silver ore.	Tin ore.
1896.....	44,286	(d)	52	$1,156,035	203	21,150	3,867
1897.....	43,210	(d)	(d)	1,407,447	999	17,806	3,282
1898.....	49,902	(c)	(d)	1,369,706	1,296	196,707	2,882
1899.....	43,803	(d)	(d)	1,593,834	6,726	424,552	3,333
1900.....	51,549	4,221	9,343	1,538,727	5,141	453,519	2,693
1901.....	49,963	11,401	10,141	1,436,326	1,422	804,463	2,516
1902.....	49,647	8,630	7,869	1,467,454	2,424	47,226	1,989
1903.....	49,856	3,891	6,791	1,237,925	6,076	43,103	2,414
1904.....	62,090	(d)	8,826	1,362,587	6,950	51,959	2,104
1905.....	52,825	(d)	9,919	1,520,101	6,401	76,424	3,953
1906.....	53,742	2,270	8,847	1,240,650	2,642	88,513	4,545
1907.....	59,833	1,261	8,378	1,350,836	3,048	91,216	4,412
1908.....	62,044	1,204	8,974	1,179,950	3,657	62,022	4,593

(a) From *Statistics of the Colony of Tasmania*. (b) Where value of gold is reported, £1=$4.866. (c) Included with lead-silver ore. (d) Not reported.

MINERAL PRODUCTION OF NEW ZEALAND. (a) (b)
(In metric tons or dollars; £1=$5.) (c)

Year.	Antimony Ore.	Chrome Ore.	Coal.	Coke.	Copper Ore.	Gold. (c)	Kauri-gum.	Manganese Ore.	Silver—*Kg.*
1896.....	21	.....	805,537	107	.....	$5,067,589	7,240	66	2,933.3
1897.....	10	.....	854,164	..	.....	4,769,673	6,748	183	5,719.8
1898.....	.....	.....	921,546	9	2	5,258,642	10,063	220	9,140.0
1899.....	.....	.....	990,838	18	.....	7,363,100	11,294	137	10,985.6
1900.....	3	28	1,111,860	.....	12	7,005,103	10,322	166	10,202.0
1901.....	30	.....	1,259,521	.....	3	8,533,908	7,662	211	17,762.0
1902.....	.....	128	1,386,881	.....	.....	9,495,673	7,549	...	20,970.3
1903.....	.....	.....	1,542,953	.....	6	9,916,086	9,507	71	28,364.3
1904.....	.....	.....	1,537,838	.....	.....	9,671,180	9,203	196	34,042.3
1905.....	.....	.....	1,585,756	15	4	10,189,093	10,353	55	36,695.0
1906.....	.....	.....	1,613,301	5	.....	11,050,219	9,300	16	43,251.5
1907	100	.....	1,860,305	15	57	9,985,766	8,847	5	48,603.5
1908.....	5	.....	1,890,751	.....	13	9,755,303	5,618	*Nil.*	53,834.8

(a) From New Zealand Mines Statement, by the Hon. James McGowan, Minister of Mines, Wellington. (b) The exports are stated to be identical with the production, with the exception of coal, the exports of which were as follows: In 1896, 80,796 long tons; in 1897, 77,280 tons; in 1898, 57,333 tons; in 1899, 90,912 tons; in 1900, 116,216 tons; in 1901, 162,197 tons; in 1902, 191,696 tons; in 1903, 154,769 tons; in 1904, 165,220 tons; in 1905, 122,817 tons; in 1906, 141,641 tons; in 1907, 128,950 tons and 100,502 long tons in 1908. (c) Where gold is reported £1=$4.866.

MINERAL PRODUCTION OF VICTORIA. (a)
(In metric tons or dollars.)

Year.	Coal.	Lignite.	Gold. (c)	Stone, Building, etc.	Tin Ore.
1896	230,187	5,908	$16,640,997	$485	47
1897	240,057	4,894	16,799,824	(e)125,000	48
1898	246,845	2,915	17,205,547	100,000	87
1899	266,578	(b)	17,662,410	(b)	158
1900	215,052	(b)	16,767,261	175,000	71
1901	212,678	152	16,320,029	225,000	78
1902	228,777	(b)	14,899,876	266,975	10
1903	65,230	5,752	15,360,815	213,245	34
1904	123,695	*Nil.*	15,824,952	1,488,075	72
1905	157,649	*Nil.*	15,443,438	(b)	126
1906	163,201	*Nil.*	15,962,804	(b)	108
1907	140,802	*Nil.*	14,377,166	(b)	105

(a) From Annual Reports of the Secretary for Mines of the Colony. (b) Not reported. (c) Where gold is reported, £1=$4.866. (e) Estimated value.

MINERAL PRODUCTION OF WESTERN AUSTRALIA. (a)
(In metric tons or dollars.)

Year.	Antimony.	Coal.	Copper Ore.	Gold. (b) (c)	Iron Ore.	Lead Ore.	Limestone.	Silver. *Kg.*	Tin Ore.
1900	..	120,205	6,282	$27,461,865	12,448	272	16,183	894	836
1901	..	119,721	10,319	32,698,941	20,898	(d)21	18,501	1,893	746
1902	..	143,145	2,298	37,026,119	4,877	(d)36	5,162	2,500	630
1903	22	135,568	20,854	40,560,927	224	*Nil.*	1,301	5,229	830
1904	..	140,773	4,033	39,557,933	1,465	*Nil.*	13,612	12,416	869
1905	..	129,402	2,389	38,045,366	3,264	*Nil.*	9,291	11,189	1,096
1906	*Nil.*	152,151	7,548	35,888,278	1,300	*Nil.*	9,624	8,776	1,518
1907	*Nil.*	144,651	19,282	34,579,849	1,112	10	3,660	5,887	1,650

(a) From the *Report of the Department of Mines of Western Australia.* (b) £1=$4.866. (c) The value of gold produced in 1895 was $4,280,855; in 1896, $5,200,821; in 1897, $12,481,176; in 1898, $19,418,735. (d) Silver-lead ore.

MINERAL IMPORTS OF AUSTRALIA. (a)
(In metric tons, cwts. of 112 lb., or dollars; £1=$5.) (b)

Year.	Cement. *Cwts.*	Coal.	Coke.	Copper. Ore. *Cwts.*	Gold. (b)				
					Ore.	Bullion.	Specie.	Foil. (c)	Total Value.
1900	1,182,442	7,714	44,169	31,386	$ 14,880	$4,556,007	$ 78,888	$51,224	$4,700,999
1901	1,422,647	10,141	36,814	14,520	37,473	3,709,848	18,053	34,704	3,800,078
1902	1,074,482	5,149	9,846	29,236	2,375,513	3,834,510	505,899	30,028	6,740,950
1903	954,606	389	4,294	5	66,908	5,935,300	6,530	38,680	6,047,918
1904	561,237	398	4,270	12	68,309	5,684,164	6,297	43,215	5,801,985
1905	700,245	7,866	5,553	80	103,709	7,067,534	422,127	52,144	7,645,514
1906	793,928	706	6,202	873	93,116	10,053,463	397,990	53,356	10,597,925
1907	513,326	14,973	9,981	3,652	136,352	6,920,111	9,967	45,227	7,111,657

Year.	Graphite. *Cwts.*	Iron and Steel.			Lead Mfrs. *Cwts.*	Petroleum Products.		
		Bars, Rods, Girders, Sheets etc. *Cwts.*	Galvanised Plates and Sheets. *Cwts.*	Pig and Scrap. *Cwts.*		Kerosene—*gal.*	Naphtha—*gal.*	Paraffin.
1900	3,020	2,223,731	983,399	985,265	.....	11,125,905	48,863	1,275
1901	3,419	2,081,423	905,709	732,512	.....	20,924,640	114,092	1,040
1902	2,659	.....	1,104,701	766,725	8,300	10,399,931	116,170	1,913
1903	5,557	1,211,437	886,570	989,998	9,525	15,009,609	127,445	2,163
1904	4,263	1,399,783	1,027 859	883,397	6,243	14,791,319	277,737	530
1905	4,386	1,482,334	1,112,467	940,757	8,859	16,416,734	292,670	.....
1906	6,531	1,878,851	1,245,211	1,220,236	14,830	15,473,570	488,961	.....
1907	6,991	1,225,852	1,414,052	985,901	2,940	19,273,955	683,679	2,758

Year.	Potassium Nitrate. Cwts.	Quicksilver.	Salt.(d) Cwts.	Silver. (b) Ore. Cwts.	Silver. (b) Bullion. Kg.	Silver. (b) Specie.	Sulphur. Cwts.	Zinc. Spelter. Cwts.
1900	8,142	63.2	486,457	190	190.4	$1,226,208	109,647	13,582
1901	6,559	91.0	560,560	16,385	14.9	772,020	99,270	14,291
1902	7,955	92.6	571,548	5,562	13.6	439,186	173,176	20,965
1903	4,659	87.5	312,681	.....	14.2	160,111	180,719	14,197
1904	7,812	92.6	355,599	.....	39.8	154,534	252,744	23,316
1905	9,010	82.1	492,727	.....	3908.0	261,397	177,304	26,211
1906	8,112	78.6	326,042	380	9756.4	703,820	269,704	24,233
1907	8,571	59.5	409,852	2	114	375,937	264,060	24,026

(a) From Trade and Customs Returns, Commonwealth of Australia. Previous to 1900 each Colony reported its own imports and exports. (b) Where gold, silver or platinum values are reported, £1=$4.866. (c) Includes a small quantity of silver foil. (d) Brown Rock and N. E. I.

MINERAL EXPORTS OF AUSTRALIA (a).

(In metric tons, cwts. of 112 lb., or dollars; £1 = $5.)

Year.	Alunite. Cwts.	Antimony Ore. Cwts.	Bismuth Ore. Cwts.	Cement. Cwts.	Chrome Ore. Cwts.	Coal.	Coke.	Cobalt. Ore. Cwts.	Copper. Ore. Cwts.	Copper. Ingot and Matte. (Cwts.)
1900	38,300	5,197	194	.....	48,300	1,774,980	6,005	2,865	90,589	350,382
1901	62,920	2,206	993	.....	41,035	1,750,066	4,465	2,212	231,644	389,041
1902	72,880	1,428	156	.....	10,000	1,657,621	6,080	748	165,149	464,715
1903	49,600	947	832	11,168	39,022	2,063,016	27,345	3,080	61,560	616,277
1904	7,400	2,177	1,918	26,305	7,941	1,637,113	2,771	167	90,098	540,998
1905	54,040	7,811	2,222	17,283	(c)	2,058,190	2,316	1,320	17,380	632,183
1906	37,120	66,188	1,574	39,737	(c)	2,094,793	11,382	.....	33,476	744,357
1907	41,750	74,440	653	75,600	54,503	2,689,917	35,063	.....	157,071	853,236

Year.	Gold. (b) Ore.	Gold. (b) Bullion.	Gold. (b) Specie.	Gold. (b) Total Value.	Iron and Steel. Bars, Rods, etc. Cwts.	Lead. Pig and Matte. Cwts.	Lead. Argentiferous. Cwts.	Lead. Manufactures. Cwts.
1900	$2,379	$19,604,657	$41,898,304	$61,505,340	6,263	379,259	655,129	21,797
1901	65,341	22,416,198	43,233,515	65,715,054	4,396	281,391	668,955	22,611
1902	1,214,208	20,736,800	41,954,939	63,905,947	3,182	365,830	638,359	17,429
1903	80,591	29,691,889	53,634,629	83,407,109	5,753	633,816	553,308	28,783
1904	46,894	27,073,767	49,284,833	76,405,494	4,952	1,626,292	790 435	20,552
1905	49,507	25,788,574	27,523,288	53,361,369	4,821	1,302,428	753,008	34,629
1906	20,296	24,113,950	47,937,681	72,071,927	11,560	1,031,605	781,426	20,358
1907	17,501	(c) 19,615,286	33,329,093	52,961,880	13,163	1,774,207	907,790	21,765

Year.	Molybdenum Ore. Cwts.	Salt. Cwts.	Shale Oil.	Silver. Ore. (d) Cwts.	Silver. Bullion, Kg.	Tin. Ore. Cwts.	Tin. Block. Cwts.	Zinc. Spelter. Cwts.
1900	.....	100,893	16,792	1,598,789	192,328	6,815	72,172	37,352
1901	.....	156,760	19,587	1,630,252	196,136	5,012	60,129	1,722
1902	160	238,192	27,896	1,430,374	189,703	10,201	63,424	4,461
1903	783	155,613	14,483	1,653,794	202,730	26,900	82,473	60,206
1904	1,100	141,553	8,202	2,235,385	227,972	40,339	99,476	309,422
1905	1,381	174,987	11,818	581,651	208,134	55,153	106,963	3,006,372
1906	1,867	198,851	7,203	1,010,707	174,457	51,793	130,120	2,562,018
1907	2,025	(f) 189,194	5,686	907,790	294,679	65,006	131,407	5,393,784

(a) From "Trade and Customs Returns," Commonwealth of Australia.—Note. Previous to 1900 each Colony reported its own exports separately. (b) Where gold, platinum or silver values are reported £1 = $4.866. (c) Included with iron ore. (d) Includes lead ore. (e) Includes gold contained in matte. (f) Brown rock and N. E. I.

AUSTRIA-HUNGARY.

In the following tables the mineral and metal productions of the two Kingdoms are reported separately, together with that of Bosnia and Herzegovina.

MINERAL AND METALLURGICAL PRODUCTION OF AUSTRIA. (a)
(In metric tons.)

Year.	Alum.	Alum and Pyritic Shale.	Antimony.		Asphaltic Rock.	Bismuth Ore.	Coal.	
			Ore.	Metal.			Bituminous.	Lignitic.
1895	885	5,716	695	296	404	185.0	9,722,679	18,289,147
1896	919	25,184	905	422	390	*Nil*	9,899,522	18,882,537
1897	851	21,585	864	425	300	1.0	10,492,771	20,488,093
1898	1,037	28,914	679	343	643	*Nil*	10,947,522	21,083,361
1899	604	19,879	410	271	2,635	0.3	11,455,139	21,751,794
1900	620	3,004	201	153	887	4.0	10,992,545	21,539,917
1901	442	2,551	126	114	541	16.0	11,738,840	22,473,510
1902	62	2,866	18	24	897	8.0	11,045,039	22,139,683
1903	*Nil*	2,978	41	14	1,273	10.0	11,498,111	22,157,521
1904	*Nil*	2,337	103	36	1,435	1.7	11,868,245	21,987,651
1905	*Nil*	1,657	1,673	90	4,363	1.7	12,585,263	22,692,076
1906	*Nil*	1,020	1,071	*Nil*	2,840	2.7	13,473,307	24,167,714
1907	*Nil*	*Nil*	910	207	3,858	*Nil*	13,850,420	26,262,110

Year.	Copper			Copperas.	Gold.		Graphite.	Iron.	
	Ore.	Metal.	Sulphate.		Ore.	Bullion.		Ore.	Pig & Cast
1895	7,435	865	246	160	104	$49,841	28,443	1,284,911	660,549
1896	6,823	1,001	265	170	416	46,386	35,972	1,448,615	693,188
1897	7,405	1,083	276	125	647	44,924	36,504	1,613,576	762,685
1898	6,791	1,041	209	260	448	47,515	33,062	1,733,649	837,767
1899	6,731	1,123	235	475	387	50,306	31,819	1,725,143	872,352
1900	5,825	881	234	474	227	47,183	33,663	1,894,458	879,132
1901	7,406	776	256	472	143	31,234	29,992	1,963,246	884,844
1902	8,455	914	248	271	74	4,552	29,527	1,742,498	991,827
1903	12,688	961	310	298	2,148	5,316	29,590	1,715,964	970,832
1904	10,701	889	808	414	12,653	47,183	28,620	1,719,219	988,364
1905	10,677	870	540	116	35,987	133,218	34,416	1,913,782	1,119,614
1906	20,255	877	578	154	33,033	83,401	38,117	2,253,062	1,222,230
1907	10,400	592	579	*Nil*	30,711	91,719	49,425	2,540,118	1,383,524

Year.	Lead.			Manganese Ore.	Mineral Paint.	Petroleum.	Quicksilver.		Salt.
	Ore.	Pig.	Litharge.				Ore.	Metal.	
1895	12,919	8,085	2,034	4,352	3,164	188,634	86,663	535	278,875
1896	14,563	9,769	1,738	3,950	3,979	262,356	83,305	564	308,933
1897	14,145	9,680	1,626	6,012	3,653	275,204	88,238	532	331,064
1898	14,363	10,340	1,520	6,132	3,213	323,142	88,519	491	341,959
1899	12,820	9,736	1,526	5,411	2,055	309,590	92,323	536	342,050
1900	14,314	10,650	1,288	3,804	2,823	347,213	94,747	510	330,377
1901	16,688	10,161	1,317	7,796	1,701	404,662	97,260	525	333,238
1902	19,055	11,264	1,023	5,646	1,486	520,845	90,040	511	311,806
1903	22,196	12,162	923	6,179	1,691	672,508	83,321	523	359,015
1904	22,514	12,645	783	10,189	1,829	.....	88,279	536	369,877
1905	23,339	12,968	865	13,788	798	.....	86,856	520	343,375
1906	19,683	14,846	1,059	13,402	943	.....	91,494	526	378,912
1907	22,792	13,598	863	16,756	1,091	.....	89,370	527	395,009

Year.	Silver.		Sulphuric Acid.	Sulphur Ore.	Tin.		Tungsten Ore.	Uranium.		Zinc.	
	Ore.	Bullion. (*Kg*)			Ore.	Block.		Ore.	Salts.	Ore.	Spelter.
1895.....	18,113	40,061	7,431	830	24	60	35	31	4 5	25,862	6,456
1896.....	18,701	39,904	7,972	643	15	54	22	30	4.2	26,887	6,883
1897.....	20,628	40,026	8,515	530	16	48	31	44	4.4	27,463	6,236
1898.....	20,886	40,304	7,003	496	13	48	36	51	4.3	27,395	7,302
1899.....	21,454	39,564	7,814	555	54	41	50	49	7.6	37,100	7,192
1900.....	21,641	39,572	7,067	862	51	40	50	52	11.3	38,243	6.742
1901.....	21,363	40,205	7,073	4,911	42	49	45	48	13.0	36,072	7,566
1902.....	22,288	39,544	8,781	3,721	47	50	45	46	10.0	31,927	8,209
1903.....	21,958	39,812	9,105	4,475	57	34	49	45	6.0	29,544	8,949
1904.....	21,949	39,032	8,742	6,288	77	38	52	17	11.0	29,226	9,159
1905.....	21,047	38,453	1,007	8,407	52	53	55	16	13.9	29,983	9,226
1906.....	21,944	38,940	745	15,125	55	42	56	16	16.1	32,037	10,804
1907.....	13,380	38,742	*Nil*	24,099	53	47	44	11	11.2	31,970	11,208

(*a*) From the *Statistisches Jahrbuch des K. K. Ackerbau-Ministeriums.*

MINERAL AND METALLURGICAL PRODUCTION OF HUNGARY. (*a*)

(In metric tons or dollars; 1 crown=$0.203.)

Year.	Antimony.		Asphalt.	Asphaltic Rock.	Bismuth	Carbon Bisulphide.	Coal.			
	Ore.	Regulus.					Bituminous. (*d*)	Lignite. (*d*)	Coke.	Briquets.
1895 .	1,240	465	2,285	....	....	237	1,068,046	3,517,901	12,033	29,421
1896 .	1,361	500	2,740	....	....	352	1,132,625	3,761,728	25,550	31,179
1897 .	1,800	523	3,057	....	4.7	432	1,118,024	3,870,530	(*c*)	27,022
1898 .	2,201	855	3,125	....	3.1	771	1,239,498	4,516,581	(*c*)	31,781
1899 .	1,965	940	3,060	....	3.0	1,120	1,238,855	4,292,584	10,336	31,127
1900 .	2,373	846	2,700	....	2.0	1,250	1,447,047	5,128,277	12,973	69,353
1901 .	(*b*) 323	706	2,878	25,161	1.6	2,087	1,365,270	5,179,829	10,975	40,182
1902 .	(*b*) 748	683	2,774	24,873	0.9	2,320	1,162,785	5,132,053	8,204	88,089
1903 .	(*b*) 205	732	2,422	21,552	1.5	2,357	1,233,410	5,271,781	9,442	101,197
1904 .	1,080	1,007	2,221	17,660	0.9	2,512	1,155,320	5,319,349	5,103	103,451
1905 .	949	756	173	19,372	1.4	2,760	919,193	6,015,452	69,303	144,697
1906 .	1,807	954	4,111	34,664	2.0	2,756	1,103,529	6,229,712	79,930	151,657
1907 .	2,598	841	3,920	33,096	0.4	2,990	1,042,344	6,408,322	97,776	154,783

Year.	Copper.	Copperas.	Gold.	Iron.			Lead.		Litharge.	Manganese Ore.
				Ore. (*d*)	Pig.	Cast.	Ore.	Pig.		
1895.....	286	521	$2,118,100	9,955,262	322,206	....	.....	2.277	615	3,525
1896	159	595	2,131,876	1,269,680	393,698	....	.....	1,911	465	2,101
1897.....	213	592	2,038,839	1,421,130	402,503	....	525	2,527	155	4,039
1898.....	153	745	1,839,474	1,666,837	448,621	20,784	771	2,305	188	8,087
1899	165	771	2,039,504	1,537,600	451,637	19,631	526	2,166	213	5,073
1900.....	181	700	2,173,079	1,666,383	432,817	22,738	612	2,030	201	5,746
1901.....	162	805	2,189,692	1,557,300	430,686	20,640	(*b*) 10	2,029	238	4,591
1902.....	89	909	2,260,135	1,562,238	416,835	18,569	(*b*) 20	2.244	219	7,237
1903.....	45	982	2,243,521	1,439,132	396,674	18,875	(*d*)3,698	2,057	257	5,311
1904.....	63	1,277	2,437,998	1,524,036	370,297	17,203	(*d*)3,922	2,104	710	11,537
1905.....	73	920	2,439,451	1,661,358	403,719	17,563	686	2,146	209	5,706
1906.....	69	1,306	2,487,156	1,698,291	402,527	17,164	564	1,925	698	7,176
1907.....	85	1,212	2,391,137	(*e*) 646,604	423,134	17,103	8	1,468	441	5,705

Year.	Mineral Paints.	Petroleum	Pyrites.	Quicksilver *Kg.*	Salt.	Silver. *Kg*	Sulphur	Sulphuric Acid.	Zinc.	
									Ore. (*b*)	Spelter
1895	371	2,083	69,195	1,129	169,395	20,432	102	4,223	(*d*)	..
1896.....	334	2,168	52,697	1,100	180,133	19,916	138	3,550	(*d*)	..
1897.....	460	2,229	44,454	700	193,463	26,790	112	3,397	30	..
1898.....	247	2,471	58,079	6,800	197,593	18,799	93	1,318	30	..
1899.....	394	2,125	79,519	27,000	200,525	20,991	116	1,463	1,197	..
1900.....	370	2,199	87,000	31,800	212,957	20,202	123	1,371	326	..
1901.....	305	3,296	93,907	33,003	215,581	23,636	137	1,464	693	14
1902.....	283	4,347	106,490	44,000	217,079	23,020	105	1,193	364	..
1903.....	263	3,010	96,619	43,700	214,536	19,281	135	1 543	46	26
1904.....	273	2,134	97,148	45,169	230,943	16,352	143	1,329	203	..
1905.....	196	471	106,848	36,000	238,642	15,946	135	1,410	173	..
1906.....	221	2,692	112,623	50,100	245,402	13,642	133	1,457	243	146
1907.....	259	2,404	99,503	40,400	(*c*)	12,660	(*c*)	1,223	(*c*)	(*c*)

(*a*) From the *Annuaire Statistique Hongrois.* (*b*) Includes only that part of the crude output that was not smelted into a refined product. (*c*) Not reported. (*d*) Total production. (*e*) Exported.

MINERAL AND METALLURGICAL PRODUCTION OF BOSNIA AND HERZEGOVINA. (*a*)

(In metric tons.)

Year.	Chrome Ore.	Copper.		Iron.		Lignite	Manganese Ore.	Pyrites.	Quicksilver.	Salt.
		Ore.	Metal.	Ore.	Pig.					
1895....	707	(*b*)	105	(*b*)	2,569	195,422	8,145	....	(*b*)	12,755
1896....	443	(*b*)	296	(*b*)	10,120	222,724	6,821	....	(*b*)	13,720
1897	396	3,847	135	37,095	15,506	229,643	5,344	....	(*b*)	13,919
1898....	458	3,760	156	57,935	15,263	270,752	5,320	3,670	4.0	14,496
1899....	200	3,980	180	67,030	13,730	303,000	5,270	....	3.3	15,030
1900....	100	3,008	141	133,454	38,960	394,516	7,939	1.700	6.7	15.791
1901....	505	3,696	199	122,569	39,296	445,007	6,246	4,570	9.3	16,865
1902....	270	3,657	166	133,348	43,992	424,753	5,760	5,170	7.2	17.348
1903....	147	1,073	191	114,059	39,833	467,962	4,538	6,589	8.1	18,459
1904....	279	640	115	127,297	47,678	483,617	1,114	10,421	8.1	18,021
1905....	186	670	39	122,540	43,074	540,237	4,129	19.045	10.0	(*b*)
1906....	320	765	25	136,513	45 660	594,172	7,651	11,347	5.1	22,671
1907....	310	245	*Nil*	150,684	48,946	621,179	7,000	7,229	1.2	21,148

(*a*) From *Oestr. Zeit. f. B.- u. H.* (*b*) Not reported.

MINERAL EXPORTS OF AUSTRIA-HUNGARY. (*a*)

(In metric tons or dollars; 5 crowns=$1.)

Year.	Alum.	Aluminum, Sulphate and Chloride.	Antimony.		Arsenic, Arsenious Acid, and Orpiment.	Asbestos.		Asphalt.	
			Ore.	Regulus.		Crude.	Manufactured.	Rock and Earth.	Mastic and Bitumen.
1895.....	60	231	193	369	36	122	10	145	1,183
1896.....	47	267	218	441	26	48	10	134	1,692
1897.....	70	210	289	359	16	56	19	102	2,593
1898.....	83	253	266	679	29	150	28	183	2,126
1899.....	54	233	562	240	47	71	60	1,143	2,619
1900.....	44	164	247	276	65	47	168	1,218	2,177
1901.....	55	211	179	385	80	36	165	198	1,909
1902.....	102	135	174	290	89	65	275	520	301
1903.....	77	14	128	249	63	89	495	921	483
1904.....	38	2	200	673	72	290	1,582	403	728
1905.....	68	34	.....	774	42	330	1,397	1,060	457
1906(*f*).....	68	80	.....	912	66	376	1,708	2,824	799
1907.....	75	81	.....	698	43	350	630	3,787	770

Year.	Barium		Chloride of Lime.	Cement.	Chrome Ore.	Kaolin and Feldspar.	Coal.		Coke.
	Sulphate. (b)	Chloride.					Bituminous.	Lignitic.	
1895	.....	.....	267	12,804	385	56,203	640,963	7,143,234	119,051
1896	.....	.....	114	16,721	142	67,381	658,368	7,562,721	116,006
1897	.....	.....	111	19,786	153	68,609	701,919	8,108,975	145,056
1898	.....	.....	113	23,989	121	74,003	824,730	8,351,955	194,289
1899	65	.....	203	38,193	53	78,537	879,337	8,662,788	252,971
1900	23	.....	192	46,761	22	103,178	815,097	7,864,410	262,793
1901	55	4,098	738	44,723	62	97,037	748,802	8,076,575	303,651
1902	64	4,552	426	39,920	51	100,546	691,680	7,888,218	234,911
1903	52	5,091	674	40,239	100	110,181	754,957	8,027,347	280,395
1904	74	4,233	254	43,110	36	127,984	815,570	7,588,555	253,886
1905	26	4,626	978	52,830	46	137,125	903,156	8,035,718	287,790
1906 (f)	2,395	4,503	271	64,883	102	133,326	750,420	7,150,339	246,914
1907	3,116	3,086	308	81,407	161	157,894	849,792	8,876,408	323,243

Year.	Fluorspar.	Copper.			Copper Sulphate.	Copperas.	Cryolite.
		Ore.	Crude and Old.	Bars, Sheets Plates, etc.			
1895	44	17	151	354	162	301	11
1896	40	12	228	189	47	392	2
1897	27	0.1	159	180	14	648	10
1898	22	12	173	266	29	539	22
1899	309	74	534	298	67	808	101
1900	45	801	471	200	57	748	237
1901	6	1,042	435	334	23	548	231
1902	42	1,018	436	381	44	857	363
1903	12	1,308	1,226	451	45	898	521
1904	36	574	747	577	50	1,170	574
1905	5	2,328	1,253	746	49	836	638
1906(f)	Nil.	341	1,007	816	99	861	Nil.
1907	Nil.	489	624	870	11	1,580	Nil.

Year.	Gold.			Graphite.	Gypsum.		Hydrochloric Acid.
	Ore.	Bullion. (e)	Specie. (e)		Crude.	Calcined.	
1895	1	$203,352	$8,885,815	11,923	1,496	1,439	1,460
1896	45	253,194	13,555,706	13,091	899	1,376	1,246
1897	37	158,827	18,598,931	14,229	662	1,804	1,439
1898	13	17,943	23,779,858	17,109	718	2,163	1,614
1899	67	17,864	12,711,454	19,451	634	1,539	1,495
1900	1	120,988	11,582,571	18,995	502	1,723	1,659
1901	0	42,427	6,880,888	14,900	461	1,206	1,632
1902	.....	22,939	13,485,087	16,771	550	1,041	791
1903	3	10,150	11,052,944	17,302	342	1,510	3,530
1904	64	5,278	9,649,605	17,430	392	1,510	3,722
1905	1059	9,338	10,995,089	18,535	363	1,652	4,085
1906 (f)	936	88,264	8,015,967	16,871	1,970	666	2,942
1907	996	1,234,291	13,061,517	21,704	3,841	801	3,706

Year.	Ore.	Iron.			Lime. Hydraulic and Caustic.	Magnesium. Chloride and Glauber Salts.	Magnesite (Calcined).
		Pig and Old.	Manufactures.	Iron and Steel Bars, Sheets, Wire etc.			
1895	165,402	9,786	18,698	9,993	34,093	661	(c)
1896	214,390	11,712	17,674	12,428	76,895	2,291	(c)
1897	247,856	12,084	21,064	17,387	83,110	6,910	(c)
1898	302,317	15,803	22,724	23,231	89,067	7,248	(c)
1899	326,951	27,738	30,822	50,197	85,570	5,721	(c)
1900	263,421	53 426	40,344	65,019	86,273	7,321	(c)
1901	229,624	26,304	46,508	28,841	82,399	7,960	40,236
1902	241,806	42,592	30,137	45,517	81,634	5,333	53,467
1903	252,520	60,237	40,807	63,031	95,644	2,360	69,058
1904	295,017	66,442	60,252	64,698	101,753	2,151	53,781
1905	373,077	63,780	63,828	69,672	94,751	1,272	92,359
1906 (f)	234,924	43,694	73,575	50,247	87,468	4,094	87,765
1907	220,767	60,345	56,452	43,458	89,305	6,896	113,695

Year.	Lead.						Manganese Ore.	Millstones.	Mineral Paints.	Nickel and Cobalt ores.
	Ore.	Ash.	Litharge.	Metal and Alloys.	Red and Yellow.	White.				
1895	3,758	118	782	208	24	233	425	1,977	2,244	139
1896	3,076	113	597	272	33	171	701	1,831	1,700	113
1897	2,438	114	355	241	24	47	622	1,773	1,621	117
1898	2,253	100	188	545	45	55	1,961	2,109	2,153	121
1899	2,502	99	188	258	45	41	1,127	1,904	2,061	75
1900	2,628	66	242	393	34	34	463	1,871	1,906	114
1901	4,143	112	179	68	32	23	398	1,971	1,947	120
1902	5,478	154	124	109	25	37	411	1,886	2,136	34
1903	8,961	147	145	152	19	25	724	2,311	1,873	12
1904	7,575	144	167	464	54	52	1,234	2,276	1,840	26
1905	7,944	342	141	957	60	39	995	2,232	2,091	16
1906 (*f*)	4,891	223	302	602	16	52	4,170	1,763	1,367	42
1907	8,360	420	255	205	9	54	5,272	2,422	1,697	29

Year.	Nitric Acid.	Ozokerite.	Petroleum Products.			Paraffin.	Potash.	Potassium Chloride.	Pyrites.	Sulphur.
			Peat and Peat coke.	Petroleum. (*d*)	Benzine.					
1895	418	5,054	3,753	5,317	.....	.....	5,665	1,074	383	989
1896	360	5,722	2,701	24,921	.....	.....	4,164	1,026	341	1,231
1897	310	5,153	1,655	14,682	.....	.....	5,997	1,005	255	947
1898	294	4,462	3,400	4,138	.....	.....	7,252	994	3,039	923
1899	420	5,441	4,010	11,756	20,646	10	10,113	974	5,201	885
1900	519	5,162	5,607	33,032	18,361	26	7,792	879	17,162	1,285
1901	632	2,717	4,558	19,804	17,021	14	4,234	909	16,491	1,225
1902	769	2,285	4,927	40,683	13,884	24	3,229	772	9,547	1,136
1903	908	2,258	3,638	74,454	14,000	1,153	3,409	802	10,857	1,123
1904	858	2,093	3,980	122,419	13,706	5,992	4,604	445	9,891	968
1905	1,377	1,614	3,746	200,736	8,187	8,996	5,511	1,048	9,168	859
1906 (*f*)	1,303	2,034	2,517	198,325	13,472	9,996	3,814	1,005	7,208	760
1907	753	1,813	4,010	212,526	12,637	14,758	5,864	1,280	5,646	802

Year.	Sulphuric Acid.	Tin.			Whet-stones.	Zinc.				
		Ingot and Old.	Bars, Plates, Sheets, etc.	Ash.		Ores.	Crude and Old.	Sheets, etc.	White.	Ash.
1895	6,466	53	90	248	2,169	7,491	504	1,158	1,688	179
1896	6,212	130	78	281	2,035	9,453	1,256	1,139	1,825	277
1897	7,903	87	75	306	2,323	12,914	770	993	1,673	197
1898	9,880	36	72	324	2,316	14,065	1,184	757	1,240	298
1899	12,422	167	77	273	2,215	20,461	1,614	1,313	1,096	73
1900	12,693	153	102	208	2,270	20,379	1,088	502	1,719	149
1901	10,373	162	109	257	2,359	23,150	1,374	813	2,720	167
1902	9,451	193	128	188	2,852	24,519	2,002	1,127	3,113	237
1903	8,369	292	111	158	2,599	15,108	4,420	729	3,446	267
1904	9,101	126	102	123	2,159	17,314	4,606	532	3,666	158
1905	12,823	197	94	78	2,355	19,602	5,023	498	3,861	113
1906 (*f*)	10,493	221	62	83	1,541	15,933	4,578	323	3,504	83
1907	15,027	333	83	159	1,899	19,516	4,608	351	4,873	159

(*a*) From *Statistik des Auswaertigen Handels des Oesterreichisch-Ungarischen Zollgebiets.* (*b*) Includes artificial barium sulphate. (*c*) Previous to 1901, magnesite was included with other minerals not elsewhere specified. (*d*) From 1895 to 1898 inclusive, includes crude and refined petroleum; from 1899 to 1905 inclusive, lubricating oil is also included. (*e* Where gold or silver values are reported 1 crown=$0.203. (*f*) Last 10 months only.

MINERAL IMPORTS OF AUSTRIA-HUNGARY. (a)

(In metric tons or dollars; 5 Crowns=$1.)

Year.	Alum.	Aluminum and Alloys.	Aluminum, Sulphate and Chloride.	Antimony.		Arsenic. (b)	Asbestos.	
				Ore.	Regulus—Kg.		Crude. (c)	Manufactures
1895	338	48	1,278	15	2,100	293	432	108
1896	359	50	1,128	16	700	309	185	165
1897	346	67	1,351	8	600	259	625	134
1898	338	101	1,822	12	28,200	287	609	138
1899	332	121	1,299	10	30,400	284	866	1347
1900	430	154	1,435	46	23,000	320	1,085	1238
1901	413	153	1,882	27	1,500	351	1,678	1032
1902	537	151	2,161	40	18,200	351	2,038	798
1903	508	150	2,670	42	87,200	371	3,395	1221
1904	602	231	2,346	64	21,000	384	2,517	1240
1905	774	477	2,775	.....	24,700	342	5,962	208
1906 (f)	513	216	2,840	.....	38,900	83	7,025	111
1907	545	255	3,200	.....	99,900	71	5,729	172

Year.	Asphalt.		Barytes.	Borax.		Cement.	Chloride of Lime.	Chrome Ore.
	Crude Rock.	Mastic and Bitumen.		Crude and Boric Acid.	Refined.			
1895	2,410	872	5,098	1,908	62	32,012	2,538	1,827
1896	4,715	1,621	5,377	1,363	76	35,290	1,989	1,891
1897	5,824	1,309	4,947	1,206	63	32,479	1,520	1,109
1898	5,973	1,117	5,012	784	185	30,745	2,351	2,206
1899	7,301	1,546	5,443	2,212	130	21,410	3,749	1,574
1900	8,301	1,564	5,945	3,056	93	25,747	3,326	2,823
1901	5,702	1,106	6,336	1,687	233	23,559	3,326	960
1902	5,732	1,273	6,266	2,168	174	18,658	2,596	2,668
1903	5,871	1,272	7,057	2,192	150	23,256	2,791	2,121
1904	8,211	1,064	6,238	2,752	142	20,259	3,407	1,209
1905	8,553	1,139	6,187	3,099	205	21,950	1,847	2,305
1906 (f)	13,381	895	9,654	3,519	126	21,833	2,491	1,612
1907	9,394	1,717	7,155	3,762	138	23,923	4,807	2,795

Year.	Kaolin and Feldspar.	Coal.		Coke.	Copper.			Copper Sulphate.
		Bituminous.	Lignitic.		Ore	Bullion and Scrap.	Bars, Sheets, Wire, etc.	
1895	6,532	4,503,003	16,797	533,402	31	11,747	98	895
1896	7,425	5,174,321	19,981	491,028	1	13,666	126	2,084
1897	6,913	5,121,475	19,609	533,463	81	15,926	94	6,822
1898	7,991	5,396,760	19,303	606,783	64	17,443	159	5,271
1899	8,152	5,296,700	20,879	564,005	Nil	16,185	156	2,345
1900	6,847	6,242,939	67,740	620,776	16	18,970	121	3,516
1901	7,697	5,827,332	22,253	612,209	112	17,504	83	2,822
1902	9,085	5,766,377	29,601	547,406	100	18,498	149	2,839
1903	9,940	5,907,760	30,007	519,281	209	18,701	89	3,526
1904	10,854	6,190,030	30,001	548,272	1,107	22,532	89	4,508
1905	13,656	6,418,042	36,000	554,147	1,397	22,652	73	3,791
1906 (f)	13,219	5,942,897	17,464	406,088	267	20,948	481	1,597
1907	17,961	9,692,645	23,699	677,750	44	26,181	622	3,981

Year	Copperas.	Cryolite.	Fluorspar.	Gold. (d)		Graphite.	Gypsum.		Hydrochloric Acid.
				Bullion.	Coin.		Crude.	Burned.	
1895	871	229	3,528	$3,470,945	$17,384,964	640	850	10,916	467
1896	575	265	3,821	8,674,371	16,956,256	697	821	11,736	529
1897	401	211	4,201	22,374,069	18,164,128	948	980	12,101	721
1898	466	275	4,169	323,086	8,853,354	1,109	991	13,300	766
1899	409	343	4,959	432,187	7,662,641	815	1,336	13,441	350
1900	343	342	5,649	1,111,831	7,230,251	302	1,348	15,462	577
1901	269	428	5,774	13,865,103	20,353,592	318	1,405	15,880	576
1902	274	447	5,902	14,509,019	15,695,960	221	1,588	16,430	588
1903	155	521	5,445	9,825,200	9,817,233	405	1,969	18,665	603
1904	238	313	7,061	12,703,740	8,586,394	423	2,284	19,357	459
1905	169	220	7,601	1,047,792	9,204,968	785	1,553	21,266	656
1906(f)	186	217	7,795	989,604	5,229,591	854	4,104	10,308	476
1907	186	613	8,779	1,089,657	5,670,855	934	5,813	11,981	629

Year.	Iron.				Lead.					
	Ore.	Pig and Old.	Manufactures.	Iron and Steel Bars, Sheets, Wire, etc.	Ore.	Pig.	Alloys, Crude.	Litharge.	Red and Yellow.	White.
1895	117,600	175,400	$3,990,400	30,909	416	208	8,974	355	371	187
1896	107,018	148,217	4,258,400	27,809	540	218	7,221	233	432	156
1897	134,778	164,433	4,582,400	18,625	441	148	5,887	224	543	111
1898	178,507	173,919	4,627,200	26,421	459	153	9,746	280	555	115
1899	212,412	126,371	4,395,356	12,340	465	235	8,836	224	466	80
1900	233,156	95,530	4,533,599	10,313	501	175	7,916	141	354	106
1901	218,476	90,287	4,443,670	10,902	1,270	311	10,722	189	433	135
1902	197,525	43,314	4,304,818	11,584	1,879	348	8,706	149	428	221
1903	217,979	47,354	4,508,224	11,025	1,355	409	9,190	141	423	173
1904	182,515	35,091	4,976,342	9,402	1,436	349	7,917	146	372	138
1905	228,149	49,383	5,722,976	.....	247	475	7,282	101	349	88
1906 (f)	232,558	57,341	6,153,698	5,085	189	.....	6,989	82	310	75
1907	390,322	155,148	6,336,239	13,807	204	.....	9,967	97	380	126

Year.	Magnesium Chloride.	Manganese Ore.	Millstones.	Mineral Paints.	Nickel, Old and Crude.	Nickel and Cobalt Ores.	Nitric Acid.	Peat and Peat Coke.
1895	1,353	2,772	1,229	4,244	168	1,020	16	1,993
1896	1,333	7,371	1,205	4,362	161	719	21	2,002
1897	1,530	8,018	1,275	4,553	157	55	23	2,189
1898	2,096	5,396	1,429	4,979	137	510	22	1,511
1899	2,043	5,855	1,458	5,106	119	198	39	2,075
1900	2,100	7,016	1,672	4,958	258	406	36	2,664
1901	2,529	6,367	1,595	5,109	277	788	22	2,896
1902	2,621	15,595	1,410	4,831	265	225	90	3,234
1903	3,118	38,529	1,395	4,733	268	385	7	3,097
1904	2,997	35,357	1,282	5,563	402	656	24	2,676
1905	3,495	30,483	1,467	6,018	632	391	14	2,432
1906 (f)	4,050	33,406	1,176	4,660	773	Nil.	12	1,918
1907	5,006	46,481	1,469	6,047	1,192	Nil.	12	4,460

Year.	Petroleum Products.			Phosphorus and Phosphoric Acid.	Potassium Salts.			Pyrites.
	Crude Oil.	Refined Oil.	Paraffin.		Carbonate.	Chloride.	Chromate.	
1895	120,479	16,876	.....	226	285	2,679	29	54,610
1896	69,013	17,943	.....	224	987	2,475	34	50,691
1897	70,873	21,249	.....	209	333	2,206	34	49,462
1898	58,580	22,299	.....	209	300	2,258	3	52,282
1899	75,885	21,823	6,968	221	526	3,264	1	54,844
1900	20,813	22,963	5,080	204	1,029	3,633	11	60,317
1901	22,545	18,067	5,294	222	1,442	4,356	21	54,202
1902	24,830	15,864	4,238	225	485	3,377	11	60,235
1903	19,710	19,382	2,598	237	197	3,727	9	73,335
1904	20,110	22,715	1,470	193	222	3,557	3	65,397
1905	18,974	24,961	888	222	154	3,864	5	86,338
1906 (f)	13,522	9,693	403	178	602	3,729	5 (e)	87,973
1907	18,342	11,433	523	219	114	4,807	38 (e)	130,270

Year.	Quicksilver Kg.	Salt.	Silica, Quarts and Sand.	Silver.		Slag and Slag Wool.	Slate and Other Roofing.
				Bullion, Kg.	Specie.		
1895	4,200	40,396	58,494	49,370	890,353	981	15,667
1896	1,300	53,680	59,150	138,420	137,244	240	15,932
1897	1,000	46,057	61,532	99,900	75,944	4,717	16,758
1898	6,300	41,870	70,396	15,400	103,424	9,655	16,025
1899	2,600	37,883	71,279	28,900	112,056	5,665	15,562
1900	1,300	30,822	77,980	29,300	199,955	4,679	13,047
1901	2,600	30,625	83,401	41,800	207,669	3,068	11,555
1902	1,300	46,128	92,617	177,900	237,104	4,176	14,378
1903	1,600	48,793	94,492	150,400	250,299	3,850	11,531
1904	2,500	94,103	97,364	36,700	420,413	4,716	9,170
1905	2,400	.....	104,195	36,100	143,152	4,094	8,852
1906 (f)	1,200	32,182	134,526	43,000	200,754	.....	6,020
1907	1,200	41,665	180,280	88,182	194,815	.....	15,303

Year.	Sodium Salts.						Sulphur.
	Bi-Sulphate.	Carbonate.	Carbonate. (Calcined).	Hydrate.	Nitrate.	Sulphate.	
1895.....	137	40	551	1,163	43,059	6,617	14,709
1896.....	144	57	1,332	835	33,086	4,678	15,221
1897.....	91	45	2,787	1,450	39,600	2,879	21,406
1898.....	89	53	2,408	1,498	41,773	4,476	20,655
1899	85	62	1,123	1,669	47,301	5,394	23,504
1900.....	73	104	1,141	1,836	54,559	5,110	27,795
1901.....	96	77	911	1,280	63,283	4,452	25,300
1902.....	17	97	312	1,030	39,958	5,997	23,878
1903.....	13	110	327	956	54,896	6,116	22,625
1904.....	103	103	1,109	659	54,837	5,409	30,505
1905.....	167	168	965	475	66,740	5,258	30,227
1906 (*f*)	86	382	303	218	49,862	7,508	26,755
1907.....	72	153	283	305	57,023	7,342	3,547

Year.	Sulphuric Acid.	Tin Ingot, Crude, Old, etc.	Whetstones.	Zinc.			
				Calamine and Other Ores.	Spelter.	Bars, Sheets Wire, etc.	White.
1895	1,566	3,038	3,559	7,691	17,156	611	510
1896.....	3,522	3,344	3,851	9,022	17,539	552	596
1897.....	5,877	3,467	4,151	7,863	16,599	356	577
1898..	9,724	3,760	3,490	14,112	17,471	453	697
1899.....	10,245	3,005	3,717	12,730	15,225	481	750
1900.....	10,643	3,439	3,643	14,181	17,844	667	875
1901.....	11,712	3,671	3,445	18,403	16,921	579	718
1902.....	12,474	3,638	3,599	20,723	17,034	651	636
1903.....	16,148	3,564	3,774	22,344	17,973	746	698
1904	19,878	3,528	4,272	24,039	20,787	731	840
1905..... ...	17,320	3,845	4,376	22,890	21,874	568	972
1906 (*f*)	17,020	3,320	4,377	24,014	19,467	595	347
1907.....	20,431	4,433	5,552	24,289	24,192	601	218

(*a*) From *Statistik des Auswaertigen Handels des Oesterreichisch-Ungarischen Zollgebiets*. (*b*) Includes arsenious acid and sulphide. (*c*) Includes burned asbestos. (*d*) The values of gold are figured at the rate of one crown = $0.203. (*e*) Potassium and Sodium. (*f*) Last 10 months only.

BELGIUM.

The mining and metallurgical production in Belgium, and the imports and exports, according to the latest official statistics, are as follows:

MINERAL, METALLURGICAL AND QUARRY PRODUCTION OF BELGIUM. (a)
(In metric tons except where otherwise noted.)

Year.	Barytes.	Chalk, Marl. *Cubic Meters.*	Coal.		Coke.	Flint. *Cubic Meters.*	Iron Ore.
			Bituminous.	Briquets.		For Earthen ware.	
1896	25,000	191,100	21,252,370	1,213,760	2,004,430	23,450	307,031
1897	23,000	204,600	21,492,446	1,245,114	2,207,840	23,050	240,774
1898	21,700	287,805	22,088,335	1,351,884	2,161,162	22,150	217,370
1899	25,900	351,800	22,072,068	1,276,050	2,304,007	25,185	201,445
1900	38,800	377,550	23,462,817	1,395,910	2,434,678	25,700	247,890
1901	22,800	449,000	22,213,410	1,587,800	1,847,780	17,700	218,780
1902	33,000	390,700	22,877,470	1,616,520	2,048,070	17,430	166,480
1903	21,000	501,920	23,796,680	1,686,415	2,203,020	16,250	184,400
1904	60,000	450,400	22,761,430	1,735,480	2,211,820	18,070	206,730
1905	26,000	372,000	21,775,280	1,711,920	2,238,920	12,800	176,940
1906	22,365	568,170	23,569,860	1,887,090	2,414,490	14,900	232,570
1907	23,000	478,880	23,705,190	2,040,670	2,473,790	15,050	316,250

Year.	Iron, Crude.					Iron, Manufactures of.			
	Forge Pig.	Foundry Pig.	Bessemer Pig.	Basic Pig.	Total Pig.	Merchant Bars.	Sheet and Plate.	Wrought	Other Mfres.
1896	362,451	84,275	193,518	307,779	950,414	81,394	112,597	851	298,163
1897	426,332	78,410	183,701	333,958	1,035,037	106,608	100,252	872	263,644
1898	306,875	93,645	173,085	307,891	979,755	123,993	91,686	993	267,521
1899	317,029	84,165	169,664	453,718	1,024,576	93,601	97,604	662	283,331
1900	305,344	88,335	176,557	447,271	1,018,561	61,458	73,572	1,411	284,591
1901	178,250	86,170	166,820	332,940	764,180	249,390	65,760	550	64,900
1902	104,540	254,710	199,170	510,630	1,069,050	260,290	62,740	450	58,150
1903	91,600	256,890	229,160	638,430	1,216,080	274,520	56,550	390	60,920
1904	99,350	224,410	217,390	742,040	1,287,597	246,240	41,000	370	67,580
1905	98,170	206,390	220,210	784,850	1,311,120	270,840	39,250	40	67,490
1906	96,090	218,225	177,900	870,860	1,375,775	265,010	37,540	20	55,680
1907	92,280	189,190	88,650	1,008,170	1,406,980	274,400	37,950	20	46,130

Year.	Steel.					Lead.		Manganese Ore.
	Ingots, Blooms and Billets.	Rails.	Tires.	Wrought.	Plates.	Ore.	Pig.	
1896	598,947	147,183	10,497	6,702	64,653	70	17,222	23,265
1897	616,541	136,911	10,870	23,104	64,266	108	17,023	28,372
1898	653,523	117,751	10,953	17,902	87,219	133	19,330	16,440
1899	731,249	123,119	11,212	32,180	68,051	137	15,727	12,120
1900	655,199	134,428	11,934	25,985	55,307	230	16,365	10,820
1901	529,340	132,260	12,380	3,310	83,810	220	18,760	8,510
1902	786,980	(c)268,220	12,790	2,910	94,360	164	73,357	14,440
1903	988,160	(c)351,540	17,810	2,920	118,200	90	68,700	6,100
1904	1,065,870	(c)266,900	23,540	4,300	149,270	91	23,470	485
1905	1,227,110	241,640	25,810	6,080	179,470	126	22,895	*Nil.*
1906	1,440,860	274,920	32,070	5,070	186,610	121	23,765	120
1907	1,521,610	314,760	34,700	5,190	157,850	210	27,450	2,100

Year	Mineral Paints. Ochers. *Cubic Meters.*	Phosphate of lime. *Cubic Meters.*	Pyrites	Slate. *Pieces*	Silver. *Kg*	Zinc. Ore. (Blende)	Zinc. Ore. (Calamine)	Zinc. Spelter.	Zinc. Sheets.
1896.....	700	297 470	2,560	35,980,000	28,509	7,070	4,560	113,361	36,238
1897.....	350	350,056	1,828	41,422,000	30,073	6,804	4,150	116,067	37,011
1898.....	290	156,920	147	42,311,000	116,035	7,350	4,125	119,671	35,587
1899.....	300	190,090	283	44,167,000	134,854	5,736	3,730	122,843	34,289
1900.....	300	215,670	400	43,941,000	146,548	5,715	3,000	119,317	38,825
1901.....	(b)2,100	(b) 222,520	560	39,030,000	169,450	4,445	2,200	127,170	37,390
1902.....	(b) 200	(b) 135,850	710	37,120,000	212,249	3,568	284	124,780	37,070
1903.....	(b) 200	(b) 184,120	720	38,953,000	232,740	3,565	65	131,740	42,280
1904.....	(b) 450	(b) 202,480	1,075	41,240,000	252,920	3,698	4	137,323	41,490
1905.....	(b) 300	(b) 193,305	976	41,435,000	201,935	3,929	*Nil.*	142,555	45,320
1906.....	(b) 250	(b) 152,140	908	43,801,000	173,535	3,858	*Nil.*	148,035	44,525
1907.....	(b) 200	(b) 181,230	397	40,102,000	178,020	3,485	5	152,370	45,330

(a) From *Statistique des Industries Extractives et Métallurgiques et des Appareils à vapeur en Belgique.* (b) Metric tons. (c) Includes beams.

MINERAL IMPORTS OF BELGIUM. (a)

(In metric tons or dollars; 5 f.=$1.)

Year.	Cement.	Coal.	Coal Briquets.	Coke.	Copper and Nickel. Crude.	Copper and Nickel. Hammered, Drawn or Rolled.	Copper and Nickel. Wrought	Diamonds Crude and Uncut.	Fertilizers (All).
1896.....	30,565	1,693,376	1,561	260,273	15,506	1,109	$188,931	.....	(b) 25,946
1897.....	17,681	2,017,344	632	269,606	14,821	1,418	193,242	.....	(b) 5,162
1898.....	34,039	2,202,517	1,756	180,590	14,947	1,821	205,705	.....	(b) 10,657
1899.....	18,649	2,844,274	10,722	296,508	8,327	2,174	226,853	.....	(b) 15,072
1900.....	12,773	3,288,513	21,814	289,673	13,768	2,087	231,800	$8,051,200	163,229
1901.....	13,558	2,930,874	17,160	154,247	11,381	1,780	272,600	8,463,200	149,984
1902.....	13,269	3,232,510	33,235	230,612	14,197	1,998	251,400	8,537,600	158,924
1903.....	19,698	3,554,807	43,835	308,877	13,602	2,035	313,400	15,557,800	188,389
1904.....	37,593	3,701,240	45,600	338,127	13,422	2,267	490,400	16,059,400	157,402
1905.....	34,610	4,230,313	72,643	356,136	12,379	2,421	439,017	18,674,270	174,435
1906.....	29,003	5,358,789	147,302	352,316	22,007	2,687	419,265	18,683,536	211,067

Year.	Gold (Including Platinum) Ore. *Kg.*	Gold (Including Platinum) Unwrought. *Kg.*	Gold (Including Platinum) Specie.	Iron. Ore.	Iron. Pig and Scrap.	Iron. Tin Plate.	Steel. Ingots, Blooms, and Billets.	Lead. Pig.	Lead. Manufactures.
1896.....	93	4,923	$ 599,540	2,069,676	378,191	3,203	28,435	35,221	$17,231
1897.....	.....	3,824	1,726,700	2,544,377	354,178	3,875	25,370	43,840	91,580
1898.....	8,390	1,282	372,000	2,252,553	370,117	3,848	25,142	54,867	50,723
1899.....	51	1,136	744,000	2,621,152	423,968	3,900	11,666	60,649	191,508
1900.....	1,250	1,728	459,420	2,528,615	371,726	5,036	19,705	58,141	143,000
1901.....	.....	.....	.....	1,768,441	222,230	(c)4,705	68,228	54,720	123,200
1902.....	.....	.....	.....	2,550,347	348,337	(c)6,608	103,286	71,085	164,800
1903.....	.....	.....	.....	3,054,808	387,884	(c)9,776	144,370	63,386	126,600
1904.....	8,516	2,385	256,680	3,359,430	388,732	(c)7,899	182,336	63,813	234,400
1905.....	49	1,483	571,600	3,382,832	558,414	(c)7,426	167,513	61,668	397,400
1906.....	11,749	6,934	1,288,360	3,549,391	778,260	(c)9,605	115,038	54,690	531,317

Year.	Lime.	Petroleum.		Salt.		Silver.		
		Crude.	Refined.	Crude.	Refined.	Ore.	Bullion. Kg.	Specie.
1896	11,522	.95	158,979	92,408	38,785	1,477	8,980	$6,461,840
1897	13,184	988	149,501	96,805	39,193	2,533	467,851	2,083,040
1898	12,674	382	161,281	92,200	50,136	461	299,369	7,655,200
1899	12,811	2,479	166,404	81,224	50,647	2,523	105,723	14,272,080
1900	11,448	1,751	158,064	97,812	59,375	923	11,366	7,324,560
1901	11,288	305	160,327	93,043	48,974	.....	.....	.....
1902	13,820	247	183,692	102,110	50,704	.....	.....	.....
1903	13,525	237	210,905	108,886	52,272	64	.....	.....
1904	16,896	1,893	192,805	110,516	54,615	310	.....	.....
1905	17,220	143	194,584	111,317	59,575	499	12,876	4,916,480
1906	12,035	*Nil.*	196,458	113,035	60,928	313	32,396	16,509,760

Year.	Sodium Salts.			Sulphur.	Tin. Block.	Zinc. Spelter.
	Carbonate.	Nitrate.	Sulphate and Sulphite.			
1896	(*d*)	194,202	(*d*)	14,399	4,617	20,182
1897	(*d*)	181,676	(*d*)	13,261	1,609	16,320
1898	(*d*)	152,164	(*d*)	13,322	1,208	17,441
1899	(*d*)	249,756	(*d*)	8,449	1,113	11,058
1900	20,603	153,318	49,020	17,516	1,563	11,478
1901	19,465	167,489	47,558	14,775	1,841	13,896
1902	13,584	135,937	60,721	12,367	1,416	17,830
1903	19,459	149,942	54,926	21,637	2,677	20,586
1904	28,006	157,005	44,122	24,788	3,416	17,424
1905	26,525	202,241	50,667	19,617	3,270	15,692
1906	19,438	175,180	64,706	18,627	2,975	15,553

(*a*) From *Statistique de la Belgique; Tableau Général du Commerce avec les Pays Etrangers*, Brussels. (*b*) Guano only is reported. (*c*) Includes wrought tin-plate. (*d*) Included under nitrate.

MINERAL EXPORTS OF BELGIUM. (*a*)

(In metric tons or dollars; 5 fr.=$1.)

Year.	Cement.	Coal	Coal Briquets.	Coke.	Copper and Nickel.			Diamonds. Crude and Uncut.	Fertilizers. (All Kinds.)
					Crude.	Hammered Drawn or Rolled.	Wrought.		
1896	277,615	4,649,799	459,974	863,067	11,700	2,073	$168,524	.....	(*b*)14,633
1897	322,024	4,448,544	615,074	909,496	9,994	1,996	198,665	.....	(*b*)14,044
1898	419,132	4,579,955	666,265	878,435	8,511	1,770	161,726	.....	(*b*)21,626
1899	445,602	4,568,938	525,625	1,008,470	4,665	2,111	261,060	.....	(*b*)18,213
1900	408,284	5,260,993	604,864	1,073,315	8,411	2,097	276,600	$ 8,601,000	258,366
1901	492,882	4,820,300	714,455	829,421	6,309	1,988	198,200	8,814,200	299,709
1902	542,547	5,078,278	671,700	824,256	7,320	1,656	235,400	8,855,400	359,607
1903	599,092	4,923,368	623,691	841,142	7,638	1,702	256,400	16,708,200	497,355
1904	589,295	5,067,037	539,364	879,383	7,255	2,243	182,200	17,028,400	555,956
1905	679,426	4,704,063	480,247	977,095	6,805	2,056	315,648	18,674,270	511,395
1906	813,329	4,972,340	459,753	856,475	14,661	2,032	287,400	19,993,394	639,326

Year.	Gold (Including Platinum)		Iron.			Steel. Ingots, Blooms and Billets.	Lead.		Lime.
	Unwrought, Kg.	Specie.	Ore.	Pig and Scrap. (*c*)	Tin Plate.		Pig.	Manufactures.	
1896	3,713	$2,666,620	389,235	63,906	7,952	1,145	31,366	$36,821	477,213
1897	2,547	605,120	410,817	60,678	1,191	1,201	35,988	33,286	520,488
1898	1,231	578,120	384,047	40,522	973	1,018	40,302	16,450	546,199
1899	504	998,200	318,415	76,047	1,436	1,259	41,618	37,022	537,357
1900	549	613,180	420,180	79,172	940	975	46,566	9,000	617,666
1901	.....	.....	327,499	70,027	(*d*) 642	290	47,971	6,600	579,123
1902	.....	.....	368,560	96,302	(*d*)1,023	1,463	58,495	9,400	623,617
1903	.....	.....	400,972	101,549	(*d*)3,577	3,047	7,765	8,200	659,125
1904	671	726,020	441,059	85,489	(*d*)2,378	5,250	59,334	13,400	706,351
1905	608	1,860,000	443,511	80,896	(*d*)1,322	19,075	53,378	53,669	720,123
1906	13,040	8,361,940	436,465	99,983	(*d*)3,478	31,860	54,207	67,939	699,300

Year.	Petroleum.		Salt.		Silver.		
	Crude.	Refined.	Crude.	Refined.	Ore.	Bullion, *Kg.*	Specie.
1896	2	29,321	1,434	129	19	40,118	$ 667,540
1897	1	18,068	493	231	423	57,933	20,851,320
1898	782	19,556	298	396	.....	107,395	13,083,640
1899	2,146	25,970	506	885	.....	54,358	13,483,160
1900	1,759	21,812	2,345	799	.....	33,331	1,304,840
1901	*Nil.*	22,091	1,611	2,454	.....	.....	.....
1902	*Nil.*	23,344	1,378	1,077	.....	.....	.....
1903	*Nil.*	29,155	928	618	.....	.....	.....
1904	*Nil.*	30,209	1,899	1,955	.....	.....	6,870,000
1905	*Nil.*	36,491	305	2,186	1,789	44,682	723,720
1906	*Nil.*	31,908	17	1,724	2	46,732	4,086,680

Year.	Sodium Salts.			Stone. Roofing Slate, 1000 *Pieces.*	Sulphur.	Tin. Block.	Zinc. Spelter.
	Carbonate.	Nitrate.	Sulphate and Sulphite.				
1896	(c)	42,857	(c)	15,435	5,335	1,055	100,369
1897	(c)	9,054	(c)	17,304	6,041	347	100,228
1898	(c)	106,252	(c)	16,948	6,355	508	108,507
1899	(c)	109,253	(c)	15,316	6,769	659	101,244
1900	25,569	39,346	10,577	12,836	7,363	495	969,233
1901	22,520	44,303	15,833	12,947	6,722	299	106,656
1902	12,881	33,319	14,316	13,715	7,349	234	118,118
1903	9,454	36,057	17,434	11,984	10,234	838	119,988
1904	15,049	36,970	18,715	11,362	9,020	815	116,289
1905	16,571	58,815	15,367	11,934	5,925	201	125,423
1906	20,619	60,548	7,707	11,596	5,841	294	132,774

(a) From *Statistique de la Belgique; Tableau Général du [Commerce avec] les Pays Etrangers.* (b) Guano only is reported. (c) Includes iron and steel filings. (d) Includes wrought tin plate. (e) Included under nitrate.

CANADA.

The statistics of mineral production in the Dominion of Canada as reported by the Geological Survey are summarized in the following tables. The statement of imports and exports for 1907 is for the nine months ending March 31, in consequence of a change in the law whereby the fiscal year was changed from June 30:

MINERAL PRODUCTION OF THE DOMINION OF CANADA. (a)
(In metric tons or dollars.)

Year.	Arsenic.	Asbestos and Asbestic.	Barytes.	Cement—Barrels.		Chromite.	Coal.	Cobalt.	Coke.
				Natural Rock.	Portland.				
1896	*Nil.*	11,113	132	70,705	78,385	2,124	3,398,091	.....	45,004
1897	*Nil.*	27,617	518	85,450	119,763	2,392	3,434,756	.....	55,042
1898	*Nil.*	21,577	971	87,125	163,084	1,333	3,784,532	.....	79,453
1899	52	22,938	653	131,387	255,366	1,796	4,467,021	.....	91,444
1900	275	27,797	1,213	125,428	292,124	2,335	5,087,060	.....	142,521
1901	630	36,477	592	133,328	317,066	1,274	5,648,308	.....	331,537
1902	726	36,657	994	127,931	594,594	900	6,524,180	.....	455,353
1903	725	37,902	1,055	92,252	627,741	3,509	6,933,107	.....	509,115
1904	(d) 66	44,131	1,253	51,555	771,650	5,511	6,812,834	.....	493,107
1905	*Nil.*	61,928	3,049	14,184	1,346,547	7,781	7,961,397	.....	622,154
1906	*Nil.*	72,025	3,628	8,610	2,139,164	7,936	9,033,073	.....	(b)
1907	317	82,117	1,829	5,775	2,368,593	6,527	9,533,442	.....	(b)
1908(f)	634	82,331	3,710	1,044	2,665,289	6,553	9,890,351	840	784,788

Year.	Copper. (In Ore, etc.)	Corundum.	Feldspar.	Gold. (c)	Graphite.	Grindstones.	Gypsum.	Iron Ore.	Iron, Pig. All kinds.
1896	4,260	(b)	882	$2,754,774	126	3,368	187,778	83,359	61,012
1897	6,032	(b)	1,270	6,027,016	395	4,147	217,340	45,989	52,612
1898	8,048	(b)	2,268	13,775,420	1,107	4,476	198,864	52,917	69,853
1899	6,838	(b)	2,721	21,261,584	1,188	4,091	221,821	67,678	93,367
1900	8,588	3	288	27,908,153	1,743	5,024	228,656	110,654	87,504
1901	17,155	403	4,852	24,128,503	2,004	4,155	266,476	284,477	248,859
1902	17,598	697	6,871	21,336,667	993	5,835	301,165	366,431	324,617
1903	19,357	880	12,633	18,843,590	660	5,023	285,242	239,715	270,182
1904	19,497	834	10,057	16,400,000	410	4,091	309,133	317,287	275,267
1905	21,596	1,492	10,617	14,486,533	491	4,093	395,341	263,113	475,491
1906	25,863	2,063	14,397	12,023,932	405	5,029	378,904	269,842	550,628
1907	26,025	1,716	11,414	8,264,755	525	4,881	431,286	.....	590,444
1908(f)	29,202	942	7,144	8,300,885	227	3,285	309,254	184,565	611,160(g)

Year.	Iron and Steel, Rolled.	Lead. (In ore, etc.)	Manganese Ore.	Mica.	Mineral Paints. (Ochers.)	Natural Gas.	Nickel. (In ore, etc.)	Petroleum, Crude. Barrels. (a)
1896	76,244	10,975	(d) 112	$60,000	2,142	$276,301	1,541	726,822
1897	78,253	17,695	(d) 14	76,000	3,542	325,873	1,813	709,857
1898	101,748	14,469	45	118,375	2,019	322,123	2,502	758,391
1899	112,412	9,914	1,434	163,000	3,555	387,271	2,605	808,570
1900	102,201	28,648	27	166,000	1,783	417,094	3,211	710,498
1901	113,799	23,537	(d) 399	160,000	2,025	330,476	4,167	623,392
1902	164,069	10,411	(d) 156	135,904	4,494	195,992	4,849	530,624
1903	131,588	8,226	83	177,857	5,683	202,210	5,671	486,637
1904	(b)	17,241	(d) 112	152,919	3,562	247,370	4,786	552,575
1905	(b)	25,391	(d) 20	168,170	4,632	314,249	8,565	634,095
1906	(b)	24,580	(d) 84	(d) 581,043	6,201	528,868	9,745	569,753
1907	(b)	21,570	.....	333,022	7,115	748,541	9,610	788,872
1908(f)	640,532(h)	20,747	.....	191,602	7,025	1,012,060	8,685	527,987

Year.	Phosphate (Apatite).	Pyrites.	Salt.	Silver—Kg. (In ore, etc.)	Soapstone and Talc.
1896	517	30,580	39,572	99,699	372
1897	824	35,291	46,574	172,891	142
1898	665	29,223	51,828	138,496	367
1899	2,721	25,112	53,820	106,116	408
1900	1,283	36,308	56,284	133,980	1,288
1901	937	31,982	53,901	172,292	235
1902	776	32,304	58,462	133,478	625
1903	1,205	30,822	56,644	99,489	898
1904	832	29,980	62,411	115,666	762
1905	1,180	29,713	41,159	185,539	454
1906	(b)	35,927	60,283	266,521	1,119
1907	650	35,494	65,936	390,259	1,391
1908(f)	1,447	42,934	72,537	397,505	976

(a) From *Reports Compiled by the Geological Survey of Canada.* (b) Not reported. (c) Gold values are calculated at the rate of $20.67 per oz. (d) Export. (e) One barrel contains 35 imp. gal. (f) From preliminary unrevised reports. (g) From Canadian ore, 99,420 sh. tons. (h) Total steel production.

MINERAL IMPORTS OF THE DOMINION OF CANADA. (a.)

(In metric tons or dollars.)

Year. (b)	Aluminum.		Antimony. (c)	Arsenic.	Asbestos. (d)	Asphalt.	Cement.
	Manufactures.	Ingots, Sheets, etc.					
1897	$5,717	.....	61	68	$19,032	342	$ 260,842
1898	7,102	.....	71	132	26,389	6,006	365,624
1899	9,275	.....	131	264	32,607	8,196	477,617
1900	12,543	.....	90	105	43,455	2,825	513,770
1901	16,202	.....	159	72	50,829	2,849	666,350
1902	30,496	.....	229	48	52,464	3,426	863,646
1903	14,201	$ 13,930	393	135	75,465	3,037	890,745
1904	16,065	101,427	190	188	83,827	7,093	1,014,713
1905	28,418	154,599	85	122	116,836	5,096	1,263,328
1906	23,565	168,405	183	202	138,000	7,178	1,003,022
1907(o)	20,656	218,399	146	158	127,509	11,929	540,006
1908(r)	37,197	131,762	220	228	191,204	14,113	865,275

Year.	Coal.		Coal Tar. *Barrels.*	Coke.	Copper. Ingots, Pig and Scrap.	Copper. Sulphate	Gold and Silver. Coin and Bullion. (g)
	Anthracite. (f)	Bituminous. (f)					
1897	1,321,767	1,604,517	23,661	75,580	22	516	$ 4,676,094
1898	1,324,856	1,735,576	26,702	122,499	476	738	4,390,844
1899	1,583,132	2,220,250	39,296	128,145	751	726	4,705,134
1900	1,500,542	2,512,334	50,484	170,405	519	752	8,297,438
1901	1,753,488	2,658,257	54,928	280,069	432	673	3,537,294
1902	1,498,773	3,208,005	55,376	242,298	801	711	6,311,405
1903	1,320,239	3,684,502	29,325	232,848	924	1,010	8,976,797
1904	2,064,444	4,230,436	55,172	200,590	960	795	7,874,313
1905	2,361,952	4,377,667	77,856	337,035	882	934	10,308,435
1906	1,996,183	5,003,029	66,540	435,561	1,191	844	7,078,603
1907(o)	1,260,723	4,022,843	43,017	363,286	1,186	897	7,029,047
1908(r)	2,803,681	7,681,464	130,329	561,677	1,638	1,161	6,548,661

Year.	Graphite.		Gypsum.		Iron and Steel.		
	Crude.	Manufactures. (h)	Crude and Ground.	Plaster of Paris.	Pig and Scrap.	Slabs, Blooms, Bars, Etc.	Alloys of Iron.
1897	$1,406	$38,537	482	440	23,442	2,566	387
1898	1,862	52,291	1,057	150	31,577	7,391	1,287
1899	4,979	57,824	310	225	69,819	5,640	1,053
1900	4,437	60,518	72	335	94,489	11,576	1,043
1901	2,357	75,536	289	228	59,033	10,659	1,372
1902	3,649	64,123	516	215	71,882	18,208	5,910
1903	2,870	69,676	1,007	286	129,641	17,396	5,762
1904	1,802	67,563	626	291	86,087	9,088	2,700
1905	2,499	75,288	2,972	3,595	90,698	14,420	11,738
1906	2,791	86,028	5,743	6,579	112,937	29,520	13,626
1907(o)	3,176	57,430	8,334	9,730	137,654	17,369	17,785
1908(r)	3,030	78,380	8,519	6,955	190,994	35,534	16,139

Year.	Kainite.	Lead.				Lime.		Mineral Paints, (Ochers)
		Pig and Scrap.	Bars and Sheets.	Litharge.	Pigments and Zinc White.	Burned. *Barrels.*	Chloride of.	
1897	206	2,962	477	546	4,678	16,108	1,361	682
1898	49	4,012	1,008	519	5,754	12,850	1,765	965
1899	30	5,202	2,032	432	6,583	15,720	1,857	1,110
1900	143	2,829	703	415	6,661	12,865	1,967	1,122
1901	88	(i)3,871	739	505	4,647	19,657	1,605	1,031
1902	85	(i)5,548	844	590	7,071	24,602	1,806	1,148
1903	259	(i)4,471	523	632	8,715	31,108	2,104	1,459
1904	339	4,292	.....	.....	7,679	54,359	2,080	1,256
1905	306	2,589	800	811	9,695	98,676	2,507	1,417
1906	306	3,751	730	461	6,947	134,334	2,645	809
1907 (o)	511	3,811	622	513	2,215	88,919	2,302	570
1908(r)	743	2,902	782	864	5,743	129,379	3,421	788

Year.	Nickel.	Petroleum Products—*Gallons.*		Platinum.	Potassium Salts.		Quicksilver.
		Illuminating oil, etc., Crude or Refined.	Paraffin Wax and Candles.		Except Saltpeter.	Saltpeter.	
1897	$4,737	8,415,302	74	$9,031	265	456	35
1898	5,882	9,074,311	75	9,781	244	627	27
1899	9,446	10,394,208	70	9,671	472	930	47
1900	6,988	9,633,647	35	57,910	733	602	39
1901	12,029	11,082,822	74	20,263	476	581	64
1902	15,448	13,220,005	123	19,357	771	690	44
1903	26,177	18,799,312	307	21,251	1,060	916	75
1904	14,682	24,521,115	228	28,112	1,151	898	69
1905	19,076	13,229,855	98	61,719	945	1,048	47
1906	15,976	10,981,611	375	54,494	1,317	1,141	68
1907 (o)	19,461	8,066,403	189	113,967	1,074	638	44
1908 (r)	.....	8,844,129	102	63,582	3,396	2,653	81

Year.	Sal-Ammoniac	Salt.	Silex.	Sodium Salts Except Chloride.	Sulphur.	Tin and Tinware.	Zinc.
1897	69	103,337	116	13,938	3,932	$1,274,108	542
1898	38	96,962	141	16,026	17,248	1,550,551	1,595
1899	53	88,397	179	20,742	11,121	1,372,813	852
1900	60	92,823	182	16,748	9,584	2,418,455	1,304
1901	76	103,402	162	18,631	10,827	2,339,109	931
1902	78	114,629	199	17,133	11,180	2,203,958	1,582
1903	114	112,188	159	18,887	11,077	2,712,168	1,209
1904	93	103,635	252	25,118	8,786	2,389,557	1,540
1905	143	97,723	405	26,219	10,633	2,791,757	1,721
1906	209	99,788	338	30,401	19,512	3,105,876	3,383
1907 (o)	130	73,156	542	25,068	11,725	2,473,572	2,761
1908 (r)	172	105,286	1,131	39,154	23,494	1,619,647	2,521

EXPORTS OF DOMESTIC MINERAL PRODUCE FROM THE DOMINION OF CANADA (a).
(In metric tons or dollars)

Year (b)	Antimony Ore.	Asbestos.	Chromite.	Coal.	Coke.	Copper (c).
1897	.....	9,954	(k) 1,911	1,000,061	1,092	4,596
1898	1,118	16,718	(k) 1,527	981,963	3,275	6,319
1899	.....	13,176	(k) 1,369	1,035,245	4,024	3,843
1900	6	16,483	(k) 334	1,489,139	12,588	6,274
1901	219	24,242	(k) 2,049	1,713,737	60,129	11,954
1902	13	30,011	(k) 672	1,649,278	52,873	13,789
1903	128	27,823	658	1,796,689	39,616	13,445
1904	87	31,444	2,103	1,494,106	61,750	20,279
1905	340	37,320	3,702	1,465,809	116,387	17,431
1906	388	40,367	1,640	1,651,203	50,004	20,062
1907 (e)	832	37,194	604	1,165,809	44,669	11,845
1908 (r)	693	53,543	1,585	1,702,673	50,343	25,824

Year.	Gold. Quarts, Dust, etc.	Graphite	Grindstones	Gypsum Crude.	Iron Ore.	Lead (p).
1897	$ 2,804,101	78	$15,760	163,829	(n) 3,056	13,636
1898	3,387,953	348	13,785	163,660	(n) 1,975	19,944
1899	3,272,702	662	18,619	148,565	(n) 2,881	15,445
1900	14,143,543	1,742	22,196	211,792	(n) 5,012	8,998
1901	24,445,156	1,246	33,304	156,080	(n) 54,208	29,747
1902	19,668,015	783	21,878	243,629	(n) 478,503	13,890
1903	16,437,528	530	14,169	271,899	(n) 267,000	7,386
1904	18,715,539	269	12,676	247,741	(n) 214,309	7,329
1905	15,208,380	201	27,985	290,574	204,091	23,094
1906	12,991,916	180	15,793	367,203	134,270	6,158
1907 (e)	7,226,954	3	33,929	349,780	31,011	8,330
1908 (r)	8,817,041	167	28,726	340,235	23,863	12,650

Year.	Manganese Ore.	Mica.	Nickel in Ore, Matte, etc.	Petroleum, Crude and Refined.	Pyrites.	Salt, *Bushels.*	Silver, *Kg.* (In Ore, Matte, etc.)
1897	74	217	3,415	1,931	14,219	4,702	127,440
1898	7	231	6,097	9,530	18,752	5,559	211,012
1899	24	538	6,546	4,268	11,707	5,209	137,400
1900	57	490	6,122	6,758	13,507	15,151	71,015
1901	33	444	4,227	19,942	22,146	56,461	125,110
1902	500	452	1,762	2,478	24,088	21,778	114,610
1903	137	632	4,098	413	16,762	7,959	100,861
1904	62	393	6,456	1,208	15,582	42,662	99,472
1905	84	461	5,431	6,441	20,473	5,663	112,076
1906	15	603	10,866	1,741	18,898	23,168	203,323
1907 (e)	84	631	7,355	(q) 3,167	20,148	5,113	274,175
1908 (r)	1	409	8,596	(q) 3,389	17,835	35,543	515,161

(a) From Tables of the *Trade and Navigation of the Dominion of Canada.* (b) Fiscal year ending June 30. (c) Includes regulus and salts of antimony. (d) Asbestos in any form except crude, and all manufactures of. (e) Includes manufactures. (f) Includes coal dust. (g) Coin, gold and silver, except U. S. silver coin. (h) Includes black lead, and crucibles (clay or graphite). (i) Includes Canadian lead ore refined in the United States. (k) Calendar year. (n) Includes chromic iron ore. (o) Returns for the 9 months of the fiscal year ending March 31. (p) Lead contained in ore, etc. (q) Gallons. (r) Fiscal year ending March 31.

FRANCE.

In the following tables are given the statistics of mineral and metal production in France and the French colonies—Algeria, New Caledonia and Tunis—together with the foreign commerce of France in mineral and metal products:

MINERAL AND METALLURGICAL PRODUCTION OF FRANCE. (a)
(In metric tons.)

Year.	Aluminum.	Antimony.		Arsenic Ore.	Asphaltum.	Barytes.	Bauxite.	Bitumen. (c)	Cement.
		Ore.	Metal.						
1896.....	370	5,675	969	.....	17,717	2,791	33,820	225,784	934,624
1897.....	470	4,685	1,033	.....	17,982	3,209	41,740	233,328	976,813
1898.....	565	4,433	1,226	.....	18,832	2,763	36,723	229,108	1,072,025
1899.....	763	7,392	1,499	2,600	22,100	4,058	48,215	258,449	1,144,271
1900.....	1,026	7,843	1,573	4,705	25,228	3,635	58,530	266,474	1,147,670
1901.....	1,200	9,867	1,786	7,491	20,391	4,145	76,620	249,655	1,127,206
1902.....	1,355	9,715	1,725	5,372	.....	4,223	96,900	258,295	962,930
1903.....	1,570	12,380	2,748	6,658	.....	5,731	133,890	243,295	898,393
1904.....	1,650	9,065	2,116	3,117	22,000	6,944	75,640	227,177	903,632
1905.....	1,905	12,543	2,396	3,627	20,000	5,504	103,207	188,403	922,531
1906.....	3,396	18,567	3,433	6,534	38,231	11,680	117,781	196,375	1,257,861
1907.....	4,700	24,000	3,950	7,900	33,000	11,150	158,000	177,000	1,253,546

Year.	Coal.			Copper.		Gold.	Gypsum.	
	Bituminous.	Lignitic.	Peat.	Ore.	Metal.		Crude.	Calcined.
1896.....	28,750,452	439,448	130,207	106	6,544	$217,308	264,187	1,429,550
1897.....	30,337,207	460,422	98,067	956	7,376	183,416	292,753	1,389,269
1898.....	31,826,127	529,977	104,265	382	7,834	177,435	303,531	1,449,384
1899.....	32,256,148	606,564	99,230	2,021	6,640	179,429	263,879	1,372,067
1900.....	32,721,562	682,736	95,630	3,031	6,446	134,904	192,916	1,405,845
1901.....	31,633,300	691,700	118,433	3,413	7,000	85,727	355,995	1,623,710
1902.....	29,305,047	632,423	109,941	828	6,300	(b)	219,487	1,572,687
1903.....	34,217,661	688,757	100,348	10,892	6,921	(b)	162,766	1,468,830
1904.....	33,502,394	665,572	95,716	2,756	6,900	(b)	106,173	1,481,303
1905.....	35,218,000	709,000	98,500	5,068	7,576	235,147	78,832	1,299,313
1906.....	33,458,000	738,000	92,469	2,547	5,770	511,665	79,568	1,297,861
1907.....	35,989,000	765,000	90,952	2,400	7,800	847,290	87,370	1,316,567

Year.	Iron.				Lead.		Lime.	Manganese Ore.	Millstones.
	Ore.	Pig.	Wrought Iron.	Wrought Steel.	Ore. (d)	Pig. (e)			
1896...	4,069,390	2,339,537	928,758	916,817	19,042	8,232	2,224,847	31,318	28,237
1897...	4,582,236	2,484,191	584,540	994,891	21,212	9,916	2,201,428	37,212	32,175
1898...	4,731,394	2,525,100	700,000	1,174,000	23,342	10,920	2,339,850	31,935	(f) 38,929
1899...	4,985,702	2,578,400	834,000	1,240,000	17,505	15,981	2,343,377	39,897	41,535
1900...	4,676,740	2,714,298	672,172	1,226,537	24,276	15,210	2,377,110	28,992	41,103
1901...	4,260,747	2,398,823	612,362	1,175,454	20,644	21,000	2,443,062	22,304	33,236
1902...	5,003,782	2,405,000	639,600	1,245,900	22,634	19,000	4,796,807	12,536	34,504
1903...	6,219,541	2,840,517	598,910	1,305,709	23,080	23,258	4,727,543	11,583	35,031
1904...	7,022,841	2,999,787	554,632	1,482,708	14,173	18,800	4,583,522	11,254	37,409
1905...	7,395,409	3,077,000	670,000	1,442,000	12,118	24,100	3,694,725	6,751	33,468
1906...	8,481,423	3,314,100	747,900	1,683,500	11,795	25,614	3,869,772	11,189	32,407
1907...	10,008,000	3,590,000	580,000	1,560,000	18,009	24,800	2,438,409	18,200	(b)

Year.	Mineral Paints (Ochers).	Nickel.	Phosphate Rock.	Pyrites.	Salt.	Silver.	Sulphur Ore. (g)	Zinc.	
								Ore.	Metal.
1896....	27,499	1,545	582,667	282,064	1,042,614	.....	9,720	81,346	35,585
1897....	32,299	1,245	535,390	303,488	948,003	.....	10,723	83,044	38,067
1898....	33,780	1,540	568,558	310,972	999,283	.....	9,818	85,550	37,155
1899....	32,750	1,740	645,866	318,832	1,193,532	.....	11,744	84,813	39,274
1900....	33,080	1,700	587,919	305,073	1,088,634	.....	11,551	67,059	36,305
1901....	35,704	1,800	535,676	307,447	910,000	.....	7,000	61,539	37,600
1902....	34,770	1,600	543,900	318,235	863,927	.....	8,021	57,982	36,300
1903....	34,042	1,500	475,783	322,118	967,531	.....	7,375	66,922	37,416
1904....	34,945	1,500	423,521	271,544	1,153,754	.....	5,447	52,842	41,600
1905....	37,800	1,800	476,720	267,114	1,130,088	1,631,099	4,637	62,150	43,200
1906....	35,550	1,750	469,408	265,261	1,335,420	1,176,852	2,713	53,466	46,536
1907....	32,856	1,500	432,237	283,000	1,226,000	1,661,197	2,000	44,000	47,900

(a) From *Statistique de l'Industrie Minérale*. (b) Not reported. (c) Includes pure bitumen, bituminous schist and sand, and asphaltic limestone. (d) Argentiferous lead ore. (e) Lead produced from native ores only. (f) Finished product. (g) Sulphur and limestone impregnated with sulphur.

MINERAL PRODUCTION OF ALGERIA. (a)
(In metric tons.)

Year.	Anti-mony Ore.	Copper Ore.	Gypsum.		Iron Ore.	Lead-silver Ore.	Mercury.	Onyx.	Phos-phate Rock.	Salt.	Zinc Ore.
			Crude.	Plaster.							
1896..	658	427	300	29,870	374,476	117	.....	900	165,738	19,658	17,587
1897..	781	289	350	29,120	441,467	145	.....	364	228,141	23,222	32,269
1898..	138	488	150	29,750	473,569	120	.....	219	269,500	21,300	29,800
1899..	200	472	200	31,900	550,921	389	.....	217	324,983	17,378	42,970
1900..	93	.....	500	37,100	174,000	222	.....	228	319,422	19,325	30,281
1901..	.....	7,267	600	34,740	161,303	1,614	.....	294	265,000	18,518	26,913
1902..	39	1,955	600	35,500	525,012	26	.....	150	305,174	27,263	33,139
1903..	490	100	300	33,000	588,893	499	.....	67	320,834	26,329	43,313
1904..	160	1,804	350	38,420	468,737	511	.....	121	343,317	18,563	47,192
1905..	.....	1,784	.....	34,743	568,609	7,470	.....	270	334,784	26,986	67,922
1906..	50	2,786	.....	27,950	779,826	11,246	.....	216	333,531	22,615	74,351
1907..	799	16,259	*Nil.*	26,400	973,445	15,264	590	328	373,763	20,390	71,048

(a) From *Statistique de l'Industrie Minérale*.

MINERAL PRODUCTION OF NEW CALEDONIA. (a)
(In metric tons.)

Year.	Chrome Iron Ore.	Cobalt Ore.	Copper Ore	Nickel Ore.	Year.	Chrome Iron Ore.	Cobalt Ore	Copper Ore.	Nickel Ore.
1897..	3,949	3,200	2,200	26,464	1903...	21,437	8,292	10	77,360
1898..	7,712	2,373	*Nil.*	74,614	1904...	42,197	8,964	*Nil.*	98,655
1899..	12,634	3,294	6,349	103,908	1905...	51,374	7,920	*Nil.*	125,289
1900..	10,474	2,438	2	100,319	1906...	84,241	2,600	207	118,890
1901..	17,451	3,123	6,349	132,814	1907...	31,552	29,800	437	120,106
1902..	10,281	7,512	3,720	129,653	1908 (b)	46,309	3,404	10	120,028

(a) From *Statistique de l'Industrie Minérale*. (b) From *Le Bulletin du Commerce*, Nouméa.

MINERAL PRODUCTION OF TUNIS. (a)
(In metric tons.)

Year.	Salt.	Lead Ore.	Phosphate of Lime.	Zinc Ore.	Year.	Salt.	Lead Ore.	Phosphate of Lime.	Zinc Ore.
1896	5,500	(b)	1,000	12,100	1902	21,600	12,892	264,930	18,400
1897	8,100	2,123	(b)	11,830	1903	18,846	12,752	352,088	21,262
1898	7,300	2,375	(b)	21,477	1904	23,600	16,800	455,197	27,200
1899	8,350	2,263	70,000	20,079	1905	52,900	15,200	522,000	37,100
1900	9,160	6,864	178,000	16,596	1906	62,600	14,800	796,000	32,400
1901	16,900	8,158	172,000	17,879	1907	78,200	18,600	1,069,000	22,900

(a) From *Statistique de l'Industrie Minérale*. (b) Not reported.

MINERAL IMPORTS OF FRANCE. (a)

(In metric tons or dollars. 5 f. = $1.)

Year.	Alum.	Bitumen. (f)	Borax.	Bromides.	Cement.	Coal and Coke.	Copper. Ore.	Copper. Ingot and Mfres.	Copper. Sulphate.	Copper. Oxide.
1895....	199	43,975	442	12	13,441	10,261,069	10,450	38,196	24,404	24
1896....	41	30,954	255	13	14,395	10,180,449	8,584	46,830	33,803	22
1897....	54	29,931	264	18	15,141	10,457,255	11,960	54,460	30,132	29
1898....	27	20,385	139	30	11,290	10,445,090	8,779	52,976	30,897	52
1899....	34	30,770	123	46	13,640	11,896,030	8,517	58,419	21,733	36
1900....	23	39,598	111	10	13,612	14,601,981	9,766	61,638	22,820	84
1901....	39	28,888	128	3	16,232	13,925,623	13,383	47,035	15,313	162
1902....	36	26,053	141	3	15,720	13,137,720	17,362	54,484	22,273	111
1903....	138	27,573	312	9	21,152	14,029,687	9,796	59,126	25,428	129
1904....	370	17,178	3,113	17	21,702	13,936,475	9,942	09,183	30,856	142
1905....	63	24,606	1,736	31	21,954	13,910,523	14,252	70,101	23,805	57
1906....	105	99,336	189	93	24,974	17,848,284	11,932	64,590	15,358	97
1907....	.....	31,700	.....	.....	24,839	18,706,000	12,063	76,282	.....	.....
1908....	.....	.....	.....	.....	.....	.....	15,284(*k*)	74,425(*k*)	.....	.....

Year.	Cobalt Oxide.	Iron. Ore.	Iron. Pig.	Iron. Iron and Steel, Mfres. of.	Iron. Sulphate.	Iron. Oxide.	Kaolin.	Lead. Ore.	Lead. Carbonate.	Lead. Pig, Scrap and Mfres.
1895....	5	1,651,369	36,247	66,240	3,882	855	.....	5,032	1,077	66,241
1896....	5	1,362,043	18,323	48,423	3,086	897	38,703	5,569	892	79,752
1897....	9	2,137,860	35,633	60,804	1,353	1,125	42,384	13,981	1,327	86,589
1898....	9	2,032,240	(b)	47,325	896	1,021	40,352	14,377	1,376	74,902
1899....	9	1,950,665	(b)	64,178	1,698	1,037	36,904	12,637	2,029	67,149
1900....	9	2,119,003	149,755	118,152	1,589	1,022	39,842	19,772	1,739	70,857
1901....	8	1,662,875	61,085	77,742	45	1,001	41,972	15,430	1,789	59,051
1902....	10	1,563,334	38,521	60,697	17	1,051	41,165	13,121	2,223	58,694
1903....	11	1,832,820	121,726	119,799	36	1,207	47,534	20,172	2,040	75,416
1904....	69	1,738,514	135,252	125,709	319	1,151	50,465	25,731	2,221	76,198
1905....	35	2,151,954	122,102	150,480	709	1,330	52,603	35,103	2,306	73,938
1906....	41	2,015,550	156,618	342,411	132	1,311	44,772	43,137	2,072	67,651
1907....	.....	1,999,000	154,031	617,123	.....	.....	42,342(*k*)	42,342	.....	53,359
1908....	.....	.....	.....	.....	.....	.....	40,722(*k*)	.....	.....	70,399

Year.	Lime. Chloride of.	Manganese Ore.	Nickel. Ore.	Nickel. Metal.	Petroleum.	Phosphate Rock.	Platinum. *Kg.*	Potassium. Chloride.	Potassium. Chromate (*h*)
1895....	1,047	41,400	10,303	252	258,700	139,600	926	3,524	2,875
1896....	2,033	61,600	15,756	425	272,093	256,388	2,117	11,499	2,538
1897....	1,713	85,500	17,441	316	288,671	313,608	1,069	11,630	2,352
1898....	1,288	100,243	24,935	330	291,961	336,842	505	10,929	2,890
1899....	1,887	106,630	28,620	286	306,078	242,021	817	13,335	3,147
1900....	1,215	120,790	17,687	299	302,482	283,921	2,398	13,524	3,293
1901....	1,400	94,365	30,497	252	225,962	275,285	1,857	13,299	2,784
1902....	2,130	85,629	58,374	301	148,170	302,898	2,940	10,802	2,861
1903....	919	109,930	13,933	427	(*g*)476,230	343,012	3,764	12,275	2,760
1904....	1,679	105,652	20,698	313	(*g*)435,730	419,720	5,650	14,734	2,618
1905....	406	140,871	49,698	632	(*g*)512,727	447,738	4,023	21,819	2,619
1906....	593	127,235	44,960	480	(*g*)213,462	533,213	5,708	26,523	3,024
1907....	.....	192,448	45,892	979	(*g*)311,000	636,549(*l*)	4,373	.....	.....
1908....	.....	170,457(*k*)	42,153(*k*)	.....	.....	.....	.....	.....	.....

Year.	Potassium. (Cont'd)		Pyrites.	Quicksilver.		Sal-Ammoniac.	Salt.	Sodium.	
	Nitrate.	Carbonate.		Ore.	Metal.			Hydrate.	Nitrate.
1895	775	796	67,930	23	178	9,923	17,528	1,021	8,624,200
1896	2,614	1,526	45,788	25	234	15,256	17,191	1,109	9,025,400
1897	1,309	1,769	69,470	24	248	27,454	32,917	1,378	8,105,400
1898	1,008	2,418	71,569	19	221	20,426	35,863	1,772	8,026,400
1899	1,015	2,779	109,696	21	276	12,210	37,970	1,494	9,341,600
1900	1,928	2,768	156,825	22	161	15,205	32,045	1,062	11,995,820
1901	757	2,520	205,617	23	205	9,268	32,347	869	10,526,400
1902	1,547	1,539	170,783	24	224	15,446	32,505	643	9,372,600
1903	1,530	3,019	205,322	20	220	12,462	48,556	781	10,810,775
1904	2,117	3,781	230,097	22	208	13,744	46,232	1,068	9,074,859
1905	1,022	3,542	271,684	.....	228	11,639	45,241	860	11,336,752
1906	684	2,206	349,514	.....	242	18,146	38,361	614	13,678,848
1907	.....	.....	355,300	.....	216	.....	30,000	.....	.....
1908	.....	.....	.....	.....	.....	.....	.....	.....	.....

Year.	Sulphur.	Sulphuric Acid.	Superphosphate of Lime.	Tin.		Zinc.	
				Ore.	Metal.	Ore.	Metal.
1895	110,989	3,461	150,758	104	7,691	41,622	25,652
1896	111,515	3,995	185,602	7	8,400	50,909	33,450
1897	136,118	3,147	195,853	149	7,642	58,074	31,211
1898	130,289	4,666	178,560	357	9,247	60,481	32,342
1899	120,062	4,583	171,631	486	6,907	78,192	25,516
1900	133,531	4,254	143,437	512	7,324	66,178	33,144
1901	101,301	5,286	165,261	365	7,314	74,553	29,812
1902	85,339	7,793	116,093	748	8,575	69,451	36,564
1903	109,594	13,241	89,229	1,808	9,573	67,258	39,305
1904	148,547	11,212	72,921	1,344	9,352	88,083	35,737
1905	129,877	10,915	81,729	1,362	9,898	105,069	29,163
1906	131,678	5,268	44,502	1,038	7,687	106,307	26,960
1907	106,050	.....	.....	961	7,693	114,699	33,503
1908	.....	.....	.....	.....	8,027(k)	137,789(k)	37,206(k)

MINERAL AND METALLURGICAL EXPORTS OF FRANCE. (a)

(In metric tons.)

Year.	Aluminum.	Antimony.		Arsenic.	Cement.	Coal.	Copper		Gold. Kg. (d)
		Ore.	Metal.				Ore. (c)	Metal.	
1895	110	832	68	.....	(b)	(b)	1.772	8,829	1,353
1896	793	736	74	.....	242,247	1,044,820	1,261	10,494	2,193
1897	224	623	61	.....	244,504	1,142,195	2,000	12,667	3,235
1898	192	616	101	.....	241,150	1,320,616	1,783	14,350	1,812
1899	256	304	255	.....	244,480	1,220,000	2,078	17,949	2,622
1900	324	154	336	.....	232,577	1,201,210	9,197	16,791	883
1901	307	645	741	.....	242,010	908,583	16,006	14,776	1,869
1902	748	595	686	.....	210,590	910,700	20,489	14,423	1,517
1903	666	904	1,358	.....	233,835	2,238,735	12,487	11,403	3,139
1904	664	1,191	720	.....	260,686	2,384,928	14,238	12,663	1,537
1905	9 8	98'	8 5	...	75.5 3	(f) 3,348 0 0	13, 63	3,890	5 749
1906	1 522	3 5 1	8 1	.....	323,879	(f) 1, 8 3 0	8 5	5,1	11,727
1907	1 113	3,0 0	1 27	.....	306,624	(f) 1,234,0 0	1 15	(k) 4 6	6 289
1908 (k)	...	3,233	1 968	8,56	.....	.....	3.562	5,2 8	...

Year.	Iron.				Lead.		Manganese Ore.
	Ore.	Pig.	Bars.	Steel.	Ore.	Metal.	
1895	236,923	150,540	.....	29,074	8,670	8,037	16,193
1896	238,430	195,212	24,721	44,795	8,597	10,856	10,913
1897	299,589	108,645	39,894	45,809	12,007	10,364	19,464
1898	236,169	162,991	27,424	47,278	10,216	8,663	12,229
1899	291,346	153,792	29,112	33,584	3,909	1,163	12,289
1900	371,799	114,361	18,763	19,535	2,345	958	8,392
1901	253,925	96,463	25,220	56,347	3,490	718	5,289
1902	422,677	213,081	23,828	121,932	2,414	648	1,948
1903	714,173	196,444	40,533	215,737	2,313	13,048	717
1904	1,219,149	191,819	40,374	246,738	1,860	13,467	1,392
1905	1,355,932	218,227	67,240	343,612	3,064	12,903	662
1906	1,759,443	143,142	58,826	236,617	1,354	997	4,103
1907	2,147,000	249,708	84,557	291,434	1,210	1,912	5,167
1908	.....	.....	.....	.....	6,727	1,344	.....

Year.	Nickel Refined.	Phosphate Rock.	Plaster.	Pyrites.	Silver. *Kg.* (*a*)	Tin (Metal).	Zinc.	
							Ore.	Spelter, Sheets and Scrap.
1895	408	.....	.....	37,968	13,567	660	61,291	5,849
1896	490	48,719	89,952	44,232	9,849	744	62,415	10,485
1897	498	69,188	107,823	54,867	5,274	651	79,909	10,977
1898	526	93,742	106,790	60,406	1,886	587	60,664	16,995
1899	280	70,517	112,520	53,395	.....	666	76,104	14,958
1900	599	89,135	108,357	64,530	15,470	716	54,663	12,712
1901	1,031	81,405	101,063	52,952	16,745	438	42,995	15,022
1902	397	62,375	110,270	63,920	17,184	654	47,724	16,158
1903	720	73,252	131,245	119,173	43,690	1,994	62,731	12,657
1904	906	78,612	139,551	40,833	23,105	2,200	57,780	19,063
1905	1,583	55,240	124,561	21,257	66,904	2,611	72,512	17,802
1906	1,088	81,660	142,339	26,216	87,952	601	67,258	19,607
1907	1,414	100,508	137,356	24,417	58,199	729	54,316	21,928
1908	1,045	.....	.....	.....	.....	463	57,758	20,064

(*a*) From *L'Economiste Français* (representing the *Commerce Spécial*) except for 1903-06, inclusive, which are from *Tableau Général du Commerce et de la Navigation*. (*b*) Not reported. (*c*) Includes matte. (*d*) Gold and platinum in ore, sheets, leaves or threads. (*e*) Silver in ore, sheets, leaves, wire, etc. (*f*) Includes bitumen, bituminous schist and sands and asphaltic limestone. (*g*) Crude and refined. Transposition from hectoliters to tons was performed by assuming specific gravity of petroleum to be 0.9. (*h*) Includes chromate of soda. (*i*) Includes coke. (*k*) From *L'écho des Mines et de la Métallurgie*.

. GERMANY.

The mineral production and foreign commerce of the German Empire are given in the following tables in metric tons unless otherwise specified, or in dollars, on the basis of four marks to the dollar.

MINERAL PRODUCTION OF GERMANY. (a)

Year.	Alum.	Aluminum Sulphate.	Arsenic.		Asphaltum.	Boracite	Cadmium. Kg.	Coal.	
			Ore.	Salts.				Bituminous.	Lignitic.
1897.....	2,995	37,053	3,777	2,989	61,645	198	15,531	91,054,982	29,419,503
1898.....	4,069	35,366	3,527	2,679	67,649	230	14,943	96,309,652	31,648,898
1899.....	3,358	37,693	3,834	2,423	74,770	183	13,608	101,639,753	34,204,666
1900.....	4,355	44,372	4,379	2,415	89,685	232	13,553	109,290,237	40,498,019
1901.....	4,145	46,807	4,035	2,549	90,193	184	13,144	108,539,444	44,479,970
1902.....	4,108	47,905	3,959	2,828	88,374	196	.....	107,473,933	43,126,281
1903.....	3,934	49,727	4,369	2,768	87,454	159	16,565	116,637,765	45,819,488
1904.....	3,850	55,881	4,390	2,829	91,736	135	25,245	120,815,503	48,635,080
1905.....	4,127	52,892	4,913	2,535	103,006	183	24,568	121,298,607	52,512,002
1906.....	4,494	55,969	6,249	3,052	117,413	161	.....	137,117,926	56,419,567
1907.....	4,200	59,473	4,878	2,904	126,649	114	32,949	143,185,691	62,546,671
1908.....	3,802	3,348	.....	.....	89,009	.....	.....	148,537,417	66,746,057

Year.	Cobalt, Nickel and Bismuth Ores.	Copper.				Gold.	Graphite.
		Ore.	Matte. (b)	Ingots.	Sulphate.		
1897.....	3,355	700,619	315	29,408	5,549	$1,848,114	3,861
1898.....	3,157	702,781	62	30,695	4,352	1,891,974	4,593
1899.....	1,270	733,619	95	34,634	5,142	1,731,153	5,196
1900.....	4,495	747,749	4,207	30,929	5,076	2,030,200	9,248
1901.....	10,479	777,339	365	31,317	5,192	1,830,835	4,435
1902.....	12,433	761,921	447	30,578	4,997	1,770,361	5,023
1903.....	14,607	772,695	583	31,214	5,200	1,709,223	3,720
1904.....	14,016	798,214	641	30,264	6,584	1,819,538	3,784
1905.....	10,848	793,488	1,635	31,713	6,988	2,611,812	4,921
1906.....	.....	768,523	771	32,275	6,757	2,931,750	4,055
1907.....	2,809	771,227	527	31,946	5,284	3,111,379	4,033
1908.....	6,081	727,384	.....	30,001	.....	3,161,884	4,844

Year.	Iron and Steel.					Lead.		
	Iron Ore.	Pig Iron. (c)	Castings.	Steel.	Sulphate. (d)	Ore.	Pig.	Litharge
1897.....	15,465,980	6,881,466	1,473,211	6,248,141	10,351	150,178	118,881	3,441
1898.....	15,901,263	7,312,766	1,597,434	6,941,278	10,422	149,311	132,742	3,857
1899.....	17,989,635	8,153,133	1,776,878	7,532,524	10,931	144,370	129,225	3,562
1900.....	18,964,294	8,520,540	1,812,603	7,377,275	10,913	148,257	121,513	3,088
1901.....	16,570,182	7,880,087	1,520,617	7,033,438	11,148	153,341	123,098	4,101
1902.....	17,963,591	8,529,900	1,575,525	8,317,231	.....	167,855	140,331	4,197
1903.....	21,230,650	10,017,901	1,721,781	9,226,898	12,243	165,991	145,319	4,428
1904.....	22,047,393	10,058,273	1,879,879	9,239,302	13,585	164,440	137,580	4,332
1905.....	23,444,073	10,875,061	2,045,477	10,309,000	12,949	152,725	152,590	3,786
1906.....	26,734,570	12,292,819	.....	11,307,807	13,376	140,914	150,741	4,137
1907.....	27,697,127	12,875,159	.....	12,063,632	14,033	147,272	142,271	4,325
1908.....	24,224,762	11,805,321	.....	11,186,379	.....	156,842	164,079	5,339

Year.	Magnesium Salts.		Manganese Ore.	Nickel. (e).	Petroleum.	Potassium Salts.				
	Chloride.	Sulphate.				Chloride.	Kainite. (f)	Sulphate.	Potassium and Magnesium Sulphate.	Unspecified.
1897.....	18,014	35,072	46,427	1,464	23,303	168,001	992,389	13,774	7,812	953,798
1898.....	19,819	30,295	43,354	1,691	25,989	191,347	1,103,643	18,853	13,982	1,105,212
1899.....	21,370	39,540	61,329	1,747	27,027	207,506	1,108,159	26,103	9,765	1,384,972
1900.....	19,397	48,591	59,204	1,989	50,375	271,512	1,227,373	30,853	15,363	1,822,758
1901.....	21,018	46,714	56,691	2,207	44,095	294,666	1,498,569	37,394	15,612	2,036,325
1902.....	19,658	39,262	49,812	2,196	49,725	267,512	1,322,623	28,278	18,147	1,962,384
1903.....	22,990	37,344	47,994	2,637	62,680	280,248	1,557,243	36,674	23,631	2,073,720
1904.....	25,730	39,412	52,886	3,063	89,620	297,238	1,905,893	43,959	29,285	2,179,471
1905.....	29,017	58,568	51,463	3,317	78,869	373,177	2,387,643	47,994	34,222	2,655,845
1906.....	38,468	43,041	52,485	.....	81,350	403,387	2,720,594	54,490	35,211	2,821,073
1907.....	32,891	41,105	73,105	.....	106,379	473,138	2,624,412	60,292	33,368	3,124,955
1908.....	29,775	42,976	67,693	.....	141,990	508,622	2,589,804	55,755	33,149	3,500,635

Year	Pyrites.	Salt.		Silver and Gold Ore.	Silver. *Kg.*	Sodium Sulphate.	Sulphur.	Sulphuric Acid.
		Rock.	Evaporated.					
1897.....	133,302	763,412	543,272	9,708	448,068	68,822	2,317	702,445
1898.....	136,849	807,792	565,683	14,702	480,578	69,111	1,954	754,151
1899.....	144,623	861,123	571,058	13,506	467,590	79,062	1,663	813,141
1900.....	169,447	926,563	587,464	12,593	415,735	90,468	1,445	829,376
1901.....	157,433	985,050	578,751	11,577	403,796	76,066	963	835,000
1902.....	165,225	1,010,412	572,846	11,724	430,610	90,742	.....	894,409
1903.....	170,867	1,095,541	598,394	11,467	396,253	83,087	219	928,190
1904.....	174,782	1,079,868	621,064	10,405	389,827	75,171	209	963,384
1905.....	185,368	1,165,495	612,062	10,286	399,775	68,454	205	1,228,211
1906.....	196,971	1,235,041	635,171	8,066	393,442	81,175	178	1,325,128
1907.....	196,351	1,285,137	665,547	8,280	386,933	80,347	176	1,402,398
1908.....	219,455	1,331,984	665,413	7,654	407,833	72,667	.....	1,385,740(*h*)

Year.	Tin.			Uranium and Tungsten Ores.	Zinc.		
	Ore.	Block.	Chloride.		Ore.	Spelter.	Sulphate.
1897.....	55	929	.....	38	663,850	150,739	5,488
1898.....	51	993	.....	50	641,706	154,867	6,104
1899.....	72	1,481	.....	50	664,536	153,155	7,117
1900.....	80	2,031	(*g*) 143	43	639,215	155,790	6,027
1901.....	82	1,464	(*g*) 135	43	647,496	166,283	5,552
1902.....	104	2,779	.....	31	702,504	174,927	.....
1903.....	110	3,065	1,064	35	682,853	182,548	5,994
1904.....	99	4,216	816	23	715,732	193,058	6,185
1905.....	123	5,233	811	26	731,271	198,208	5,896
1906.....	.....	6,597	987	.....	704,590	205,691	6,092
1907.....	.....	5,838	1,812	3	698,425	208,195	5,145
1908.....	.....	.....	.....	.....	706,441	216,491	.....

(a) From the *Vierteljahrshefte zur Statistik des Deutschen Reichs*. (b) Includes black copper. (c) Includes ferromanganese and spiegeleisen. (d) Contains a small quantity of copper and iron sulphate mixed. (e) Includes nickeliferous by-products, metallic bismuth, and uranium compounds. (f) Compound of potassium chloride and magnesium sulphate. (g) Includes nickel sulphate. (h) Distilled 128,522; undistilled, 1,257,218.

MINERAL PRODUCTION OF BADEN. (a)

(In metric tons and dollars; 4 marks = $1.)

Year.	Alumina Sulphate.	Barytes.	Coal.	Gypsum.	Iron.		
					Cast, Foundry.	Pig.	Wrought.
1896	1,824	130	4,001	32,801	31,356	3,418	1,118
1897	1,824	400	4,752	40,702	36,235	3,875	1,167
1898	2,051	1,100	4,133	28,037	39,988	3,875	1,167
1899	2,153	2,430	4,700	29,419	53,608	3,830	1,402
1900	2,286	2,970	4,930	26,381	50,102	3,532	1,364
1901	2,260	3 991	3,650	28,183	40,100	8,739	1,158
1902	2,374	6,234	2,078	33,150	40,973	12,663	1,052
1903	2,498	8,857	1,990	29,423	45,233	7,666	863
1904	2,392	9,078	1,485	26,984	64,320	7,667	783
1905	2,581	11,094	668	28,823	74,128	8,053	842
1906	2,583	11,984	1,000	25,643	81,387	11,063	466
1907	2,644	9,303	2,075	29,153	98,430	10,818	533
1908	(b)	(c)8,554	2,473	35,217	(b)	(b)	(b)

Year.	Lead Ores.	Salt.	Sulphuric Acid.	Tripoli.	Zinc Ore.
1896	(b)	29,227	14,226	9	(b)
1897	(b)	31,445	13,365	9	(b)
1898	(b)	31,445	13,365	6	(b)
1899	(b)	31,197	13,660	12	357
1900	67	32,699	15,938	9	3,004
1901	369	32,835	17,081	8	2,870
1902	450	32,192	19,265	11	2,958
1903	350	32,383	19,755	11	3,171
1904	265	32,148	35,517	12	5,063
1905	264	31,393	40,781	12	4,046
1906	246	31,288	38,655	15	1,466
1907	278	32,078	42,831	25	2,198
1908	329	33,993	(b)	13	2,793

(a) From the *Uebersicht der Production des Bergwerks-, Hütten-, und Salinen-Betriebes in dem Bayerischen Staate.* (b) Not reported. (c) Includes fluorspar.

MINERAL PRODUCTION OF BAVARIA. (a)

(In metric tons; 4 marks = $1.)

Year.	Barytes.	Kaolin.	Coal.	Coal. (Lignite).	Copperas and other Sulphate.	Emery.	Feldspar.	Fluorspar.	Graphite.
1896	3,397	19,080	900,080	35,934	601	249	1,315	5,218	5,248
1897	3,365	24,086	917,022	39,043	981	217	1,689	4,904	3,561
1898	4,339	29,196	964,611	38,663	886	280	1,949	4,440	4,593
1899	6,214	25,822	1,004,421	35,736	900	399	287	3,631	5,196
1900	10,515	58,795	1,185,296	39,165	916	414	460	7,456	9,248
1901	8,711	35,450	1,203,702	25,224	590	366	788	5,220	4,435
1902	8,034	92,073	1,233,568	27,337	691	225	447	5,460	5,023
1903	8,642	88,140	1,356,556	25,189	814	220	1,060	3,410	3,719
1904	9,411	95,160	1,341,925	53,517	893	265	1,866	4,770	3,784
1905	10,030	99,910	1,317,951	154,128	844	255	1,710	4,413	4,921
1906	19,817	98,138	1,381,175	140,290	836	330	1,740	5,570	4,055
1907	21,500	115,387	1,495,895	286,256	850	326	2,125	4,780	4,033

Year.	Gypsum.	Iron.						Marl. (For Cement).	Mineral Paints.
		Ore.	Bar.	Cast, 1st Fusion	Cast, 2d Fusion	Pig.	Steel.		
1896	28,799	161,279	53,573	114	71,006	79,621	101,954	94,481	8,667
1897	26,153	172,699	58,200	138	78,008	83,418	115,530	97,531	8,673
1898	25,688	171,987	58,342	97	84,227	84,144	120,623	110,757	8,748
1899	29,727	181,981	61,415	(b)	92,459	83,821	134,007	220,716	9,287
1900	35,484	178,441	49,727	29	89,692	82,327	135,411	180,032	11,507
1901	3,581	158,820	29,978	76	76,191	72,071	109,464	76,663	84,929
1902	31,701	157,375	38,429	56	81,874	83,123	115,354	178,301	13,947
1903	30,894	162,500	36,853	41	89,804	90,168	127,141	200,407	19,486
1904	22,766	180,342	37,780	40	108,025	92,200	125,483	170,698	19,107
1905	46,247	182,389	36,459	24	112,875	94,242	134,755	231,310	18,285
1906	50,763	203,596	38,508	.....	122,115	97,812	150,129	230,271	22,304
1907	48,975	277,280	36,883	.....	138,659	98,143	150,148	230,583	21,219

Year.	Pyrites.	Rock Salt.	Salt, Brine.	Soapstone.	Sodium Sulphate.	Stone.	Sulphuric Acid.
						Lithographic.	
1896	1,997	708	40,400	3,051	663	10,868	7,064
1897	2,211	1,161	41,533	2,464	2,318	13,941	7,041
1898	2,304	736	39,717	1,912	2,332	12,029	103,385
1899	2,516	802	41,207	2,197	1,570	11,962	123,273
1900	2,120	1,298	44,431	1,977	1,821	16,030	123,910
1901	2,649	1,319	41,217	2,291	1,893	9,500	115,775
1902	2,635	832	.....	.....	.....	.....	.....
1903	2,324	879	41,782	1,866	.....	9,890	.....
1904	3,427	1,139	43,048	1,709	.....	13,836	.....
1905	3,301	911	42,591	1,872	.....	11,360	.....
1906	3,918	1,053	.....	.....	.....	.....	.....
1907	5,085	1,393	.....	.....	.....	.....	.....

(a) From the *Uebersicht der Production des Bergwerks-, Hütten-, und Salinen-Betriebes in dem Bayerischen Staate.* (b) Not reported.

MINERAL PRODUCTION OF PRUSSIA. (a)

(Metric tons; 4 marks=$1.)

Year.	Alum Shale.	Antimony and Alloys.	Arsenic Products.	Arsenic Ore.	Asphalt.	Boracite.	Cadmium. *Kg.*	Coal.
1897	129	1,552	1,924	3,377	11,466	185	15,531	84,253,393
1898	107	2,612	1,624	3,298	12,822	216	14,943	89,593,528
1899	145	3,003	1,469	3,265	16,458	171	13,608	94,740,829
1900	103	3,025	1,585	3,531	23,891	217	13,533	101,966,158
1901	611	2,404	1,446	3,050	26,450	164	13,144	101,203,807
1902	219	3,542	1,514	2,909	28,035	172	12,625	102,115,315
1903	580	3,224	1,583	3,538	23,518	135	16,565	108,809,384
1904	106	2,774	1,573	3,527	26,348	115	25,245	112,755,621
1905	97	2,795	1,493	4,022	28,872	151	24,568	113,000,657
1906	634	2,953	1,551	5,430	32,270	124	21,486	123,295,948
1907	154	3,515	1,591	4,224	39,243	90	32,949	134,044,080

Year.	Coal. (Lignite.)	Cobalt Ore.	Cobalt Products.	Copper.	Copper and Iron Sulphate.	Copper Ore.	Copper Matte.
1897	24,222,911	121	51	25,997	225	690,338	274
1898	26,035,814	34	44	27,216	120	691,866	62
1899	28,418,598	17	46	20,902	154	722,884	95
1900	34,007,542	4	52	27,156	113	747,901	4,207
1901	37,491,412	36	66	28,422	78	765,241	281
1902	36,228,285	76	74	27,893	119	751,496	346
1903	38,462,766	65	87	28,386	110	761,188	488
1904	41,153,576	41	85	27,450	95	782,049	601
1905	44,148,751	22	99	28,874	102	769,381	1,052
1906	47,912,721	7	98	29,166	94	755,812	525
1907	52,660,597	*Nil.*	109	28,945	64	755,203	499

Year.	Copper Sulphate.	Epsom Salt.	Gold. *Kg.*	Iron.	Iron Ore.	Iron Sulphate.	Lead.
1897	2,689	2,248	1,087.1	4,892,059	4,183,536	9,064	108,880
1898	1,701	2,061	1,036.3	5,176,943	4,020,809	9,144	119,346
1899	1,586	1,793	1,016.4	5,644,614	4,295,575	10,186	116,995
1900	1,660	1,511	1,076.6	5,781,892	4,268,069	10,225	112,738
1901	1,951	1,952	1,087.1	5,315,628	3,831,670	10,239	113,939
1902	1,937	761	1,138.0	5,633,089	3,362,887	11,214	127,283
1903	2,254	421	949.5	6,614,768	3,786,743	11,066	133,405
1904	3,364	289	1,061.9	6,573,507	3,757,651	12,524	128,294
1905	3,065	338	1,034.9	7,106,975	4,130,210	12,075	143,270
1906	2,724	144	750.2	8,154,880	4,713,928	12,473	140,690
1907	2,129	263	771.0	8,626,300	5,077,773	13,014	133,366

Year.	Lead Ore.	Litharge.	Manganese Ore.	Nickel.	Nickel Ore.	Nickel Sulphate.	Ocher and Mineral Paints.
1897	133,158	1,999	45,254	898	204	167	2,400
1898	133,637	2,360	42,232	1,108	79	127	2,376
1899	128,942	2,482	60,379	1,115	91	123	2,770
1900	133,483	2,366	58,016	1,376	3,896	115	2,850
1901	139,285	2,885	55,866	1,660	9,922	120	2,800
1902	152,282	2,516	48,882	1,605	11,816	159	2,780
1903	151,746	2,710	47,110	1,945	14,058	173	2,650
1904	150,328	2,517	52,092	2,333	13,518	207	3,200
1905	138,928	2,272	51,048	2,631	10,432	220	3,170
1906	127,322	2,744	51,881	2,648	7,472	187	3,635
1907	133,528	2,959	72,442	2,093	7,557	189	3,707

Year.	Petroleum.	Potassium Salts.		Pyrites.	Quicksilver. *Kg.*	Salt.	
		Kainite.	All Other.			Common.	Rock.
1897	2,600	716,348	640,236	121,766	4,867	274,888	310,755
1898	2,545	744,240	718,957	128,077	4,717	286,051	329,959
1899	3,405	744,657	941,055	134,564	2,611	288,588	331,943
1900	27,731	857,271	1,264,993	159,186	1,711	287,005	354,603
1901	24,098	1,068,237	1,131,703	148,457	1,713	290,869	353,557
1902	29,520	943,450	1,344,541	155,410	1,828	291,296	359,006
1903	41,733	1,118,270	1,344,038	159,234	2,145	317,475	409,199
1904	67,604	1,261,930	1,447,323	163,209	3,030	328,933	394,910
1905	57,741	1,580,530	1,734,033	174,641	2,597	328,051	436,942
1906	59,196	1,923,088	1,937,181	186,849	5,084	339,675	492,339
1907	80,255	1,839,409	2,070,978	184,962	5,080	353,290	480,563

Year.	Silver *Kg.*	Silver and Gold Ores.	Sulphur.	Sulphuric Acid.	Tin.	Zinc.		
						Ore.	Metal.	Sulphate.
1897	289,960	6	2,091	484,289	912	663,739	150,739	3,583
1898	291,969	43	1,757	531,838	979	641,671	154,643	4,158
1899	293,858	7	1,419	573,733	1,461	663,763	152,987	4,864
1900	266,577	1	1,207	593,109	2,010	636,068	155,760	3,742
1901	246,286	6	772	609,041	1,443	644,504	166,223	3,369
1902	273,901	17	250	677,798	2,753	699,392	174,892	3,381
1903	255,722	13	16	724,784	3,042	679,320	182,472	3,586
1904	252,020	8	16	868,424	4,193	710,599	192,903	3,696
1905	266,072	4	14	921,219	5,196	727,104	198,179	3,506
1906	264,427	239	16	980,188	6,570	702,933	205,632	3,630
1907	249,348	34	7	1,004,599	5,819	696,039	207,849	3,057

(*a*) From *Zeitschrift für das Berg, Hütten, und Salinenwesen.*

MINERAL IMPORTS OF GERMANY. (a)

Year.	Aluminum, Refined and Crude.	Ammonium Sulphate.	Antimony.	Antimony and Arsenic Ores.	Asbestos, Crude.	Asphalt.	Bituminous Rock.	Barium Chloride.	Barytes. (b)
1897	.....	33,113	.....	.....	.....	.....	.....	.....	.....
1898	.....	30,254	.....	.....	.....	.....	.....	.....	.....
1899	.....	28,868	.....	.....	.....	61,534	.....	.....	.....
1900	943	23,105	1,461	1,291	6,850	80,765	48,986	3,062	7,282
1901	1,090	44,408	1,494	1,098	5,500	62,299	41,733	1,768	5,764
1902	1,100	42,252	1,495	1,231	3,415	88,536	36,791	2,135	5,040
1903	1,155	35,168	2,281	1,741	5,727	94,377	40,873	2,374	5,534
1904	2,422	35,166	2,003	1,687	5,251	85,049	38,812	2,428	6,742
1905	3,252	48,005	1,680	567	7,830	3,401	61,196	2,114	7,981
1906	3,586	35,366	2,044	2,417	9,828	15,095	118,238	2,559	17,246
1907	3,974	33,522	2,496	4,913	11,096	4,793	128,257	2,781	12,588
1908	3,278	47,265	2,670	2,073	10,034	2,587	130,063	2,256	19,969

Year.	Borax.	Bauxite.	Calcium Carbide.	Cement.	Chalk (d), Crude White.	Chrome Ore.	Coal.		Coke.
							Bitum., Anthracite Cannel.	Lignitic.	
1897	.....	.....	.....	42,364	.....	.....	6,072,029	8,111,076	435,161
1898	.....	.....	.....	53,519	.....	.....	5,820,332	8,450,149	332,579
1899	.....	.....	.....	63,388	.....	.....	6,220,489	8,616,751	462,577
1900	2,403	29,383	7,703	79,303	63,929	18,728	7,384,049	7,960,313	512,690
1901	2,537	24,113	9,526	87,262	29,611	18,222	6,297,389	8,108,943	400,197
1902	2,057	26,698	11,287	52,018	26,408	10,152	6,425,658	7,882,010	382,488
1903	2,567	22,316	14,081	49,870	33,362	13,919	6,766,513	7,962,123	432,819
1904	2,603	27,849	14,840	60,188	32,581	18,132	7,299,042	7,669,099	550,302
1905	2,802	39,137	17,256	148,118	(f)35,529	11,998	9,399,693	7,945,261	713,619
1906	3,044	43,117	22,819	233,119	18,871	17,124	9,253,711	8,430,441	565,561
1907	2,014	59,989	25,834	241,475	16,035	19,508	13,729,849	8,963,103	584,220
1908	1,903	48,064	29,024	168,504	17,606	16,974	11,661,503	8,581,966	575,924

Year.	Peat.	Briquettes and Peat Coke.	Cobalt and Nickel Ore.	Copper.				Copperas.	Cryolite.
				Ore.	Ingots.	Bars, Wire and Sheets.	Sulphate.		
1897	.....	.....	.....	.....	67,573	400	.....	.....	.....
1898	.....	.....	.....	.....	73,291	450	.....	.....	.....
1899	.....	.....	.....	.....	70,091	610	.....	.....	.....
1900	19,807	137,153	13,032	10,930	83,503	906	2,369	752	1,460
1901	15,102	92,037	12,186	4,614	58,620	786	1,211	501	1,249
1902	16,696	81,854	14,630	14,630	76,050	540	2,499	807	1,332
1903	14,640	84,635	36,927	13,714	83,261	568	1,691	778	1,082
1904	9,071	125,477	14,555	7,949	110,231	719	1,735	765	1,139
1905	11,439	191,753	39,590	10,137	102,218	927	2,180	666	1,143
1906	19,428	162,650	22,557	9,941	126,071	409	1,702	621	(k)
1907	15,238	195,403	29,296	19,295	124,116	772	4,519	1,165	(k)
1908	15,266	192,391	17,402	17,456	157,434	951	5,078	7,234	(k)

Year.	Gold, Silver and Platinum Ores.	Graphite.	Gypsum.	Iodine.	Iron.		Lead.		
					Ore.	Pig.	Ore.	Pig and Scrap.	Lead White.
1897	8,927	17,366	.....	164	3,185,644	423,127	.....	35,092	696
1898	7,841	20,269	.....	216	3,516,577	384,561	.....	47,497	822
1899	7,597	23,400	.....	191	4,165,372	612,652	.....	55,635	703
1900	9,153	22,495	7,571	236	4,107,840	726,712	51,338	70,252	698
1901	8,764	17,374	7,622	266	4,370,022	267,503	100,196	52,886	423
1902	6,585	19,392	8,177	220	3,957,403	143,040	71,078	39,006	357
1903	4,386	20,953	8,328	320	5,225,336	158,347	67,573	52,440	442
1904	5,960	23,533	9,550	272	6,061,127	178,256	83,807	61,388	622
1905	6,225	26,143	11,247	377	6,085,196	158,700	92,667	78,528	2,488
1906	4,819	28,175	11,062	297	7,629,730	409,083	90,027	71,191	2,342
1907	3,601	29,405	14,662	147	8,476,076	443,624	137,861	75,200	3,037
1908	1,922	34,491	14,599	194	7,732,949	252,779	133,597	77,649	3,558

Year.	Magnesite.	Manganese Ore.	Mineral Pigments.	Nickel.	Ozokerite.	Petroleum Products.		Phosphorus.
						Illuminating Oil.	Lubricating Oil.	
1897	.....	86,911	.....	1,390	.....	946,344	83,957	.....
1898	.....	130,711	.....	1,467	.....	954,646	97,028	.....
1899	.....	196,825	.....	1,391	.....	963,943	106,624	.....
1900	13,920	204,420	12,107	1,712	3,457	989,361	124,505	331
1901	8,897	222,010	9,403	1,947	1,981	985,904	118,999	313
1902	12,237	204,647	7,719	1,458	1,585	1,006,839	125,667	350
1903	14,958	223,709	9,888	1,507	1,663	1,067,697	147,837	222
1904	15,877	255,760	10,494	1,712	1,300	1,076,324	142,929	220
1905	19,459	262,311	11,473	1,955	1,114	1,070,252	143,926	198
1906	25,527	331,171	3,960	3,478	1,303	984,134	180,989	208
1907	30,857	393,327	2,166	2,182	1,653	1,115,205	226,609	165
1908	28,305	334,133	184	3,057	1,447	1,123,631	216,887	141

Year.	Phosphate Rock.	Potassium Salts.							Pumicestone.	Pyrites.
		Chloride.	Cyanide.	Iodide	Nitrate.	Carbonate.	Hydroxide.	Sulphate.		
1897	289,234	715	7	18	2,889	1,734	.....	912	.....	356,869
1898	270,988	422	2	16	1,895	1,486	.....	999	.....	376,817
1899	407,457	443	3	9	1,785	1,737	.....	533	.....	437,732
1900	320,138	484	2	10	2,047	1,522	283	856	2,154	457,679
1901	351,155	462	2	1,529	1,758	1,529	165	690	2,336	488,633
1902	430,043	261	3	10	1,889	2,112	42	266	2,070	482,095
1903	461,092	40	3	8	2,163	1,850	52	81	2,697	519,317
1904	508,634	47	2	10	2,349	1,955	61	121	3,000	503,503
1905	501,048	223	3	30	2,156	1,693	24	131	3,240	552,184
1906	531,195	181	3	18	1,918	2,099	44	257	5,463	579,355
1907	579,505	1,615	1	8	1,815	2,304	92	141	5,443	742,526
1908	736,127	1,403	4	7	2,200	1,773	50	169	6,154	659,871

Year.	Quicksilver.	Salt.	Slag and Slag Wool.	Slag, Basic Slag, Ground.	Sodium Salts.			Strontianite.
					Soda, Calcined.	Nitrate (Chile Saltpeter.)	Sulphate.	
1897	(c)	(c)	670,224	110,216	916	465,493	.....	.....
1898	560	21,957	685,118	88,374	524	425,054	.....	.....
1899	572	22,040	892,764	68,305	515	526,944	.....	.....
1900	555	21,738	974,947	103,481	373	484,544	9,450	8,701
1901	651	23,901	733,931	87,152	178	529,568	7,921	19,739
1902	648	26,404	831,282	103,107	121	467,024	7,308	34,035
1903	674	20,118	877,394	132,337	114	467,130	6,058	24,183
1904	691	18,743	846,738	150,838	179	506,172	9,598	18,055
1905	729	20,726	888,665	198,763	143	540,916	4,752	13,720
1906	698	16,997	813,388	193,895	189	593,218	7,405	5,212
1907	831	23,109	568,046	568,046	257	501,131	10,446	5,595
1908	647	24,975	562,863	562,853	293	604,457	44,037	4,211

Year.	Sulphur.	Sulphuric Acid	Superphosphate.	Tin, Crude.	Zinc.			
					Ore.	Spelter.	Drawn or Rolled.	Zinc-white, Zinc-gray, Lithophon.
1897	25,305	.....	.....	12,395	24,735	19,734	130	3,532
1898	30,269	.....	.....	14,623	48,050	24,116	53	3,653
1899	31,196	.....	.....	12,253	57,880	23,691	95	4,226
1900	40,689	20,634	72,062	12,454	68,982	24,263	145	4,884
1901	32,750	18,502	107,365	12,910	75,533	21,250	306	3,673
1902	32,798	22,205	109,374	13,760	61,407	25,946	134	3,986
1903	41,545	13,418	82,740	13,925	67,156	25,749	237	4,667
1904	41,030	16,067	91,288	14,352	93,515	26,339	151	6,461
1905	39,989	33,837	109,666	13,501	126,577	29,583	54	7,802
1906	41,390	74,536	76,384	14,098	178,953	39,314	97	9,140
1907	44,700	59,753	62,877	12,814	184,703	28,459	134	10,189
1908	44,066	61,391	71,350	14,038	199,840	32,622	285	7,079

MINERAL EXPORTS OF GERMANY. (a)

Year.	Aluminum, Refined and Crude.	Aluminum, Nickel Wares, etc.	Aluminum Sulphate.	Ammonium.		Antimony and Arsenic Ores.	Antimony.	
				Carbonate and Chloride.	Sulphate.		Metallic.	Salts.
1897	.....	1,899	.....	.....	2,623	.....	.....	.....
1898	.....	2,045	.....	.....	4,083	.....	.....	.....
1899	.....	2,312	.....	.....	1,553	.....	.....	.....
1900	269	2,398	29,372	3,196	2,431	284	131	786
1901	282	2,270	31,171	3,196	9,842	283	76	826
1902	410	2,606	34,005	3,351	5,744	410	105	954
1903	353	2,865	28,513	2,778	5,592	427	83	873
1904	407	3,077	29,311	3,106	10,696	486	250	964
1905	1,192	3,476	34,776	3,579	27,589	287	218	1,097
1906	1,111	1,321	25,937	3,555	37,288	548	221	997
1907	1,119	1,142	24,759	3,118	57,439	930	255	1,168
1908	572	673	23,370	889	72,001	604	149	1,028

Year.	Arsenic.		Asbestos.	Barytes. (b)	Barium.		Bauxite.	Borax.	Bromine.
	Metallic.	White, etc.	Crude.		Chloride and Salts of.	White.			
1897	.....	.....	.....	.....	.....	.....	.....	.....	.....
1898	.....	.....	.....	.....	.....	.....	.....	.....	.....
1899	.....	.....	.....	.....	.....	.....	.....	.....	.....
1900	14	1,573	496	59,012	5,927	2,717	44	2,894	191
1901	28	1,534	638	67,526	6,803	2,765	137	2,563	228
1902	46	2,036	709	56,026	7,358	2,922	32	2,836	153
1903	32	1,903	513	72,455	8,417	3,187	19	2,779	155
1904	50	1,956	738	69,564	8,596	3,777	21	2,741	208
1905	40	1,753	1,173	81,134	9,550	4,382	6	2,720	156
1906	.....	2,282	1,938	90,819	6,541	10,721	398	2,795	172
1907	65(m)	1,733	1,707	111,209	4,189	8,454	517	3,049	118
1908	45(m)	1,914	1,371	91,137	3,415	5,191	782	2,377	228

Year.	Bromine Salts.	Calcium.		Cement.	Chalk. (d) Crude White.	Chromium.		Coal.	
		Carbide.	Chloride.			Ore.	Alum.	Bituminous Anthracite and Cannel	Lignitic.
1897	.....	.....	.....	524,557	.....	.....	.....	12,389,907	19,112
1898	.....	.....	.....	551,744	.....	.....	.....	13,989,223	22,155
1899	.....	.....	.....	580,255	.....	.....	.....	13,943,174	20,925
1900	255	224	1,315	600,386	11,860	427	1,192	15,275,805	52,795
1901	249	275	888	560,612	14,134	581	1,299	15,266,267	21,715
1902	357	126	1,346	699,378	8,475	846	1,758	16,101,141	21,766
1903	435	335	1,831	742,381	12,211	37	1,921	17,389,934	22,499
1904	411	608	2,381	635,248	11,359	47	2,432	17,996,726	22,135
1905	634	709	2,831	675,664	13,081	43	2,507	18,156,998	20,118
1906	643	545	(l)	736,579	4,287	(k)36	2,942	19,550,964	18,759
1907	655	918	(l)	692,982	2,919	(k)149	3,110	20,056,503	22,065
1908	507	844	(l)	528,805	2,108	111	3,205	21,062,362	27,877

Year.	Coke.	Peat.	Briquets and Peat Coke.	Cobalt and Nickel Ores.	Copper.			
					Ore.	Bars, Sheets and Wire.	Ingots.	Sulphate.
1897	2,161,886	.....	.....	.....	.....	.....	7,183	.....
1898	2,133,179	.....	.....	.....	.....	.....	6,972	.....
1899	2,137,985	.....	.....	.....	.....	.....	7,061	.....
1900	2,229,188	8,849	550,222	186	25,686	9,787	5,505	1,881
1901	2,096,931	11,588	529,765	96	26,678	7,700	5,097	1,942
1902	2,182,383	13,410	697,799	3	17,031	10,599	4,678	1,366
1903	2,523,351	16,986	895,145	1	15,986	10,715	4,333	1,880
1904	2,716,855	14,830	917,526	83	19,235	12,594	4,223	2,231
1905	2,761,080	16,009	936,694	107	28,908	10,006	5,958	2,180
1906	3,415,347	15,680	1,095,029	(i)	6,414	10,728	7,241	3,018
1907	3,792,580	25,746	1,260,135	(i)	20,950	13,411	6,113	2,016
1908	3,579,278	26,816	1,493,103	(i)	21,741	17,209	6,778	2,996

Year.	Copperas.	Cryolite.	Fluor-spar.	Graphite.	Gypsum.	Iodine.	Iron.		
							Ore.	Pig.	Oxide.
1897	.....	.....	.....	2,422	.....	26	3,230,391	90,885	.....
1898	.....	.....	.....	2,936	.....	26	2,933,734	187,375	.....
1899	.....	.....	.....	2,703	.....	26	3,119,878	182,091	.....
1900	3,829	315	12,749	2,068	39,933	29	3,247,888	129,409	1,052
1901	4,125	367	13,436	1,667	40,397	27	2,389,870	150,448	1,549
1902	4,360	486	14,177	1,691	42,859	24	2,868,068	374,256	1,755
1903	3,986	349	13,028	1,810	51,874	29	3,343,510	418,072	2,006
1904	3,514	310	13,540	1,815	55,043	30	3,440,846	225,897	2,093
1905	4,495	286	15,019	1,971	52,886	27	3,698,563	380,824	2,188
1906	4,712	(k)	15,493	2,013	63,516	46	3,851,791	479,772	2,097
1907	6,212	(k)	16,624	2,176	70,737	44	3,904,400	275,170	1,066
1908	4,397	(k)	14,702	2,481	60,987	51	3,067,870	257,850	991

Year.	Lead.					Lime, Chloride of	Magnesite.	Magnesium Chloride.	Manganese Ore.
	Ore.	Pig and Scrap.	Litharge.	White.	Red.				
1897	.....	24,075	.....	14,786	.....	.....	.....	.....	8,615
1898	.....	24,867	.....	16,473	.....	.....	.....	.....	4,810
1899	.....	24,491	.....	16,360	.....	.....	.....	.....	7,040
1900	1,309	18,825	3,577	15,126	6,603	25,954	2,392	13,375	2,454
1901	891	20,820	4,876	16,966	7,776	32,705	2,485	16,102	5,584
1902	2,024	23,100	4,072	19,070	8,372	29,694	2,955	14,757	4,528
1903	1,270	30,243	5,175	20,765	7,617	28,849	2,812	17,008	11,138
1904	1,312	23,169	5,410	16,638	7,544	30,078	1,917	16,706	5,536
1905	1,496	32,515	4,466	16,478	8,902	30,667	2,552	21,673	4,116
1906	1,915	27,067	2,493	14,022	9,450	29,435	2,843	26,708	2,555
1907	1,296	38,259	4,470	13,651	9,371	24,946	3,264	29,566	3,554
1908	1,189	40,626	5,243	13,720	9,722	23,906	4,027	27,538	2,332

Year.	Mineral Pigments.	Nickel.	Ozokerite.	Petroleum Products. (f)		Phosphorus.	Phosphate Rock.
				Illuminating Oil.	Lubricating Oil.		
1897	.....	169	.....	.....	.....	.....	4,000
1898	.....	203	.....	.....	.....	.....	5,100
1899	.....	295	.....	.....	.....	.....	2,504
1900	13,958	268	1,592	843	1,455	170	1,123
1901	12,671	390	1,700	655	963	149	2,260
1902	14,392	689	1,856	824	1,177	260	1,103
1903	15,161	700	2,027	701	1,975	286	4,342
1904	16,305	1,203	2,447	760	1,763	236	3,222
1905	17,603	1,034	2,757	7,286	1,746	228	3,720
1906	4,290	954	509	673	9,982	228	5,484
1907	4,097	930	602	770	10,552	165	1,494
1908	5,560	1,349	924	1,013	10,835	141	1,196

Year.	Potassium Salts.						Potassium and Potassium-Magnesium Sulphate.	Pumice Stone.	Pyrites.
	Carbonate.	Cyanide.	Chloride.	Hydroxide.	Iodide.	Nitrate.			
1897	13,100	1,084	80,389	.....	124	8,986	.....	.....	15,387
1898	13,456	1,907	96,236	.....	135	10,969	.....	.....	19,220
1899	11,917	1,645	101,045	.....	145	15,146	.....	.....	16,985
1900	15,761	1,338	114,469	15,379	138	14,744	38,125	561	24,936
1901	15,567	2,089	118,959	14,892	145	13,439	37,216	699	23,680
1902	14,041	3,257	106,925	13,804	152	9,734	40,487	691	35,370
1903	13,121	2,017	125,302	13,006	154	9,671	56,455	794	32,611
1904	10,777	3,290	140,765	24,963	174	10,405	64,400	943	30,666
1905	11,963	4,005	156,440	22,246	170	12,140	67,286	939	35,195
1906	12,543	5,049	171,994	21,772	168	11,564	54,557	1,578	35,829
1907	13,314	5,210	173,635	20,254	146	12,668	128,344	2,590	24,183
1908	13,062	4,990	174,359	25,577	127	10,643	132,943	6,057	16,394

Year.	Quicksilver.	Salt.	Slag and Slag Wool.	Slag, Basic.	Sodium Salts.					
					Bicarbonate.	Carbonate.	Hydroxide.	Nitrate. (Chile Saltpeter)	Soda, Calcined.	Sulphate and Sulphite.
1897	(e)	.....	27,723	169,336	.....	.....	.....	11,364	45,672	.....
1898	97	225,548	29,931	187,598	.....	.....	.....	12,884	37,106	.....
1899	23	241,036	25,565	199,382	.....	.....	.....	13,910	40,566	.....
1900	23	236,291	32,494	174,563	1,314	1,392	1,913	14,159	44,316	41,572
1901	27	286,424	27,269	202,738	1,086	1,382	4,926	13,481	45,967	45,462
1902	109	328,324	22,726	162,062	954	2,449	5,650	14,737	33,109	56,748
1903	62	399,184	14,674	216,191	1,016	2,982	5,886	17,583	46,086	47,660
1904	43	347,351	38,587	258,767	1,524	3,050	5,084	21,075	43,590	45,506
1905	48	284,203	28,032	270,905	1,851	4,113	5,925	20,531	46,768	54,377
1906	21	297,878	49,912	354,116	2,120	5,880	6,101	22,099	41,598	64,217
1907	26	292,288	46,680	399,194	1,764	2,680	7,462	22,715	36,802	69,231
1908	27	319,659	74,827	353,978	1,763	4,065	7,626	23,800	57,892	78,519

Year.	Sodium and Potassium Salts.		Stassfurt Salts.	Strontium.		Sulphur.	Sulphuric Acid.
	Chromates.	Sulphides.		Carbonate.	Salts.		
1897	.....	.....	337,577	.....	.....	.....	.....
1898	.....	.....	370,829	.....	.....	.....	.....
1899	.....	.....	367,828	.....	.....	.....	.....
1900	3,741	2,461	468,277	74	496	1,146	37,738
1901	2,791	2,763	592,347	384	1,022	621	42,853
1902	2,656	4,565	499,220	762	1,546	576	47,666
1903	2,977	5,845	501,385	819	1,389	1,052	50,109
1904	2,272	5,489	631,762	613	1,207	1,418	52,696
1905	2,133	6,569	852,454	613	1,386	1,198	48,701
1906	2,877	6,730	831,293	1,726	1,578	1,582	52,720
1907	3,016	8,103	839,889	1,462	1,671	1,501	49,950
1908	4,398	6,519	818,569	1,493	1,822	1,779	60,613

Year.	Superphosphate.	Tin, Crude.	Zinc.				Zinc-White, Zinc-Gray, and Lithophon.
			Ore.	Spelter and Scrap.	Drawn or Rolled.	Sulphate.	
1897	.....	861	30,047	51,341	17,453	.....	17,631
1898	.....	874	30,408	51,324	14,477	.....	18,674
1899	.....	1,121	25,192	46,334	18,281	.....	19,489
1900	77,118	1,626	34,941	51,899	16,709	382	20,729
1901	79,190	1,683	41,002	54,490	16,517	324	24,201
1902	77,818	2,271	46,965	70,292	17,015	330	28,400
1903	99,672	2,581	40,458	67,057	15,715	264	27,527
1904	129,925	2,965	40,488	70,063	17,917	332	26,898
1905	115,886	3,259	38,972	67,675	18,982	296	27,877
1906	104,713	4,845	42,546	69,142	17,794	426	26,296
1907	115,049	4,244	34,863	93,649	21,484	425	30,453
1908	125,931	3,714	39,450	97,008	18,675	346	26,349

(a) From *Statistisches Jahrbuch für das Deutsches Reich*. (b) Includes witherite. (d) Includes precipitated chalk. (e) Not reported. (f) Of domestic production only. (h) Includes nickel ore. (i) Included under chromium ore. (k) Included under bauxite. (l) Included under magnesium chloride. (m) Includes all alkali metals.

GREECE.

The statistics of mineral production in Greece, according to the latest available reports, are summarized in the following table:

MINERAL PRODUCTION OF GREECE. (a)
(In metric tons or dollars; 1 drachma = 20 cents.)

Year.	Chrome Ore.	Emery.	Gypsum.	Iron Ore.	Iron Ore. Manganiferous.	Lead. Soft.	Lead Ore. Argentiferous.	Lead. Argentiferous.	Lead. Fume.	Lignite.
1896.....	1,600	3,650	120	225,600	166,850	480	3,200	14,700	1,550	14,000
1897.....	563	3,024	51	260,828	182,850	520	2,815	15,946	2,785	20,018
1898.....	1,367	3,932	83	287,100	213,938	305	(b)	18,888	2,655	17,310
1899.....	4,386	4,360	81	331,030	294,320	291	(b)	18,768	2,584	12,150
1900.....	5,600	6,328	129	279,880	243,920	245	878	16,150	2,045	12,940
1901.....	4,580	5 691	671	278,640	196,152	(b)	(b)	17,644	5 292	9,726
1902.....	11 680	4,727	172	364 340	170 040	(b)	430	14 048	1 647	6,500
1903.....	8 478	5 586	94	531 804	152,740	(b)	(b)	12,361	(b)	8,687
1904.....	6,530	6,182	117	422,159	108,319	(b)	(b)	15,186	(b)	13,500
1905.....	8,900	6,972	57	465,622	89,687	(b)	(b)	13,729	(b)	11,757
1906.....	11,530	7,718	85	680,620	96,382	(b)	(b)	12,308	(b)	11,582
1907.....	11,730	10,652	105	768,863	92,970	(b)	(b)	13,814	(b)	11,719

Year.	Magnesite.			Manganese Ore.	Pussolan.	Sea Salt.	Sulphur.	Zinc Ore.	
	Crude.	Bricks.	Calcined.					Blende.	Calamine. Calcined.
1896.....	11,600	892	1,514	15,500	31,300	22,800	1,540	1,750	20,950
1897.....	11,311	826	686	11,868	42,600	20,421	358	3,118	22,817
1898.....	14,829	516	129	14,097	70,700	25,250	135	1,139	20,906
1899.....	17,184	542	3,087	17,600	46,375	37,125	1,150	1,137	21,770
1900.....	17,277	534	807	8,050	49,426	22,411	891	(b)	18,751
1901.....	20,348	500	2,009	14,166	80,169	23,079	3,212	454	17,764
1902.....	23,020	935	4,730	14,960	32,514	25,200	1,391	(b)	18,670
1903.....	28,415	(b)	(b)	9,340	40,978	26,000	1,266	(b)	12,350
1904.....	9,133	(b)	(b)	8,549	44,644	27,000	1,225	(b)	19,913
1905.....	37,063	(b)	(b)	8,171	41,900	25,201	1,126	(b)	22,562
1906.....	40,584	(b)	(b)	10,040	30,622	25,167	(b)	(b)	26,258
1907.....	55,816	(b)	(b)	11,139	39,637	26,966	(b)	(b)	30,346

(a) Statistics up to 1903 communicated by E. Grohmann, Seriphos. (b) Not reported.

INDIA.

The official statistics of mineral production in British India are summarized in the subjoined table:

MINERAL PRODUCTION OF INDIA. (a)
(In metric tons or dollars: £1 = $5.)

Year.	Amber.	Coal.	Chromite.	Diamonds Carats.	Gold. (c)	Graphite.	Iron Ore.	Jade. (a) Cwt.	Magnesite.
1896	(b)	3,909,764	.....	.....	$7,085,432	(b)	50,559	215	(b)
1897	(b)	4,128,330	.....	.....	8,041,055	(b)	61,097	219	(b)
1898	$5,080	4,681,927	.....	.....	7,798,709	(b)	(d)42,524	196	(b)
1899	765	5,174,752	.....	.....	8,357,087	1,548	(d)52,832	228	(b)
1900	515	6,222,501	.....	.....	9,205,518	1,859	(d)57,912	142	(b)
1901	55	6,741,899	.....	.....	9,394,723	2,530	(d)58,725	206	(b)
1902	2,160	7,543,272	.....	.....	9,611,985	4,648	(d)77,273	174	3,597
1903	2,070	7,557,400	248	.....	11,203,926	3,448	(d)62,237	90	838
1904	4,190	8,348,561	3,596	286.5	11,513,340	2,955	72,757	130	1,193
1905	4,725	8,552,422	2,708	172.4	11,700,057	2,361	104,174	106	2,645
1906	3,545	9,939 782	4,375	305.9	10,852,946	2,642	75,295	116	1,861
1907	1,925	11,325,696	7,389	268.0	10,251,494	2,472	68,737	249	189

Year.	Manganese Ore.	Mica. (e) Cwt.	Petroleum. *Gallons.*	Rubies.	Salt.	Saltpeter (Potassium nitrate.)	Tin Ore.
1896	57,782	452	15,057,094	$171,884	1,043,171	21,425	82
1897	74,862	652	19,128,828	200,613	937,932	26,845	62
1898	61,419	527	22,234,438	289,750	1,043,862	21,224	40
1899	88,524	497	32,934,007	454,240	977,269	18,555	64
1900	129,865	1,025	37,729,211	486,630	1,071,877	20,189	94
1901	122,831	1,505	50,075,117	522,380	1,208,933	17,711	63
1902	160,311	806	56,607,668	434,475	1,116,797	17,320	91
1903	174,563	1,002	87,859,069	444,095	1,021,581	18,711	100
1904	152,708	828	118,491,382	453,060	1,319,535	14,200	63
1905	257,969	1,172	144,798,444	.....	1,426,066	15,745	68
1906	503,684	2,458	140,553,122	465,115	1,371,172	16,822	57
1907	912,718	1,977	152,045,677	577,325	1,120,236	18,165	77

(a) Records of the Geological Survey of India. (b) Not reported. (c) £1 = $4.866. (d) Production of iron ore in Bengal only. (e) Exports.

ITALY

The following tables itemize the statistics of the production and the foreign commerce of mineral and metallurgical products in Italy:

MINERAL PRODUCTION AND REFINED PRODUCTS OF ITALY. (a)
(In metric tons or dollars; 5 lire=$1.)

Year.	Alum.	Aluminum Sulphate.	Alunite.	Antimony.	Antimony Ore.	Asphalt, Mastic and Bitumen.	Asphaltic Rock.	Barytes.
1895	995	2,950	7,000	423	2,241	14,491	46,713	(b)
1896	850	2,390	6,000	538	5,086	12,490	45,456	(b)
1897	1,030	2,310	6,500	404	2,150	18,644	55,339	(b)
1898	1,165	2,915	7,000	380	1,931	17,813	93,750	12,400
1899	945	2,330	5,800	581	3,791	41,732	81,987	12,545
1900	1,097	2,403	5,200	1,174	7,609	33,127	101,738	14,003
1901	1,075	2,260	4,900	1,721	8,818	31,814	104,111	13,245
1902	.....	.....	8,200	.....	6,116	33,684	64,245	.....
1903	.....	.....	8,100	905	6,927	35,757	89,078	.....
1904	2,490	2,210	8,000	836	5,712	34,227	111,390	250
1905	2,975	2,740	8,500	327	5,083	26,838	106,586	590
1906	2,878	2,800	7,500	537	5,704	34,386	130,225	800
1907	6,350	5,110	7,600	610	7,892	35,884	161,126	1,720

Year.	Borax, Refined.	Boric Acid.		Coal. (c)	Coal. (Briquettes).	Coke	Copper.		
		Crude.	Refined.				Ore.	Ingot, etc.	Sulphate
1895	944	2,633	253	305,321	451,470	394,043	83,070	2,375	.....
1896	943	2,616	253	276,197	422,409	426,906	90,408	2,842	4,756
1897	990	2,704	260	314,222	549,050	430,617	93,377	2,980	5,337
1898	702	2,650	166	341,327	594,500	469,228	95,128	3,230	6,364
1899	709	2,674	129	388,534	566,000	485,951	94,764	3,032	7,795
1900	858	2,491	283	479,896	703,740	487,331	95,644	2,797	13,191
1901	544	2,558	347	425,614	754,800	490,803	107,750	3,097	15,374
1902	.....	2,763	.....	414,569	.....	.....	101,142	.....	14,601
1903	.....	2,583	.....	346,887	724,998	554,559	114,823	.....	18,164
1904	569	2,624	314	362,151	903,610	607,297	157,503	2,313	17,237
1905	1,007	2,700	749	412,916	842,250	627,984	149,035	1,175	26,212
1906	1,062	2,561	562	473,293	829,277	672,689	147,137	.....	34,270
1907	880	2,305	466	453,137	787,087	717,704	167,619	17,491	45,262

Year.	Gold.		Graphite.	Iron and Steel.				
	Ore.	Bullion.		Ore.	Pig.	Bar, Sheet, Pipe, Wire, etc.	Tin Plate.	Steel.
1895	7,099	$186,074	2,657	183,371	9,213	163,824	5,860	50,314
1896	7,659	172,552	3,148	203,966	6,967	139,991	2,918	65,955
1897	10,723	209,998	5,650	200,709	8,393	149,944	6,500	63,940
1898	9,549	124,869	6,435	190,110	12,387	167,499	7,200	87,467
1899	11,859	75,294	9,990	236,549	19,218	197,730	8,000	108,501
1900	5,840	38,212	9,720	247,278	23,990	190,518	10,000	115,887
1901	890	2,725	10,313	232,299	15,819	180,729	7,550	123,310
1902	1,215	.....	9,210	240,705	.....	.....	.....	.....
1903	5,734	41,933	7,920	374,790	90,744	177,392	11,275	154,134
1904	6,746	43,063	9,765	409,460	112,598	181,385	16,655	177,066
1905	1,200	9,169	10,572	366,616	181,248	205,915	18,560	244,793
1906	6,543	47,321	10,805	384,217	180,940	236,946	16,350	332,924
1907	13,475	33,562	10,989	517,952	148,996	248,157	24,423	346,749

Year.	Lead.		Manganese Ore.	Manganiferous Iron Ore.	Marble.	Petroleum, Crude.	Petroleum, Benzine, etc.	Pumice-Stone.
	Ore.	Pig.						
1895	30,632	20,353	1,569	5,860	186,900	3,594	4,191	(b)
1896	33,545	20,786	1,890	10,000	209,428	2,524	2,734	(b)
1897	36,200	22,407	1,634	21,262	236,958	1,932	3,392	(b)
1898	33,930	24,543	3,002	11,150	271,725	2,015	5,040	2,766
1899	31,046	20,543	4,356	29,874	313,744	2,242	5,384	7,300
1900	35,103	23,763	6,014	26,800	310,336	1,683	6,077	7,000
1901	43,449	25,796	2,181	24,290	334,146	2,246	4,211	8,300
1902	42,330	26,494	2,477	23,113	.....	2,633	.....	.....
1903	42,443	22,126	1,930	4,735	.....	2,486	4,577	.....
1904	42,846	23,475	2,836	2,836	390,118	3,543	6,388	11,600
1905	39,030	19,077	5,384	5,384	389,869	6,122	9,924	11,300
1906	40,945	21,268	3,060	20,500	430,202	7,452	2,322	16,366
1907	(d)43,037	22,978	3,654	18,874	434,612	8,326	2,989	11,500

Year.	Pyrites. (Cupriferous in part).	Quicksilver.		Salt.			Silver.	
		Ore.	Metal.	Brine.	Rock.	Sea.	Ore.	Bullion. *Kg.*
1895	38,586	10,504	199	10,605	18,710	448,335	870	44,189
1896	45,728	14,305	186	11,974	17,300	422,555	640	38,075
1897	58,320	20,659	192	11,725	19,801	429,253	405	45,313
1898	67,191	19,201	173	11,546	18,199	451,426	435	43,437
1899	76,538	29,322	205	11,021	18,721	363,826	540	33,645
1900	71,616	33,930	260	10,890	18,331	338,034	584	31,169
1901	89,376	38,614	278	10,690	23,054	401,443	511	32,464
1902	93,177	44,261	.....	10,581	23,677	424,239	421	.....
1903	101,455	55,528	312	10,962	25,911	451,633	405	24,388
1904	112,004	60,403	352	11,878	18,638	433,810	143	24,943
1905	117,667	63,378	369	12,756	19,669	405,274	170	20,215
1906	122,364	80,638	417	13,751	19,007	496,872	48	20,362
1907	126,925	76,561	434	19,238	31,540	454,454	62	20,502

Year.	Sulphur.			Talc. Ground.	Zinc.	
	Crude (Fused).	Ground.	Refined.		Ore.	Spelter.
1895	370,766	91,517	75,329	(b)	121,197	*Nil.*
1896	426,353	89,292	71,072	(b)	118,171	*Nil.*
1897	496,658	69,178	85,872	(b)	122,214	250
1898	502,351	146,001	99,494	12,760	132,099	250
1899	563,697	161,509	110,213	11,000	150,629	251
1900	544,119	167,466	157,957	14,415	139,679	547
1901	563,096	171,252	141,431	11,770	135,784	511
1902	510,333	.....	.....	.....	131,965	.....
1903	553,751	139,376	139,464	6,300	157,521	126
1904	527,563	189,266	163,695	6,740	148,365	189
1905	568,927	180,676	180,774	6,626	147,834	5
1906	499,814	176,476	170,990	7,894	155,751	69
1907	426,972	151,338	160,617	8,850	160,517	188

(a) From *Rivista del Servizio Minerario*. (b) Not reported. (c) Includes anthracite, lignite, fossil wood and bituminous schist. (d) Does not include 680 tons lead and zinc ore.

MINERAL IMPORTS OF ITALY. (a)
(In metric tons or dollars; 5 lire = $1.)

Year.	Antimony	Arsenic. Kg.	Asbestos.	Asphaltum.	Barytes.	Borax and Boric acid.	Cement and Hydraulic Lime.	Chalk.
1896	38	(b)	851	11,892	549	166	12,810	15,716
1897	66	2,604	619	1,632	578	253	16,680	28,937
1898	58	700	1,186	1,150	860	147	12,029	18,252
1899	64	600	1,675	1,473	936	123	14,391	13,738
1900	37	900	1,645	1,933	859	122	15,494	18,436
1901	49	1,800	2,019	1,450	825	232	14,872	20,731
1902	80	1,200	1,536	1,020	1,170	516	13,732	15,216
1903	98	4,400	1,691	1,567	1,099	504	15,547	10,063
1904	131	3,700	2,174	2,604	1,875	271	15,260	6,891
1905	117	3,400	1,806	3,252	1,444	112	15,797	5,556
1906	50	5,300	2,171	2,854	1,400	163	18,937	7,714
1907	163	3,100	3,110	3,661	1,540	307	29,024	6,156

Year.	Coal.	Copper Ore.	Copper Cement.	Copper, Brass and Bronze.	Copper and Iron Sulphates.	Gold, Unrefined. Kg.	Graphite.
1896	4,081,218	484	1,150	6,955	24,255	2,517	204
1897	4,259,643	1,611	1,049	7,999	28,878	807	315
1898	4,431,524	5,471	2,040	7,433	25,560	507	382
1899	4,859,556	2,777	1,328	7,334	27,408	326	608
1900	4,947,180	5,290	1,298	9,249	32,127	309	982
1901	4,838,994	11,047	1,987	8,659	32,058	494	102
1902	5,406,069	9,422	2,299	10,865	25,107	479	60
1903	5,546,823	9,459	649	9,588	24,566	1,396	63
1904	5,904,578	8,104	309	15,198	37,298	1,961	52
1905	6,437,539	6,879	486	18,188	30,684	5,768	107
1906	7,673,435	9,363	802	21,458	25,060	4,571	361
1907	8,300,439	18,023	888	28,937	15,989	4,443	267

Year.	Iron.			Iron and Steel	Lead.		Lead.	Mineral Paints.
	Ore.	Pig.	Wrought.	Scrap.	Ore. (c)	Metal and Alloys in Pigs.	Oxide and Carbonate	
1896	594	119,491	4,820	162,035	9,730	1,166	523	852
1897	5,831	156,019	3,801	130,938	14,854	1,178	580	888
1898	8,723	169,059	4,076	138,426	10,947	1,431	647	692
1899	20,799	191,613	4,158	245,616	7,476	3,990	662	958
1900	19,205	160,686	7,405	197,415	9,134	3,248	557	958
1901	4,054	159,972	5,695	148,305	9,063	2,926	815	865
1902	4,314	155,143	6,603	198,914	1,680	7,563	846	670
1903	5,937	126,756	6,380	206,036	689	5,398	768	859
1904	4,390	149,130	6,740	246,359	2,187	4,541	871	940
1905	4,745	136,077	7,616	276,311	465	6,764	686	974
1906	6,452	168,985	13,342	344,977	4,526	10,958	984	964
1907	22,046	231,042	11,365	362,567	4,342	9,231	953	1,119

Year.	Nickel Alloys and Manufactures.	Petroleum.	Phosphate Rock.	Potash, Ammonia and Caustic Soda.	Potassium Sulphate.	Quicksilver.	Silver, Unrefined in Bars. Kg.	Slag.
1896	411	70,217	(b)	9,841	431	30	2,291	30,275
1897	432	68,973	(b)	11,012	562	30	2,434	37,201
1898	258	70,654	65,126	11,047	928	39	991	51,199
1899	250	71,391	116,283	12,370	1,297	62	1,782	56,549
1900	232	73,089	140,281	14,077	1,670	49	2,578	32,254
1901	476	69,298	142,108	14,693	1,411	36	4,391	7,312
1902	561	68,781	159,341	17,617	1,566	57	8,768	5,634
1903	525	68,220	172,328	17,528	1,353	28	12,541	8,849
1904	652	69,233	217,162	14,846	1,663	25	15,885	3,821
1905	574	66,493	240,144	17,752	1,804	57	20,697	72,785
1906	717	64,541	307,762	16,718	1,534	12	20,410	88,118
1907	725	77,658	384,896	16,237	3,866	11	21,829	134,639

Year.	Sodium Salts.		Sod. and Pot. Nitrates, Refined.	Tin.		Zinc.			
	Carbonate.	Nitrate (Crude).		Block.	Mnfres.	Ore.	Oxide.	Spelter and Old.	Mnfres.
1896	18,927	11,685	541	1,763	91	(b)	540	2,596	3,482
1897	20,721	16,400	917	1,520	81	(b)	570	3,278	3,556
1898	20,845	19,961	702	1,722	109	216	573	2,813	3,200
1899	22,654	22,385	671	1,240	96	(b)	804	3,498	3,221
1900	23,215	27,706	511	1,643	56	85	1,034	3,627	3,543
1901	21,956	40,498	315	1,858	91	23	813	3,991	4,079
1902	26,133	24,483	314	2,114	110	131	904	3,805	4,167
1903	24,753	43,480	638	2,288	130	46	1,416	4,551	4,461
1904	27,747	32,283	613	2,170	150	362	1,124	5,202	4,168
1905	29,066	46,517	689	2,304	103	14	1,246	5,997	4,701
1906	31,170	32,508	395	3,361	167	2,042	1,920	6,835	4,421
1907	35,536	41,457	668	2,771	183	11	1,962	8,152	5,407

MINERAL EXPORTS OF ITALY. (a)

(In metric tons or dollars; 5 lire=$1.)

Year.	Antimony.	Asbestos.	Asphaltum.	Barytes.	Borax and Boric Acid.	Cement and Hydraulic Lime.	Chalk.	Coal.
1896	361	130	13,729	66	2,719	3,871	5,593	18,924
1897	271	170	15,310	143	1,618	5,330	7,556	23,191
1898	338	208	19,465	70	2,167	5,192	6,744	17,749
1899	240	245	26,402	45	2,872	5,462	5,386	20,803
1900	467	261	24,287	40	2,114	6,860	2,980	23,926
1901	765	302	21,856	32	2,190	8,463	3,428	25,594
1902	359	144	20,884	91	1,847	7,930	4,215	33,374
1903	314	222	24,303	35	901	6,325	3,802	29,219
1904	107	163	14,880	70	1,122	7,810	4,089	35,149
1905	132	236	23,740	162	2,255	8,445	5,007	38,555
1906	208	205	27,176	147	2,777	6,774	4,194	31,666
1907	115	142	26,036	152	1,330	4,47	3,118	40,769

Year.	Copper Ore.	Copper, and Iron Sulphate.	Gold.	Graphite.	Iron.		
			Unrefined. Kg.		Ore.	Pig.	Wrought.
1896	3,603	71	2,517	3,727	187,059	1,378	427
1897	2,408	18	1,381	4,164	207,619	498	1,434
1898	2,356	25	1,739	5,145	217,556	840	699
1899	1,148	20	1,162	8,114	234,515	378	611
1900	1,179	60	2,763	7,820	170,286	329	440
1901	9	20	2,955	7,169	121,592	311	499
1902	11	39	733	7,098	209,070	395	1,054
1903	15	44	1,291	7,068	98,319	810	1,670
1904	43	29	1,494	7,433	2,577	229	847
1905	77	249	1,731	6,811	11,358	1,395	951
1906	189	102	1,476	4,904	1,833	254	463
1907	179	835	802	7,474	26,000	121	687

Year.	Lead.			Mineral Paints.	Phosphate Rock.	Quicksilver.	Salt.	Silver, Unrefined, Kg.
	Ore.	Lead Alloys in Pigs.	Oxide and Carbonate.					
1896	4,731	1,419	489	2,412	(b)	155	171,740	26,854
1897	4,747	2,790	461	2,318	(b)	236	176,520	50,503
1898	4,492	5,870	414	2,884	(b)	244	126,860	68,607
1899	3,129	2,497	389	2,784	(b)	223	114,050	32,432
1900	3,741	5,018	367	2,977	1,726	259	112,900	25,310
1901	3,977	4,463	410	2,913	1,290	301	114,210	42,325
1902	3,354	5,650	404	2,953	894	215	145,190	20,427
1903	5,041	2,911	426	3,305	2,942	222	144,910	9,486
1904	5,524	1,954	347	3,231	2,812	266	130,940	24,165
1905	4,311	976	310	3,632	3,519	243	116,040	25,947
1906	8,356	2,005	315	4,502	1,652	278	126,199	18,262
1907	3,213	1,548	240	4,602	4,560	350	99,191	18,164

Year.	Slag.	Sodium Salts.			Sulphur.	Tin.		Zinc.		
		Carbonate.	Nitrate, (Crude.)	Sod. and Pot.. Nitrates, Refined.		Block.	Manufactures.	Ore.	Oxide.	Spelter and Scrap.
1896.....	4,753	279	51	306	356,370	10	89	115,454	48	33
1897.....	8,847	275	151	344	358,932	29	109	133,125	189	309
1898.....	6,861	391	79	256	405,823	34	177	130,064	110	156
1899.....	4,898	438	136	124	424,018	69	176	140,107	123	227
1900.....	4,222	486	58	129	479,139	147	153	111,870	102	359
1901.....	3,261	377	116	59	414,018	202	187	103,020	140	349
1902.....	3,615	446	346	259	439,242	236	174	114,394	122	338
1903.....	4,929	482	781	492	461,289	173	180	116,449	116	591
1904.....	4,458	376	363	230	437,067	171	151	126,393	483	263
1905.....	9,844	214	424	159	381,128	285	107	117,810	173	434
1906.....	8,990	253	80	133	336,339	303	81	144,244	667	639
1907.....	10,964	200	138	102	297,378	434	59	142,271	727	1,182

(*a*) From *Statistica del Commercio speciale di Importazione e di Esportazione.* (*b*) Not reported. (*c*) Includes argentiferous lead ore.

JAPAN.

The total mineral production of the Japanese Empire, according to the latest available returns, is shown in the following table, in metric tons, unless otherwise specified:

MINERAL PRODUCTION OF JAPAN. (a)

Year.	Antimony.		Arsenic. *Kg.*	Coal.	Copper.	Gold. *Kg.*	Graphite	Iron. Pig.	Lead.
	Sulphate.	Metal.							
1894....	1,172	403	5,387	4,300,370	19,814	806	1,091	19,474	1,577
1895....	1,061	641	7,343	4,770,313	19,103	935	77	25,863	1,978
1896....	828	517	6,043	5,100,005	20,114	964	215	27,420	1,958
1897....	348	823	13,039	5,147,103	20,425	1,037	391	28,040	772
1898....	1,005	235	7,129	6,643,047	21,023	1,159	347	23,611	1,703
1899....	712	229	5,077	6,668,608	19,421	1,675	53	23,066	1,963
1900....	81	349	4,669	7,370,667	24,317	2,124	94	24,841	1,878
1901....	118	429	10,312	8,884,812	27,392	2,475	88	29,449	1,803
1902....	88	528	12,188	9,588,910	29,034	2,975	97	32,130	1,644
1903....	153	434	6,000	10,088,845	33,245	3,140	114	33,870	1,728
1904....	104	321	4,000	10,723,796	33,187	2,765	216	38,143	1,803
1905....	96	190	8,333	11,955,946	35,944	3,048	209	53,210	2,272
1906....	97	627	5,250	13,468,529	36,963	2,873	177	57,373	4,305
1907(*d*).	(*b*)	(*b*)	4,646	13,935,952	38,001	5,090	72	43,599	3,067

Year.	Manganese Ore.	Petroleum *Gallons.*	Phosphates.	Pyrite.	Quicksilver. *Kg.*	Salt. *Hectoliters.*	Silver. *Kg.*	Sulphur.	Tin.
1894....	13,368	(*c*)5,426,071	(*b*)	(*b*)	1,547	11,411,275	79,222	18,787	38.7
1895....	17,142	7,118,962	(*b*)	(*b*)	481	(*b*)	74,815	15,557	48.3
1896....	17,967	(*c*)7,440,206	(*b*)	(*b*)	1,762	(*b*)	64,303	12,540	50.0
1897....	15,448	9,179,474	(*b*)	7,626	2,678	(*b*)	54,289	13,582	47.6
1898....	11,497	11,145,457	(*b*)	8,726	1,399	11,482,422	60,436	10,321	42.7
1899....	11,336	18,844,034	(*b*)	8,379	.....	10,483,082	56,161	10,237	18.5
1900....	15,831	30,470,068	(*b*)	16,166	270	11,890,361	58,799	14,439	12.3
1901....	16,270	39,056,820	(*b*)	17,589	750	12,463,771	54,739	16,548	14.1
1902....	10,844	34,850,129	196	18,580	1,418	11,042,192	57,635	18,287	18.6
1903....	5,616	50,724,174	191	16,149	206	6,574,890	58,704	22,914	19.0
1904....	4,324	51,573,754	13	24,886	*Nil*	7,019,650	61,339	25,587	25.0
1905....	14,017	47,132,800	1,519	25,569	349	(*b*)	82,886	24,652	26.0
1906....	54,339	60,005,957	3,037	37,038	336	(*b*)	76,247	27,589	77.0
1907(*d*)..	10,410	(*e*)70,395,921	(*b*)	36,124	547	(*b*)	88,151	28,381	30.0

(*a*) From *Résumé Statistique de l'Empire du Japon*, Tokio. (*b*) Not reported. (*c*) Crude petroleum. (*d*) Furnished by *Japan Financial and Economic Monthly*. (*e*) Estimated.

MEXICO.

Owing to the incompleteness of the Mexican statistics of production, we are unable to give any satisfactory table. Exports may, however, be taken as indicating the condition of the mining industry. We owe the statistics for 1908, together with a complete revision of this table, to the courtesy of Don Miguel M. Irigoyen, chief of the Section of Statistics, Secretaria de Hacienda y Credito Publico.

MINERAL EXPORTS OF MEXICO. (a)
(In metric tons or Mexican dollars.)

Year.	Antimony		Coal.	Copper.		Gold.				
	Metal.	Ore.		Ore.	Ingot.	Ore.	Bullion.	Specie.	Cyanide.	Sulphide.
1895....	.....	600	61,686	3,006	20,429	$ 103,773	$ 4,920,504	$ 175,098	$ 31,231	$ 3,026
1896....	.....	3,261	75,541	144	20,659	206,874	5,533,789	261,078	161,784	44,890
1897....	.....	5,873	105,298	1,094	16,858	365,226	6,220,765	202,223	226,986	33,916
1898....	.....	5,932	118,553	13,146	10,362	1,037,202	6,493,735	367,704	294,730	64,061
1899....	.....	10,382	113,192	223	25,293	335,849	7,017,286	183,474	115,961	266,782
1900....	.....	2,313	38,676	408	27,970	306,392	7,435,864	192,456	128,675	177,193
1901....	.....	5,103	17,281	5,576	33,818	284,722	8,324,681	210,431	178,803	81,744
1902....	1,218	1,280	3,406	6,101	63,609	303,979	9,079,371	129,899	78,295	40,558
1903....	2,304	7,302	1,840	10,912	51,716	264,503	9,693,692	54,636	85,465	124,020
1904(c).	1,694	81	125	48,365	57,338	537,290	10,867,272	172,532	79,129	176,090
1905....	1,978	57	497	92,540	56,634	1,513,344	29,636,117	106,470	397,814	138,033
1906....	2,418	178	91	73,193	46,767	5,369,173	21,072,014	37,746	337,294	180,348
1907....	4,615	681	1,532	115,245	51,519	3,033,090	19,653,362	5,023,404	417,162	497,893
1908....	4,046	36	719	70,900	26,214	2,746,289	30,101,546	42,389	144,959	334,944

Year.	Graphite.	Gypsum.	Lead.		Silver.					
			Ore.	Base Bullion.	Ore.	Bullion.	Specie.	Sulphide.	Cyanide.	Slag.
1895...	794	1,340	568	50,122	$10,977,079	$22,178,294	$18,300,553	$ 555,475	$14,649	$72,590
1896...	795	2,050	167	48,663	9,971,053	28,565,843	18,737,331	1,495,306	38,049	64,121
1897...	759	2,095	2	60,029	11,401,176	35,775,125	21,925,347	1,663,581	123,246	39,800
1898...	1,365	1,650	(b)	60,918	11,048,358	37,137,599	16,588,789	1,663,501	257,342	46,488
1899...	2,305	1,050	1	67,441	10,766,099	37,585,911	5,580,834	1,929,085	76,942	4,810
1900...	2,561	1,600	468	74,944	12,495,524	41,468,745	22,079,055	1,893,646	67,607	87,883
1901...	762	800	(b)	79,097	9,615,939	36,348,574	12,038,158	2,141,685	259,282	93,549
1902...	1,434	(b)	118	107,366	4,108,088	45,706,576	17,753,526	1,978,919	108,344	132,093
1903...	1,404	(b)	11	100,532	11,781,048	48,276,797	16,167,673	1,642,627	135,561	289,900
1904(c)	970	(b)	1	95,010	11,000,859	45,430,020	7,251,132	1,392,356	171,452	202,594
1905...	970	27	1	101,196	8,505,834	63,504,789	20,335,297	736,228	438,094	29,012
1906...	3,915	(b)	(b)	73,699	9,619,753	63,057,152	42,390,357	595,112	434,885	(b)
1907...	3,202	(b)	11	76,158	11,396,844	68,187,169	23,848,571	785,116	483,638	785,116
1908...	1,076	(b)	26	127,010	11,230,372	63,298,059	60,405	791,698	68,848	(b)

(a) From the *Estadistica Fiscal*. The figures for the calendar years were arrived at by combining those of the successive semesters of the different fiscal years. (b) Not reported. (c) Figures for 1904 were from *Anuario Estadistico de la Republica Mexicana* for 1904.

NORWAY.

The official statistics of mineral production, imports and exports, are summarized in the following tables:

MINERAL PRODUCTION OF NORWAY. (a)
(In metric tons or dollars; 1 Krone=27 cents.)

Year.	Apatite. (b)	Chrome Ore.	Copper.		Feldspar.	Gold.	Iron.		
			Ore.	Ingot.			Ore.	Pig and Cast.	Bars and Steel.
1896.....	1,106	.....	29,910	1,067	12,223	$9,450	2,000	335	400
1897.....	872	.....	27,606	1,064	17,392	675	3,627	417	452
1898.....	3,593	.....	37,047	941	11,355	1,539	4,425	231	379
1899.....	1,500	41	43,358	1,209	19,260	2,700	4,576	406	666
1900.....	300	165	46,858	1,280	17,609	2,430	17,925	444	614
1901.....	738	85	40,726	1,073	18,323	2,700	42,252	261	376
1902.....	2,295	22	40,499	1,347	19,591	36,990	53,675	527	461
1903.....	1,795	*Nil*	35,417	1,382	18,590	8,370	58,475	509	442
1904.....	1,456	154	36,891	1,342	20,835	*Nil*	45,228	350	395
1905.....	2,522	*Nil*	37,045	1,153	22,508	*Nil*	46,582	474	253
1906.....	3,482	*Nil.*	32,203	1,333	23,896	5,400	109,259	257	317
1907.....	1,830	107	39,887	1,517	32,907	*Nil*	139,804	*Nil*	283

Year.	Molybdenite.	Nickel.		Pyrites, Iron and Copper.	Rutile.	Silver.		Zinc Ore. (c)
		Ore.	Metal.			Ore and NativeSilver.	Metal. *Kg.*	
1896.....	.....	*Nil*	16	60,507	30	527	4,664	450
1897.....	.....	*Nil*	*Nil*	94,484	32	642	5,372	908
1898.....	.....	*Nil*	*Nil*	89,763	35	497	4,802	320
1899.....	.....	220	5	95,636	20	429	4,600	379
1900.....	.....	1,888	13	98,945	40	475	4,600	204
1901.....	.....	2,018	40	101,894	55	519	5,680	90
1902.....	20	4,040	60	121,247	*Nil*	471	6,220	30
1903.....	31	5,670	75	129,939	25	481	7,269	335
1904.....	30	5,352	73	133,603	25	1,297	8,064	42
1905.....	46	5,477	77	162,012	35	1,570	7,100	4,241
1906.....	1,026	6,081	81	197,886	56	1,565	6,370	3,308
1907.....	30	5,781	81	236,038	55	1,756	6,700	400

(a) *Tabeller vedkommende Norges Bergverkdsdrift, Statistik Aarbog for Kongerikat Norge.* (b) Exports which represent production. (c) Includes lead ore.

MINERAL IMPORTS OF NORWAY. (a)
(In metric tons.)

Year.	Borax. *Kg.*	Cement and Hydraulic Lime.	Coke, Coal and Cinders. *Hectoliters.*	Copper and Brass.		Iron and Steel.	
				Plates and Bars.	Wares.	Pig.	Bars, Hoops, etc. Wrought Iron.
1896.....	38,305	16,028	.....	1,074	479	.....	26,552
1897.....	44,495	18,734	15,374,572	1,140	591	21,606	29,038
1898.....	71,590	25,403	15,409,902	1,064	807	23,106	26,203
1899.....	62,060	33,652	18,475,996	1,000	1,120	21,445	25,379
1900.....	71,124	24,511	19,002,026	696	1,164	20,844	23,010
1901.....	68,000	20,993	17,655,349	1,018	761	19,112	20,672
1902.....	(c)	18,984	19,338,615	1,118	(c)	18,969	26,685
1903.....	(c)	17,906	20,086,974	899	309	20,652	21,977
1904.....	54,953	12,845	21,049,128	683	866	18,891	24,094
1905.....	(c)	13,797	20,973,608	882	1,146	20,828	27,740
1906.....	63	11,676	21,478,000	906	783	20,197	26,015
1907.....	80	16,647	24,274,260	954	1,107	23,345	32,764

Year.	Iron and Steel—Continued.						
	Anchors, Cables and Chains.	Rails.	Nails and Spikes.	Steel.	Sheets and Plates.	Other Manufactures.	Lead in Pigs and Sheets.
1896.....	1,090	4,315	1,760	2,754	17,930	6,831	653
1897.....	1,367	7,637	2,097	4,350	23,350	10,698	848
1898.....	1,485	10,327	2,087	2,428	26,894	17,182	732
1899.....	1,394	8,137	1,529	2,652	32,192	21,400	869
1900.....	1,203	11,952	1,219	2,085	29,318	17,493	670
1901.....	1,708	22,959	1,808	1,905	31,184	18,372	590
1902.....	2,103	15,316	2,205	1,754	36,288	22,069	(c)
1903.....	1,807	4,631	1,261	1,958	42,098	18,355	311
1904.....	2,109	5,814	1,071	1,610	42,013	5,462	498
1905.....	2,224	6,566	1,222	1,436	42,203	44,414	448
1906.....	2,585	8,086	1,012	2,018	48,969	45,959	727
1907.....	2,653	6,989	991	1,592	44,432	48,965	1,192

Year.	Lead, White and Zinc Oxide.	Petroleum and Paraffin.	Potash.	Salt.	Salt-peter.	Soda.	Sulphur. (b)	Tin in Blocks, etc.	Zinc in Bars, Plates, etc.
1896.....	1,192	35,823	945	117,920	308	5,156	9,347	142	1,101
1897.....	1,119	39,810	919	164,572	277	5,492	10,701	236	1,102
1898.....	1,491	36,504	754	127,341	477	4,823	9,589	257	1,370
1899.....	1,296	42,182	802	134,583	278	4,555	10,734	546	1,509
1900.....	1,216	39,657	638	143,365	356	4,576	14,827	149	1,254
1901.....	1,321	47,011	518	127,607	208	5,220	11,149	141	1,027
1902.....	(c)	(c)	(c)	141,415	315	(c)	(c)	(c)	1,104
1903.....	(c)	58,822	457	143,110	245	4,200	8,829	106	1,015
1904.....	1,898	50,543	477	153,699	321	3,197	12,181	176	940
1905.....	1,309	43,860	393	137,800	1,048	3,704	10,240	134	967
1906.....	1,149	41,546	396	167,390	776	4,334	11,465	261	2,791
1907.....	1,245	44,124	588	163,458	1,004	5,819	11,412	(c)	3,549

MINERAL EXPORTS OF NORWAY. (a)

(In metric tons.)

Year.	Apatite.	Copper.			Feldspar.	Iodine. *Kg.*	Iron.
		Ore.	Ingot.	Scrap.			Ore.
1896.....	1,160	30,367	1,276	712	12,223	1,959	2,051
1897.....	872	15,111	1,222	670	17,302	2,395	4,242
1898.....	3,593	13,587	1,650	1,206	11,355	5,474	4,401
1899.....	1,500	7,198	1,785	1,038	19,260	16,180	12,517
1900.....	300	5,756	1,891	1,168	17,609	11,210	27,158
1901.....	738	6,041	1,405	774	(d)18,423	10,000	39,173
1902.....	2,295	4,848	1,913	(c)	(d)19,611	.....	48,775
1903.....	1,795	3,448	1,930	888	(d)18,640	11,417	41,575
1904.....	1,456	2,673	1,124	785	20,835	9,414	45,434
1905.....	2,522	3,393	958	968	20,696	12,000	60,558
1906.....	3,482	84	875	964	19,669	13,248	81,398
1907.....	1,830	1,581	1,033	1,644	29,399	13,780	132,593

Year.	Iron—Continued.				Nickel Ore.	Pyrites.	Silver Ore.
	Pig and Scrap.	Bars and Hoops.	Nails and Spikes.	Steel.			
1896.....	5,493	12	10,664	132	*Nil.*	41,562	174
1897.....	4,631	56	9,097	167	*Nil.*	70,352	119
1898.....	3,844	25	7,270	158	30	67,502	79
1899.....	6,085	337	6,089	377	63	83,912	14
1900.....	8,141	135	5,643	220	272	84,604	90
1901.....	3,250	370	6,001	179	55	104,151	6
1902.....	7,359	166	6,431	240	1	105,980	*Nil.*
1903.....	6,350	10	6,504	200	*Nil.*	118,148	*Nil.*
1904.....	10,152	13	7,477	167	30	116,550	*Nil.*
1905.....	9,920	34	8,725	88	220	147,155	*Nil.*
1906.....	7,362	8	6,786	21	*Nil.*	164,119	*Nil.*
1907.....	4,652	7	5,879	35	11	187,983	*Nil.*

(a) From *Tabeller vedkommende Norges Bergvaerksdrift und Tabeller vedkommende Norges Handel.* (b) Includes flowers of sulphur. (c) Returns not available. (d) Includes a small quantity of fluorspar.

PORTUGAL.

The subjoined table reports the mineral production of Portugal:

MINERAL PRODUCTION OF PORTUGAL. (a)
(In metric tons.)

Year.	Antimony Ore.	Arsenic.	Coal, (Anthracite) (c)	Coal Lignitic.	Ore.	Copper.	
						Cement.	Pyrite.
1895	753	(b)	8,787	10,309	202	5,055	195,304
1896	595	(b)	8,743	8,000	436	3,453	207,440
1897	418	524	7,996	9,342	241	3,304	276,738
1898	245	751	10,250	12,291	290	3,145	302,686
1899	59	1,083	11,930	10,269	408	2,521	347,234
1900	38	1,031	24,066	(b)	(b)	2,948	402,870
1901	(b)	527	16,000	(b)	(b)	2,061	443,397
1902	68	736	11,000	5,792	655	2,205	413,714
1903	83	698	8,063	(b)	527	2,448	376,177
1904	31	1,370	12,805	(b)	297	(b)	383,581
1905	84	1,562	11,449	(b)	210	2,148	352,479
1906	481	1,322	6,762	(b)	196	3,634	350,746

Year.	Gold Ore.	Iron Ore.	Lead Ore. (Galena)	Manganese Ore.	Tin Ore and Metal.	Tungsten Ore.
1895	222.0	(b)	1,346	1,240	3	12
1896	(b)	(b)	1,333	1,494	6	14
1897	17.0	(b)	2,180	1,652	9	29
1898	6.8	2,519	3,242	907	102	59
1899	13.0	15,078	3,468	2,949	30	55
1900	(d)2.6	19,803	3,620	1,970	81	49
1901	(d)2.0	21,599	445	904	31	90
1902	(d)2.0	19,914	1,651	(b)	24	234
1903	(d)1.3	15,200	830	30	(b)	228
1904	*Nil*	12,488	50	(b)	51	358
1905	*Nil*	3,200	291	(b)	20	290
1906	*Nil*	(b)	511	22	22	570

(a) From a report specially furnished THE MINERAL INDUSTRY by Señor Severiano Augusto da Fonseca Monteiro, Chief of the Department of Mines of the Ministerio das Obras Publicas except for 1904 and subsequent years, which are from official Government reports. (b) Not reported. (c) Consumed in the country. (d) *Kg.* fine metal.

RHODESIA.

The statistics of the mineral production of Rhodesia for the last 10 years are given in the subjoined table.

MINERAL AND METALLURGICAL PRODUCTION OF RHODESIA (a).

	Gold Ozs.	Value.	Silver. Ozs.	Lead. Tons. (b)	Coal. Tons. (b)		Gold. Ozs.	Value.	Silver. Ozs.	Lead. Tons. (b)	Coal. Tons. (b)
1899..	56,742	$ 999,620	112	.....	.....	1904..	267,737	$4 711,016	70,146	455	59,678
1900..	85,367	1,498,100	951	.....	.....	1905..	407,048	7,046,692	89,278	570	97,191
1901..	172,035	2,966,490	3,132	.....	.....	1906..	551,894	9,647,581	110,575	652	103,803
1902..	194,170	3,339,286	3,445	.....	.....	1907..	612,052	10,589,385	147,324	756	115,073
1903..	231,872	4,022,756	20,715	.....	.....	1908..	606,961	12,276,394	283,425	1,069	164,114

(a) From report of Colonel Seely, Under-Secretary of State for the colonies.
(b) Long tons.

RUSSIA.

The mineral and metallurgical production of Russia, according to official statistics especially reported to THE MINERAL INDUSTRY, is given in the subjoined tables. The latest available statistics are those for 1907.

MINERAL AND METALLURGICAL PRODUCTION OF RUSSIA. (a)
(In metric tons; one metric ton = 61.05 poods.)

Year.	Asbestos.	Chrome Ore.	Coal.	Copper.	Gold. (b)	Pig-iron.	Lead.	Manganese Ore.
1895	1,131	21,014	9,098,486	5,854	$24,198,383	1,452,138	411.9	203,081
1896	1,275	6,682	9,377,560	5,832	21,667,269	1,620,814	261.5	191,645
1897	1,016	13,433	11,202,750	6,940	22,194,664	1,880,130	450.1	263,115
1898	1,665	15,466	12,307,463	7,290	22,195,208	2,241,293	241.2	329,276
1899	2,693	19,146	13,974,376	7,533	22,396,315	2,708,752	321.8	659,302
1900	3,845	18,233	16,156,055	8,258	22,369,864	2,933,786	220.7	802,236
1901	4,398	22,169	16,526,652	8,467	22,763,967	2,866,779	156.0	522,395
1902	4,508	19,656	16,465,852	8,817	22,258,343	2,598,086	225.3	536,519
1903	5,264	16,421	17,868,515	9,232	24,147,222	2,487,783	106.3	414,334
1904	7,502	26,575	19,608,631	9,835	24,627,537	2,972,115	90.3	430,090
1905	5,896	27,051	18,727,766	8,515	20,521,587	2,628,101	700.2	508,635
1906	(c)	(c)	21,593,158	9,296	20,020,862	2,694,895	906.8	1,015,686
1907	(c)	(c)	25,741,321	15,930	26,518,253	3,041,570	(c)	995,282

Year.	Petroleum.	Phosphate Rock.	Platinum. (*Kg.*)	Pyrites.	Quicksilver.	Salt.	Silver. (*Kg.*)	Sulphur.	Zinc.
1895	6,290,000	6,327	4,414	11,042	434	1,540,195	7,887	190	5,030
1896	6,371,826	3,776	4,930	11,550	491	1,346,118	7,808	437	6,257
1897	6,945,127	5,917	5,601	19,390	616	1,561,895	4,779	574	5,874
1898	8,009,828	1,867	6,016	24,570	362	1,505,602	5,143	1,018	5,664
1899	8,517,608	16,863	5,962	23,250	362	1,679,726	4,419	451	6,326
1900	9,844,390	25,663	5,089	23,154	141	1,968,007	2,293	1,587	5,963
1901	10,925,471	21,276	6,371	30,732	363	1,705,924	1,088	2,489	6,104
1902	10,445,536	13,709	6,135	26,465	416	1,847,021	1,200	1,800	8,264
1903	9,759,214	14,635	6,009	22,780	362	1,658,938	1,152	281	9,894
1904	10,058,968	20,282	5,016	31,667	332	1,908,275	726	16	10,612
1905	7,505,637	(c)	5,250	30,689	318	1,844,678	2,965	(c)	7,911
1906	8,167,934	(c)	5,776	(c)	210	1,730,934	430	(c)	9,602
1907	9,098,931	(c)	5,903	(c)	130	(c)	(c)	(c)	10,409

(a) From official sources.
(b) The value of gold is taken at $20.67 per ounce.
(c) Not reported.

SOUTH AMERICA.

The following tables itemize the statistics of the production and the foreign commerce, or both, of mineral and metallurgical products of South American countries so far as available. No statistics later than those given in the tables have been published.

MINERAL AND METAL PRODUCTION OF BOLIVIA. (a)
(In metric tons.)

Year.	Antimony. Ore.	Bismuth.	Borate of Lime.	Cobalt. Ore.	Copper. (c)	Gold. (b)	Silver. (d)	Tin. Ore.	Tungsten. Ore.
1903.....	59	288	1,197	3.8	4,093	$33,810	89,063	18,425	68
1904.....	7	406	1,080	1.5	3,225	17,130	21,172	20,692	700
1905.....	17	593	2,146	.....	6,708	15,044	8,266	26,428	68
1906.....	.....	231	.....	.....	4,347	17,403	.....	29,374	.....

(a) From a British Consular report. (b) Reduced to U. S. currency. (c) Includes ingots, precipitate, matte and ore. (d) Includes ingots, ore and sulphide.

MINERAL EXPORTS OF BRAZIL. (a)
(In metric tons or dollars.) (d)

Year.	Agate.	Carbonado.	Copper Ore.	Diamonds.	Gold.	Manganese Ore.	Mica and Talc.	Monazite	Platinum. (Grams.)	Precious Stones. (b)	Rock Crystal.
1902	81	$49,611	234	$79,071	$636,739	157,295	11.0	1,205	.....	$4,332	.....
1903	74	66,888	316	62,248	684,389	161,926	7.0	3,299	1,315	8,247	.....
1904	54	32,063	610	34,975	611,198	208,260	14.0	4,860	2,122	12,505	35
1905	83	113,157	658	142,459	647,581	224,377	1.0	4,437	72,000	88,463	23
1906	121	319,743	1,484	340,137	771,611	121,331	6,123	4,351	*Nil*	141,395	37
1907	(c)	111,157	(c)	33,713	603,640	236,778	4,501	4,438	*Nil*	33,385	37

(a) As reported by the *Brazilian Review*. (b) Other than carbonado and diamonds. (c) Statistics not available. (d) The par exchange value of the *Mil Reis* in 1907 was $0.546 U. S. gold. Common exchange value was in 1902, $4.155; in 1903, $4.134; in 1904, $4.146; in 1905, $3.153; in 1906, $3.103; and in 1907, $3.301

MINERAL PRODUCTION OF CHILE (a).
(In metric tons.)

Year.	Borax.	Coal.	Cobalt Ore.	Copper.	Gold, *Kg.*	Guano.	Iodine.	Salt.	Silver, *Kg.*	Sodium Nitrate.	Sulphur.
1896		(c)	(d)	23,649	1,634	(f)	(e)	2,434	150,480	1,158,088	940
1897		(c)	(d)	21,128	1,538	(f)	(e)	5,867	140,732	1,148,096	664
1898		(c)	(d)	26,331	2,037	(f)	(e)	6,684	131,995	1,283,563	1,256
1899	86,992	(c)	(d)	25,719	2,060	(f)	274	9,937	129,503	1,389,823	989
1900	(b)	(c)	(d)	25,715	1,975	(f)	302	9,879	73,071	1,460,100	2,472
1901		(c)	(d)	30,155	1,100	(f)	269	10,099	70,237	1,273,800	2,516
1902		(c)	(d)	27,066	1,286	(f)	242	9,532	57,418	1,400,408	2,636
1903	16,879	827,112	290	29,923	994	11,134	387	16,264	28,552	1,444,920	3,560
1904	16,733	751,628	125	31,025	1,135	2,669	461	17,674	28,501	1,487,598	3,594
1905	19,612	793,927	28	29,126	1,055	19,380	564	12,108	16,315	1,669,806	3,470
1906	28,996	932,488	0.19	25,829	1,135	4,709	331	17,116	21,216	1,822,144	4,598
1907	28,374	832,612	.....	28,863	1,907	7,518	4,202	18,982	28,280	1,846,096	2,905

(a) From *Estadistica Minera de Chile*. (b) The combined output of the years 1894 to 1902 inclusive. (c) The combined output of Chile up to the end of 1902 is estimated at 20,650,000 tons. (d) The combined output of Chile up to the end of 1902 is estimated at 5941 tons. (e) Not reported. (f) The combined output of Chile up to the end of 1902 is estimated at 163,704 tons, valued at 5,041,560 pesos ($1,940,169).

MINERAL EXPORTS OF CHILE. (a)
(In metric tons.)

Year.	Borate of Lime.	Coal.(d)	Cobalt Ore.	Copper Matte.	Copper and Silver Matte.	Copper, Silver and Gold Matte.	Copper Ore.	Copper and Silver Ore.	Copper, Silver and Gold Ore.	Copper Ingots.	Gold Bullion, Kg.
1894	6,700	205,201	4.6	342	1,508	2.5	11,106	90.3	460	19,640	1,475.4
1895	4,425	195,115	13.4	417	664	15.3	6,963	84.4	2,012	20,042	1,184.5
1896	7,486	204,858	(b)	2,528	1,059	7.6	6,159	62.3	29,542	20,592	1,061.3
1897	3,154	243,968	6.0	2,519	904	(b)	3,396	161.8	(b)	19,011	1,131.7
1898	7,028	282,663	18.2	3,079	419	17.8	20,301	87.0	5,733	20,600	1,630.5
1899	14,951	241,995	55.	1,710	1,094	93.0	35,854	184.0	12,000	17,311	1,625.0
1900	13,177	325,042	26.8	4,838	1,918	241.8	20,213	238.5	360	20,340	1,871.1
1901	11,457	(b)	76.0	2,905	1,779	208.0	15,921	119.0	60	24,480	637.0
1902	14,327	(b)	464.0	2,094	(b)	220.0	22,622	133.0	2,000	21,197	762.0
1903	16,879	200,000	284.5	2,689	864	863.6	17,961	89.8	440	22,196	207.5
1904	16,733	213,262	125.0	472	460	1,092	21,899	88.3	1,152	26,442	397.0
1905	19,612	227,800	28.6	2,657	2,656	1,658	17,045	36.2	1,211	23,410	328.7
1906	28,435	1,935	Nil.	5,427	(b)	400	20,371	29	1,828	19,053	(e) 893

Year.	Gold Ore.	Iodine.	Lead and Silver in Bars.	Manganese Ore.	Mineral Specimens.	Silver Ore.	Silver and Gold Ore.	Silver Ingots, Kg.	Silver Lead Ore.	Silver-Sulp'de Ore.	Sodium Nitrate.
1894	192	323	87	47,994	$1,150	370	56	153,723	15	127	1,081,337
1895	270	144	93	24,075	2,800	2,137	113	148,747	21	99	1,220,427
1896	367	206	594	26,151	700	2,750	666	151,226	Nil.	160	1,111,757
1897	64	243	369	23,529	20,300	984	260	143,541	6	183	1,057,640
1898	8	235	13	20,851	1,400	284	269	139,756	12	290	1,294,227
1899	12	304	171	40,931	64,521	302	370	75,899	32	339	1,380,718
1900	129	318	14	25,715	3,550	225	217	45,623	1	172	1,465,935
1901	66	385	455	18,480	(b)	6,166	196	46,164	(b)	264	1,291,958
1902	115	244	99	12,990	(b)	114	610	31,812	161	176	1,330,598
1903	57	387	Nil.	17,110	292	55	1,216	(c) 10,857	102	17	1,443,286
1904	301	461	17	2,324	220	88	1,549	6,658	5	36	1,485,190
1905	103	564	0.2	1,323	18	10	1 052	2,348	0.6	31	1,668,976
1906	68	351	Nil.	35	1,705	145	983	(e) 4,077	(b)	(b)	1,760,805

(a) From *Estadistica Minera de Chile*. (b) Not reported. (c) Contains 500 kg. of gold. (d) There is no real exportation of Chilean coal for foreign consumption; that supplied to steam vessels is considered as exported. (e) Includes precipitate.

MINERAL AND METAL PRODUCTION OF PERU. (a)
(In metric tons.)

Year.	Bismuth.	Borate.	Coal.(b)	Copper.	Gold. Kg.	Lead.	Nickel. Kg.	Petroleum.	Quicksilver. Kg.	Silver. Kg.	Salt.	Sulphur.
1903...	.....	2,466	36,920	9,497	1,078.3	1,302	.....	37,079	.....	170,800	17,637	.....
1904...	.....	2,675	59,920	9,504	601.4	2,209	.....	38,683	.....	145,165	18,545	.....
1905...	12	1,954	75,338	12,213	776.6	1,476	1,778	49,700	1,554	191,476	21,039	.....
1906...	.....	2,598	79,969	13,474	1,247.0	2,568	.....	70,832	2,304	230,300	20,226	1,830
1907...	48	2,451	185,565	20,681	777.6	5,525	.....	100,184	1,500	207,810	21,592	2,030

(a) Reported by the Cuerpo de Ingenieros de Minas del Peru, in its *Boletin*. (b) Includes asphaltum and bituminous schist.

SPAIN.

The following tables record the mineral and metal production of Spain, as reported by official authorities:

MINERAL PRODUCTION OF SPAIN. (a)
(In metric tons.)

Year.	Aluminous Earths.	Antimony ore.	Arsenic.	Asphaltum.	Asphalt Rock.	Barytes.	Cement, Hydraulic.	Coal.	
								Anthracite.	Bituminous.
1896	320	54	271	1,285	1,117	345	130,738	14,895	1,852,947
1897	409	354	244	1,378	1,656	429	159,439	8,758	2,010,960
1898	505	130	111	2,354	2,383	364	164,862	20,105	2,414,127
1899	685	50	101	2,646	2,542	887	165,645	34,842	2,665,437
1900	420	30	150	2,331	4,193	833	185,811	68,427	2,514,545
1901	305	10	120	4,182	3,956	1,067	180,909	85,266	2,566,591
1902	337	67	*Nil.*	6,034	6,301	642	201,356	109,298	2,614,010
1903	381	42	1,088	4,675	6,277	507	245,294	108,959	2,887,652
1904	925	245	400	3,463	3,761	453	286,737	163,275	2,903,771
1905	221	77	1,140	5,805	5,725	290	296,605	159,517	2,912,466
1906	386	180	1,114	6,229	7,794	330	299,294	113,747	3,095,043
1907	1,209	205	1,500	8,643	8,219	314	329,926	164,498	3,531,337

Year.	Coal (*Continued*).		Coke.	Copper Ore.		Copper.			Fluor-spar.
	Lignitic.	Briquets.		Argentiferous.	Pyritic.	Fine.	Matte.	Precipitate.	
1896	55,413	343,432	288,523	(c) 157,365	2,200,919	6	16,378	29,873	3
1897	54,232	332,272	755,394	(c) 18,488	2,161,182	7	16,120	29,652	2
1898	66,422	369,418	768,151	203	2,299,444	593	16,024	29,703	5
1899	70,901	348,838	341,443	1,103	2,443,044	4	15,755	41,927	310
1900	91,133	341,156	381,000	2,006	2,714,714	5	18,159	29,652	4
1901	95,867	338,684	455,586	(b)	2,672,365	79	15,634	28,433	(b)
1902	84,242	331,957	404,503	878	2,617,776	(b)	.....	36,045	93
1903	104,232	339,120	433,780	3,056	2,796,733	(b)	.....	27,448	4,000
1904	100,773	307,630	432,726	(b)	2,524,513	(b)	8,117	29,494	(b)
1905	168,994	290,830	448,073	(b)	2,621,054	(b)	8,243	17,983	(b)
1906	189,048	311,328	435,808	(b)	2,888,778	(b)	9,068	19,200	70
1907	191,001	355,718	476,360	(b)	3,182,645	(b)	9,886	20,887	270

Year.	Gold and Silver Ore.	Iron Ore.		Iron and Steel.			Kaolin (China Clay).	Lead (Argentiferous).	
		Argentiferous.	Non-Argentiferous.	Pig.	Forged Iron.	Steel.		Ore.	Metal.
1896	854	3,581	6,762,582	100,786	53,793	68,126	1,240	182,565	84,802
1897	2,456	5,559	7,419,768	146,940	80,894	66,007	6,294	186,692	91,258
1898	555	24,190	7,197,047	113,492	65,900	50,362	5,445	244,068	88,981
1899	(d) 1,110	17,139	9,397,733	113,071	40,332	112,982	2,790	184,906	70,874
1900	(d) 1,300	26,348	8,675,749	91,126	54,307	144,355	3,794	182,016	74,341
1901	(d) 1,595	27,726	7,906,517	135,600	47,085	121,023	2,220	207,188	73,895
1902	(d) 1,764	24,361	7,904,555	330,747	.....	163,564	3,412	227,645	74,370
1903	(d) 2,681	90,996	8,304,153	390,284	.....	199,642	2,578	179,858	56,667
1904	(b)	122,109	7,964,748	283,819	50,858	186,705	1,700	177,104	57,956
1905	(b)	152,027	9,007,245	305,462	11,366	223,545	720	160,381	56,361
1906	(b)	126,445	9,448,533	315,309	6,035	274,280	610	158,425	53,856
1907	(b)	(b)	9,896,178	355,240	14,767	310,125	640	165,289	51,430

Year.	Lead (Non-Argentiferous).		Manganese Ore.	Mineral Paints. (Ocher).	Phosphate Rock.	Pyrites (Iron).	Pyrites (Arsenical).	Quicksilver.	
	Ore.	Metal.						Ore.	Metal.
1896.....	104,160	82,215	38,265	212	770	100,000	(b)	34,959	1,524
1897.....	110,496	75,112	100,566	200	2,084	100,000	(b)	32,378	1,728
1898.....	150,472	78,370	102,228	200	4,500	70,265	230	31,361	1,691
1899.....	123,753	91,739	104,974	100	3,510	107,386	(b)	32,144	1,361
1900.....	131,437	98,189	112,897	58	4,170	34,638	515	30,216	1,095
1901.....	174,376	95,399	60,325	164	4,220	33,953	1,328	23,367	754
1902.....	100,403	103,190	46,069	(b)	1,150	145,173	5,648	26,037	1,425
1903.....	108,660	118,422	26,194	(b)	1,124	155,739	7,996	30,370	968
1904.....	93,230	127,906	18,732	(b)	3,305	161,841	3,510	27,185	1,130
1905.....	105,113	129,332	26,020	(b)	1,370	179,079	4,790	26,485	853
1906.....	105,095	131,614	62,822	164	1,300	189,243	2,434	28,965	1,568
1907.....	103,632	135,066	41,504	114	3,547	225,830	3,423	28,789	1,212

Year.	Salt.	Silver.		Soapstone.	Sulphur.		Tin Ore (Dressed).	Topaz. Kg.	Tungsten Ore.	Zinc.		
		Ore.	Metal, Kg.		Crude rock.	Refined.				Ore.	Spelter	Sheets.
1896.....	521,751	1,230	64,554	756	26,204	1,800	(c)2,348	80	31	64,828	3,485	2,648
1897.....	508,606	982	71,168	3,601	18,805	3,500	(c)2,378	44	10	73,848	3,907	2,337
1898.....	479,358	967	76,295	2,613	34,943	3,100	4	90	37	99,836	4,300	1,731
1899.....	598,108	764	88,409	4,844	58,922	1,100	57	44	151	119,710	4,100	2,084
1900.....	450,041	742	140,457	8,109	64,364	750	47	95	1,958	86,158	2,855	2,756
1901.....	345,063	391	94,977	4,880	49,856	610	115	310	6	119,706	2,573	2,781
1902.....	426,434	175	96,975	542	15,442	450	12,762	Nil.	11	127,618	5,569	(b)
1903.....	427,394	231	112,978	3,725	38,573	1,680	330	90	Nil.	154,126	5,134	(b)
1904.....	543,658	303	117,418	5,165	40,389	605	229	.....	60	156,329	5,887	2,913
1905.....	493,451	540	123,607	4,364	38,153	610	209	.....	375	160,561	6,184	2,936
1906.....	541,978	470	126,424	3,609	28,965	700	86	171	430	170,384	6,209	2,639
1907.....	605,895	702	127,435	13,875	27,054	3,612	315	266	386	191,853	6,144	2,485

(a) Figures are from *Estadistica Minera de España* except for 1896 and 1898, which were from the official *Reports of the Junta Superior Facultativa de Minas*, Madrid. (b) Not reported. (c) Represents non-argentiferous copper ore. (d) Gold ore only. (e) Undressed tin ore.

SWEDEN.

The official statistics of mineral production, imports and exports are summarized in the following tables, 1907 being the latest year for which the reports of imports and production have been published:

MINERAL PRODUCTION OF SWEDEN. (a)
(In metric tons.)

Year.	Alum.	Coal.	Copper.			Feldspar.	Gold. *Kg.*
			Ore.	Ingot.	Sulphate.		
1897	131	224,343	25,207	289	1,315	19,298	113.3
1898	153	236,277	23,335	235	1,165	20,737	125.9
1899	164	239,344	22,334	179	1,287	16,017	106.2
1900	167	252,320	22,725	136	1,265	15,228	88.5
1901	121	271,509	23,660	137	1,224	13,502	62.7
1902	132	304,733	30,095	178	1,257	17,960	94.3
1903	140	320,390	36,687	776	1,171	19,392	50.6
1904	125	320,984	36,834	533	1,248	18,021	60.9
1905	139	322,384	39,255	1,385	1,029	19,224	55.0
1906	167	296,980	19,655	1,209	562	21,014	20.3
1907	131	305,338	21,957	1,577	782	20,244	28.1

Year.	Iron and Steel.					Steel.		
	Ore.	Pig.	Blooms.	Bars, Rods, Sheets, etc.	Iron Sulphate	Bessemer.	Basic.	Crucible.
1897	2,086,119	538,197	189,633	304,537	232	107,679	165,836	691
1898	2,302,546	531,766	198,923	299,846	124	102,254	160,706	1,013
1899	2,434,606	497,727	195,331	328,999	105	91,896	179,357	1,225
1900	2,607,925	526,868	188,455	324,604	183	91,065	207,418	1,121
1901	2,793,566	528,375	164,850	269,507	140	77,231	190,877	1,088
1902	2,896,208	538,113	186,076	(b)	127	84,014	201,311	1,091
1903	3,677,520	506,825	192,342	325,200	62	84,229	232,578	1,105
1904	4,083,945	528,525	189,246	324,676	148	78,577	253,532	1,162
1905	4,364,333	539,437	182,640	356,898	144	78,204	288,675	1,319
1906	4,501,656	604,789	178,298	381,118	170	84,633	311,435	1,457
1907	4,478,917	615,778	174,405	403,994	159	77,036	341,893	1,257

Year.	Lead.		Manganese Ore.	Pyrites.	Silver-lead Ore.	Silver. *Kg.*	Sulphur.	Zinc Ore.
	Ore.	Pig.						
1897	99	1,480	2,749	517	10,068	2,218	(b)	56,636
1898	50	1,559	2,358	386	6,743	2,033	80	61,827
1899	35	1,606	2,622	150	5,730	2,290	(b)	65,150
1900	85	1,424	2,651	179	5,300	1,927	70	61,044
1901	56	988	2,271	*Nil.*	11,366	1,557	(b)	48,630
1902	63	843	2,850	*Nil.*	9,378	1,365	74	48,783
1903	25	678	2,244	7,793	9,792	1,005	(b)	62,927
1904	55	589	2,297	15,957	8,187	651	35	57,634
1905	40	576	1,992	20,762	5,397	606	.....	56,995
1906	37	753	2,680	21,827	1,938	938	.....	52,552
1907	33	813	4,334	27,113	1,987	929	.....	50,884

(a) From *Bidrag till Sveriges Officiela Statistik Bergshandteringen.* (b) Not reported.

MINERAL IMPORTS OF SWEDEN. (a)
(In metric tons or dollars; 1 krone—27 cents).

Year.	Asbestos. (c)	Asphalt.	Barytes.	Borax.	Boric Acid.	Bromine and Bromides. *Kg.*	Cement.	Chalk. White. Unground *Hectoliters.*	Coal.
1896	116	4,092	298	128	73	4,234	2,901	6,148	1,991,760
1897	119	5,458	270	175	56	5,549	1,826	14,368	2,240,247
1898	112	5,409	299	196	75	5,401	1,656	7,016	2,292,451
1899	567	6,286	292	190	65	4,914	1,363	16,079	3,047,618
1900	763	5,676	411	194	66	6,084	1,941	12,059	3,033,885
1901	178	4,524	295	253	68	6,602	2,868	13,569	2,793,309
1902	213	5,779	.....	242	71	7,278	9,822	11,583	2,911,286
1903	217	5,957	.....	240	71	7,419	11,145	41,868	3,192,990
1904	356	6,243	.....	299	77	10,128	10,526	10,115	3,367,826
1905	140	4,760	264	294	82	18,788	10,999	13,305	3,297,485
1906	287	7,134	559	321	79	9,908	13,136	10,777	3,718,884
1907	672	8,213	610	490	85	6,784	17,801	8,107	4,126,785

Year.	Copper, also Alloys of Copper.	Emery.	Graphite.	Gypsum	Iron (crude).	Lead.	Litharge.	Phosphorus. *Kg.*	Platinum. *Kg.*	Porcelain.
1896	4,037	104	135	4,940	34,549	1,911	150	52,482	34	327
1897	4,944	128	158	7,260	89,606	2,098	199	57,972	63	362
1898	5,227	131	167	7,979	76,832	2,139	160	66,466	49	298
1899	4,740	125	162	6,457	68,909	2,125	177	59,989	99	346
1900	4,745	136	213	6,794	82,957	2,067	148	67,557	59	382
1901	5,153	169	180	6,589	66,131	1,991	165	70,672	172	386
1902	6,590	147	(b)	6,754	43,828	2,509	172	68,441	130	.....
1903	6,109	132	(b)	8,795	49,411	2,644	237	112,659	116	.....
1904	7,367	221	(b)	8,868	90,102	2,849	213	47,421	84	396
1905	6,481	271	(b)	11,270	87,843	2,823	205	69,526	105	416
1906	8,899	284	(b)	13,496	108,193	3,457	255	79,048	133	486
1907	6,849	336	(b)	15,037	115,174	2,352	375	77,936	1	424

Year.	Potassium				Quicksilver. *Kg.*	Salt.		Silver.	
	Chloride.	Cyanide *Kg.*	Hydrate.	Carbonate		Crude.	Refined.	Scrap and Mfrs. *Kg.*	Specie.
1896	241	2,122	285	1,933	5,194	84,629	3,673	7,375	$204,691
1897	363	2,922	1,381	1,432	3,125	87,050	3,055	20,557	136,823
1898	259	2,604	1,451	1,112	2,631	85,246	2,188	21,696	191,766
1899	225	2,313	1,266	1,231	4,210	98,417	3,166	11,565	156,707
1900	364	2,221	1,915	1,257	3,629	70,302	3,098	11,559	62,315
1901	260	2,658	1,435	1,266	5,958	79,038	3,072	7,476	78,416
1902	222	2,950	1,720	1,238	4,866	82,439	3,037	4,853	74,826
1903	245	3,294	2,034	1,150	5,043	88,139	3,419	11,259	90,366
1904	214	3,237	2,234	1,184	5,768	84,237	4,615	19,034	86,891
1905	1,296	3,437	2,251	1,133	4,609	87,677	3,889	11,067	82,020
1906	1,986	4,106	2,486	1,082	5,535	88,341	3,700	15,253	93,990
1907	1,840	4,150	2,484	1,269	8,930	75,397	18,821	21,245	52,463

Year.	Sodium.				Sulphur.	Sulphuric Acid.	Tin.		Zinc.
	Carbonate.	Hydrate.	Nitrate. (d)	Sulphate. (e)			Salts—*Kg.*	Block.	
1896	11,425	908	12,518	8,486	11,369	615	4,437	551	2,275
1897	14,625	625	12,531	11,384	9,723	1,418	3,823	541	2,551
1898	11,917	575	15,419	11,544	10,337	1,742	3,874	595	3,030
1899	13,323	929	15,006	15,140	13,505	2,558	5,404	486	2,829
1900	12,680	1,038	14,245	15,590	20,152	2,472	3,243	630	2,912
1901	13,669	800	17,614	15,494	20,715	1,950	2,334	541	2,900
1902	.....	1,623	15,553	18,924	23,002	1,887	1,652	644	3,255
1903	.....	1,426	20,616	16,120	24,577	2,620	1,467	655	3,312
1904	11,898	2,112	19,776	17,596	18,248	2,001	1,460	719	3,705
1905	13,592	1,489	23,183	17,115	18,631	3,424	1,727	597	3,780
1906	14,974	1,478	27,174	19,948	22,745	2,535	2,102	819	4,484
1907	14,970	1,628	26,181	21,486	25,456	2,628	6,117	895	5,372

MINERAL EXPORTS OF SWEDEN. (a)
(In metric tons or dollars; 1 krone=27 cents.)

Year.	Alum.	Ammonium Sulphate	Antimony, Crude-Kg	Asbestos. Kg.	Cement.	Coal.	Copper.		Graphite.
							Ore.	Copper & Alloys.	
1896	40	100	800	2,040	22,991	141	1,094	1,911	3,500
1897	54	180	800	1,348	27,112	74	(b)	933	7,215
1898	32	36	4,700	1,055	28,676	496	1,102	1,346	9,108
1899	26	2	2,600	2,812	31,101	762	315	1,230	16,664
1900	24	2	4,600	2,436	42,564	1,108	448	2,012	17,719
1901	56	156	1,800	2,179	17,794	716	602	1,243	16,761
1902	20	174	4,090	1,864	19,499	866	845	1,516	5,420
1903	22	*Nil*	3,473	15,357	21,319	509	1,555	1,858	8,744
1904	9	219	3,810	16,339	27,509	605	749	1,396	(b)
1905	12	445	3,147	2,356	38,504	425	2,137	2,054	(b)
1906	11	30	4,584	1,510	45,960	1,352	1,841	2,662	(b)
1907	7	98	4,485	2,167	18,053	2,925	882	2,915	(b)
1908	9	218	4,188	1,335	34,164	1,235	1,114	3,450	(b)

Year.	Gypsum and Mfres.	Iron and Steel.		Lead and Mfres.	Peat.	Phosphorus. Kg.	Potassium Chloride.
		Ore.	Unwrought.				
1896	9	1,150,695	304,138	1,182	1,452	1,510	254
1897	9	1,400,801	279,525	1,473	1,816	1,527	463
1898	27	1,439,860	301,192	570	1,616	4,085	506
1899	8	1,628,011	320,742	818	1,979	1,890	335
1900	10	1,619,902	304,175	1,209	3,843	879	931
1901	55	1,761,257	268,143	1,028	3,064	1,254	708
1902	117	1,729,000	(f) 73,403	546	3,620	1,290	1,114
1903	119	2,828,000	(f) 70,788	333	3,217	300	790
1904	162	3,065,522	(f) 88,124	275	4,212	1,994	1,266
1905	156	3,316,626	(f)120,987	512	5,157	34,388	1,499
1906	6	3,661,218	(f)112,719	531	6,531	700	(b)
1907	16	3,521,717	(f)201,643	519	6,524	(b)	1,753
1908	33	3,654,268	159,095	496	5,558	(b)	1,364

Year.	Salt, Refined Kg.	Silver, Bullion. Kg.	Soda.	Sulphur.	Tin.		Zinc.	
					Block.	Mfres-Kg.	Ore.	Crude and Mfrs.
1896	830	819	772	9	18.9	2,996	41,401	184
1897	1,424	329	686	11	25.6	7,113	44,425	135
1898	216	130	509	11	20.8	1,263	49,597	184
1899	110	367	227	68	8.8	1,033	45,634	157
1900	407	296	238	20	21.5	1,521	40,879	156
1901	1,556	179	237	12	20.4	8,110	41,248	101
1902	1,945	110	621	147	25.5	1,003	43,813	68
1903	*Nil*	484	10	217	43.3	3,893	45,389	351
1904	1,883	115	45	4	45.6	3,479	44,259	332
1905	*Nil*	10	403	4	33.9	654	51,765	295
1906	8,652	77	463	12	51.0	353	45,370	410
1907	4,452	160	39	29	67.9	2,518	41,236	526
1908	14,300	136	114	34	53.9	260	38,543	906

(a) From *Bidrag till Sveriges Officiella Statistik.* (b) Not reported. (c) Includes crude and manufactures. (d) Includes a small quantity of potassium nitrate. (e) Includes sodium bisulphate. (f) Includes only crude or ballast iron.

UNITED KINGDOM.

The statistics of the mineral production, imports and exports, according to official reports, are given in the subjoined tables.

MINERAL AND METALLURGICAL PRODUCTION OF THE UNITED KINGDOM. (*a*)
(In metric tons.)

Year.	Alum Shale.	Arsenious Acid.	Arsenical Pyrites.	Barytes.	Bauxite.	Chalk.	Clay. (*c*)	Coal.
1897	621	4,232	13,347	23,087	13,540	3,920,183	12,908,479	205,364,010
1898	13,835	4,241	11,272	22,581	12,600	4,366,782	14,974,290	205,287,388
1899	5,913	3,890	13,735	25,059	8,137	4,752,982	15,305,895	223,616,279
1900	1,329	4,146	9,727	29,937	5,871	4,444,765	14,279,181	228,772,886
1901	4,019	3,416	2,620	26,844	10,357	4,399,043	14,393,196	222,614,981
1902	5,755	2,165	842	23,986	9,192	4,466,004	15,549,002	230,728,562
1903	3,337	916	58	24,659	6,226	4,541,494	16,460,526	234,019,821
1904	6,636	992	44	26,748	8,839	4,509,768	16,210,734	236,130,373
1905	7,245	1,552	651	29,528	7,417	4,608,153	15,376,910	239,906,999
1906	9,605	1625	650	35,586	6,760	4,825,299	12,450,213	255,067,622
1907	10,063	1,497	706	30,828	7,658	4,855,857	15,065,141	272,097,858
1908	.....	1,949	3,216	32,586	11,904	.....	.....	.....

Year.	Copper.		Fluorspar.	Gold.		Gravel and Sand.	Gypsum.	Bog Ore (*c*)
	Ore and Precipitate.	Fine.		Ore.	Bullion. *Kg.*			
1897	7,470	526	302	4,589	63.2	1,378,496	184,287	7,238
1898	9,277	650	57	715	12.3	1,652,701	199,174	5,505
1899	8,452	647	796	3,096	103.5	1,800,208	215,974	4,390
1900	9,643	777	1,472	21,135	437.6	1,867,211	211,436	4,221
1901	6,903	541	4,232	16,641	194.5	1,990,926	204,045	2,649
1902	6,210	490	6,388	20,432	180.0	2,100,329	228,264	4,983
1903	6,977	545	12,102	29,057	171.0	2,281,389	223,426	4,156
1904	5,552	501	18,450	23,574	610.7	2,275,426	237,749	4,616
1905	7,267	727	38,606	16,237	169.0	2,277,486	259,596	3,256
1906	7,382	(*b*)	36,860	17,662	(*b*)	2,404,857	228,627	5,512
1907	6,867	(*b*)	40,873	13,186	(*b*)	2,438,798	247,537	6,391
1908	5,526	(*b*)	25,395	7,246	(*b*)	.....	193,543	.....

Year.	Iron.		Lead.		Manganese Ore.	Mineral Paints.	Oil Shale.	Phosphate of Lime.
	Ore.	Pig.	Ore.	Pig.				
1897	14,008,484	4,942,679	35,903	26,988	609	14,653	2,259,325	2,032
1898	14,403,769	4,928,347	33,513	25,761	235	20,144	2,172,201	1,575
1899	14,602,711	4,992,468	31,494	23,929	422	16,575	2,246,197	1,469
1900	14,257,344	4,743,172	32,487	24,762	1,384	15,448	2,318,736	630
1901	12,475,700	4,158,745	33,084	20,361	1,673	14,780	2,392,812	71
1902	13,641,459	4,470,420	25,000	17,988	1,299	17,235	2,141,355	87
1903	13,935,748	4,573,202	26,993	20,278	831	14,377	2,041,851	71
1904	13,994,670	4,596,803	26,796	20,155	8,896	16,307	2,370,391	59
1905	14,824,183	(*f*) 9,746,221	28,091	20,977	14,592	16,468	2,536,734	*Nil.*
1906	15,748,412	(*f*) 9,939,211	30,710	22,693	23,126	14,437	2,586,851	*Nil.*
1907	15,983,310	(*f*) 9,850,953	33,053	20,000(*g*)	16,356	14,927	2,732,968	32
1908	.....	.....	29,561	18,000(*g*)	6,408	.....	.....	.....

Year.	Pyrites.	Salt.	Silica. (chert and flint.)	Silver. *Kg.*	Stone.			
					Granite.	Limestone.(*d*)	Sandstone.	Slate.
1897	10,752	1,933,949	95,209	7,750	1,876,880	11,179,580	5,043,535	618,941
1898	12,302	1,908,723	83,370	6,575	1,905,830	12,172,267	5,325,988	679,461
1899	12,426	1,945,531	69,955	5,999	4,785,284	12,499,736	5,296,026	650,077
1900	12,484	1,873,601	78,971	5,904	4,709,997	12,099,940	5,101,868	595,428
1901	10,405	1,812,180	132,700	5,452	5,131,787	11,363,202	5,199,234	496,756
1902	9,315	1,893,881	100,938	4,560	5,554,696	12,368,196	5,571,121	525,665
1903	9,794	1,917,274	74,355	5,440	5,512,605	12,419,120	5,496,312	540,143
1904	10,452	1,921,899	66,300	4,967	6,084,642	12,235,825	5,391,265	572,181
1905	12,381	1,920,149	71,808	5,212	6,052,210	12,701,808	5,729,799	523,892
1906	11,318	1,996,593	69,300	.....	6,264,402	12,962,725	5,315,328	500,546
1907	10,906	2,016,409	54,523	.....	5,765,262	12,709,288	5,092,246	450,651

Year.	Strontium Sulphate	Tin.		Tungsten Ore.	Uranium Ore.	Zinc.	
		Ore, Dressed.	Block.			Ore.	Spelter.
1897	15,227	7,234	4,524	127	30	18,586	7,162
1898	13,148	7,498	4,722	331	26	23,929	8,711
1899	12,831	6,494	4,077	96	7	23,505	8,837
1900	9,270	6,911	4,337	9	42	25,070	9,214
1901	16,923	7,407	4,634	21	80	23,967	8,555
1902	32,799	7,681	4,462	9	53	25,462	9,275
1903	23,209	7,500	4,351	276	6	25,287	9,430
1904	18,460	6,849	4,198	164	*Nil.*	28,097	10,427
1905	14,523	7,316	4,540	174	105	24,025	9,023
1906	14,338	6,376	.....	267	11	23,189	(*b*)
1907	10,917	7,192	.....	327	72	20,402	.....

(*a*) From *Mineral Statistics of the United Kingdom*. (*b*) Not reported. (*c*) Bog ore, which is raised in Ireland, is an ore of iron, used principally for purifying gas. (*d*) Does not include chalk. (*e*) Includes China clay, potters' clay, and fuller's earth. (*f*) Includes production from imported ore. (*g*) Estimated.

MINERAL IMPORTS OF THE UNITED KINGDOM. (*a*)

(In metric tons or dollars; £1=$5.)

Year.	Alkali.	Asphaltum.	Borax.	Coal, Coke and Pat. Fuel.	Copper.			Iron and Steel.		
					Ore.	Regulus and Precipitate.	Wrought, Unwrought and Old.	Iron Ore.	Pig Iron.	Scrap.
1897	11,557	44,541	(*b*)	9,605	83,916	90,008	62,055	6,064,179	(*c*)160,531	20,735
1898	12,179	46,398	1,255	11,191	91,141	76,201	70,018	5,555,893	(*c*)162,075	24,619
1899	12,078	59,073	3,076	1,777	130,611	84,015	60,502	7,168,061	(*c*)174,159	32,427
1900	16,360	53,061	15,667	10,112	102,365	89,123	72,223	6,398,639	178,199	31,667
1901	(*c*)13,429	74,694	15,710	7,685	102,503	93,338	68,809	5,637,670	198,536	44,721
1902	(*c*)26,292	65,896	13,390	3,331	90,007	74,884	92,349	6,542,793	226,708	39,584
1903	(*c*)14,321	(*b*)	11,959	3,535	85,644	77,884	64,591	6,417,188	132,364	17,051
1904	(*c*)14,325	(*b*)	16,012	2,812	80,771	67,739	90,717	6,198,368	132,494	19,326
1905	(*c*)16,593	(*b*)	11,552	49,277	94,198	70,235	71,294	7,172,171	128,183	23,569
1906	(*c*)14,070	(*b*)	16,955	49,269	97,789	76,073	75,487	7,634,839	90,674	36,559
1907	(*c*)15,579	(*b*)	17,551	19,135	105,409	73,101	84,035	7,462,804	104,950	27,404

Year.	Iron and Steel. (*Continued.*)								Lead.	
	Puddled and Wrought.	Sheets and Plates.	Rails.	Strips and Wire Rods.	Nails, Screws, Rivets, Bolts.	Steel Ingots, Blooms, Billets, etc.	Steel Bars, Shapes Beams Pillars	Mnfrs. Unenumerated (*h*)	Ore.	Pig and Sheet.
1897	(*f*)	(*f*)	(*g*)	(*f*)	(*f*)	40,628	(*g*)	$27,894,295	32,818	170,121
1898	(*f*)	(*f*)	(*g*)	(*f*)	(*f*)	40,875	(*g*)	33,379,160	44,457	197,591
1899	(*f*)	(*f*)	(*g*)	(*f*)	(*f*)	78,257	(*g*)	39,527,075	30,263	201,551
1900	189,891	(*f*)	38,636	(*f*)	(*f*)	182,210	94,667	17,861,660	21,566	198,416
1901	102,811	(*f*)	55,809	(*f*)	(*f*)	185,810	124,648	18,252,435	29,944	221,549
1902	178,425	(*f*)	48,942	(*f*)	45,095	285,494	129,743	14,424,835	25,838	235,522
1903	196,084	73,079	74,939	35,574	51,888	278,441	343,259	9,007,990	18,923	232,939
1904	109,289	69,552	40,438	38,214	50,649	531,069	219,510	8,630,380	8,748	250,452
1905	110,576	69,831	34,439	60,318	55,331	613,612	148,995	8,783,995	(*i*)27,649	233,214
1906	111,062	83,747	11,900	61,288	57,071	493,805	149,363	7,824,405	(*i*)30,795	211,577
1907	83,145	57,280	19,337	56,110	51,863	332,442	90,327	3,731,275	(*i*)32,549	207,970

Year.	Manganese Ore.	Mica, Sheet.	Mica and Talc.	Paraffin.	Petroleum. *Liters.*	Phosphate Rock.	Platinum Wrought and Unwrought. *Kg.*	Potassium Nitrate.	Pyrites of Iron and Copper.
1897	158,825	412	1,683	39,284	842,920,307	330,335	2,257	16,744	633,009
1898	156,390	517	1,398	48,104	829,995,751	334,884	3,389	13,323	665,544
1899	261,740	519	6,025	54,712	908,107,248	426,830	5,404	12,635	712,393
1900	270,098	469	7,952	50,033	965,167,850	361,309	5,027	12,798	752,605
1901	195,736	(*b*)	7,117	42,643	960,650,967	360,568	4,917	12,115	664,041
1902	237,066	1,078	6,127	52,023	1,078,095,152	370,697	3,027	11,526	620,948
1903	235,574	(*b*)	(*b*)	49,163	1,299,570,025	398,997	(*b*)	9,425	747,714
1904	208,458	(*b*)	(*b*)	42,882	1,373,488,176	425,978	(*b*)	12,277	754,722
1905	289,827	(*b*)	(*b*)	41,247	1,364,301,583	427,762	(*b*)	8,260	709,926
1906	314,016	(*b*)	(*b*)	44,673	1,130,667,737	450,058	(*b*)	10,125	771,473
1907	298,352	(*b*)	(*b*)	46,542	1,382,595,355	512,601	(*b*)	10,719	781,447

Year	Quicksilver.	Silver Ore. (*d*)	Sodium Nitrate.	Sulphur.	Tin.		Zinc.		
					Ore.	Block, Ingot, Bars or Slabs.	Ore.	Spelter.	Mnfres.
1897	1,862	$7,149,210	107,525	22,811	5,345	27,214	25,238	70,929	21,395
1898	1,856	5,729,525	132,412	19,642	5,710	20,665	53,945	78,761	21,613
1899	1,759	5,162,750	163,387	21,906	6,324	27,603	38,143	71,068	21,521
1900	1,113	5,154,430	143,461	22,993	7,449	33,648	42,755	61,504	21,751
1901	1,202	5,309,920	108,822	22,440	10,690	35,397	38,660	68,633	21,343
1902	1,129	5,383,515	116,791	23,863	12,255	35,713	45,312	89,688	21,717
1903	1,187	6,596,045	118,582	21,313	12,473	36,076	41,009	86,539	23,118
1904	1,130	8,271,480	122,454	17,629	15,734	39,932	54,438	90,088	22,788
1905	1,158	10,426,570	106,107	18,163	(*b*)	40,391	(*i*)23,909	92,261	20,013
1906	1,320	10,532,020	110,222	22,704	21,003	44,306	(*i*)22,824	95,203	19,664
1907	1,341	11,224,650	115,716	15,730	21,205	44,505	(*i*)20,082	90,756	20,163

(*a*) From *Accounts Relating to Trade and Navigation of the United Kingdom.* (*b*) Not reported. (*c*) Classified as soda compounds since 1901. (*d*) Includes the value of silver in argentiferous ore and metal. (*e*) Includes puddled iron. (*f*) Not separately enumerated. (*g*) Former returns not available (*h*) Prior to 1900 many manufactures were not reported separately. (*i*) *Mines and Quarries*, Part I, 1907.

MINERAL EXPORTS OF THE UNITED KINGDOM—DOMESTIC PRODUCTS. (*a*)

(In metric tons or dollars; £1= $5.)

Year.	Bleaching Materials.	Cement.	Coal.	Coke.	Patent Fuel.	Supplied to Steamers.	Coal Products (*c*)
1897	(*b*)	398,023	35,919,965	993,980	(*b*)	10,623,050	$8,340,420
1898	(*b*)	331,648	35,619,365	782,053	(*b*)	11,444,431	7,624,740
1899	(*b*)	359,273	41,839,217	881,172	(*b*)	12,422,429	7,712,965
1900	57,478	365,742	46,845,739	1,001,131	(*b*)	11,940,353	9,058,220
1901	46,912	318,216	42,547,114	820,594	1,098,459	13,804,222	5,756,265
1902	40,939	308,104	43,849,591	669,664	1,067,060	15,390,485	5,991,025
1903	49,415	406,388	45,669,258	728,957	970,449	17,068,646	7,290,825
1904	35,289	390,736	46,995,636	779,060	1,257,589	17,465,954	6,879,400
1905	42,526	463,863	48,236,334	786,498	1,126,190	17,674,484	6,742,455
1906	45,510	668,461	56,489,367	828,266	1,399,244	18,887,656	7,226,790
1907	48,856	777,741	64,618,562	1,097,185	1,504,587	18,916,698	7,726,685

Year.	Copper.				Iron.				
	Ingot.	Mixed or Yellow Metal.	Mfres.	Sulphate.	Ore.	Pig.	Scrap.	Cast Iron and Mfres.	Wrought Iron, Shapes and Mfres.
1897	21,252	11,192	15,275	60,326	(*d*)	(*e*)1,219,958	99,259	(*b*)	170,285
1898	27,102	10,452	13,765	52,573	(*d*)	(*e*)1,058,973	86,602	(*b*)	152,911
1899	32,449	7,038	11,231	40,822	(*d*)	(*e*)1,401,365	118,262	(*b*)	161,679
1900	18,300	8,940	10,765	43,601	(*d*)	(*e*)1,450,365	96,567	(*b*)	159,677
1901	26,935	9,252	11,156	36,601	(*d*)	(*e*) 852,609	86,559	(*b*)	119,962
1902	21,658	13,314	14,075	43,995	4,062	1,120,207	104,890	(*b*)	(*b*)
1903	23,723	14,425	16,975	54,307	4,534	1,082,426	143,929	62,249	217,139
1904	14,791	16,704	18,467	71,367	6,706	823,909	166,010	49,004	178,233
1905	21,232	9,959	22,128	55,219	14,664	997,001	151,619	49,193	186,340
1906	19,778	7,149	16,195	43,670	13,415	1,670,753	180,547	54,876	203,521
1907	25,652	7,994	16,676	46,049	15,538	1,978,350	162,295	43,218	215,159

Year.	Iron. (*Continued.*)								
	Rails.	Wire and Mfres. of.	Plates and Sheets.	Galvanized Sheets.	Black Plates for Tinning.	Tinned Plates.	Steel Ingots, Billets, Blooms, etc	Steel Shapes Beams and Pillars.	Total Iron and Steel and Mfres. of.
1897.....	(*f*)795,983	52,471	120,868	231,319	59,663	276,260	304,249	(*b*)	3,750,122
1898.....	(*f*)619,976	44,954	102,638	230,219	59,289	255,797	290,182	(*b*)	3,299,326
1899.....	(*f*)601,266	50,041	111,773	242,167	86,936	260,735	333,837	(*b*)	3,777,098
1900.....	379,939	39,104	39,157	251,203	66,810	278,338	313,383	(*b*)	3,602,083
1901.....	474,073	48,107	36,418	254,290	52,217	275,661	217,236	(*b*)	2,944,083
1902.....	(*b*)	(*b*)	(*b*)	336,572	58,245	317,201	306,152	(*b*)	3,529,223
1903.....	613,741	60,900	165,672	357,665	66,279	297,485	13,427	159,330	3,621,635
1904.....	533,895	61,894	154,774	391,608	63,467	365,262	4,324	176,232	3,315,047
1905.....	555,390	82,519	207,866	413,533	69,937	360,630	8,735	219,491	3,781,059
1906.....	470,652	96,641	279,459	450,221	66,749	381,421	11,924	311,231	4,763,868
1907.....	440,576	103,180	305,399	476,838	72,675	411,814	13,705	344,135	5,249,028

Year.	Lead. Pig and Mfres.	Salt.	Sodium.				Tin. Block.	Zinc.		
			Soda Ash.	Carbonate and Bicarbonate.	Hydrate.	Sulphate.		Ore.	Spelter.	Mfres.
1897.....	40,911	660,477	(*g*)252,736	(*h*)	(*h*)	(*h*)	5,050	6,072	6,951	1,047
1898.....	38,684	698,882	(*g*)191,578	(*h*)	(*h*)	(*h*)	5,557	6,483	7,577	1,237
1899.....	40,923	638,213	(*g*)193,492	(*h*)	(*h*)	(*h*)	4,785	8,171	5,492	1,249
1900.....	36,576	556,704	(*g*)185,783	(*h*)	(*h*)	(*h*)	5,713	13,913	7,136	1,159
1901.....	38,166	627,078	58,412	22,161	50,624	26,057	5,584	13,981	7,512	1,256
1902.....	33,537	624,752	59,894	24,654	61,658	35,672	6,210	16,717	6,756	1,345
1903.....	36,152	594,300	58,605	23,574	59,725	45,630	6,249	15,659	8,102	(*i*)
1904.....	35,609	632,605	61,327	25,252	61,985	40,324	5,953	14,606	7,993	(*i*)
1905.....	42,265	588,289	67,678	28,425	68,675	33,681	7,741	(*b*)	7,451	(*i*)
1906.....	45,612	629,658	86,232	26,970	72,218	44,448	8,631	(*b*)	7,962	(*i*)
1907.....	44,012	592,989	91,120	29,539	70,432	45,898	8,808	(*b*)	6,666	(*i*)

(*a*) From *Accounts Relating to Trade and Navigation of the United Kingdom*. (*b*) Not reported. (*c*) Including naphtha, paraffin, paraffin oil and petroleum. (*d*) Previous reports not available. (*e*) Includes puddled iron. (*f*) Includes railroad material of all kinds. (*g*) Includes all soda compounds; not separate; enumerated previous to 1901. (*h*) Included under soda ash. (*i*) Included under spelter.

UNITED STATES.

Of the following tables, the first records the imports of foreign mineral and metal products into the United States, whether dutiable or duty free; the second shows the exports of materials produced in the United States; and the third reports the re-exports of products of foreign origin. These statistics are as reported by the Bureau of Statistics of the Department of Commerce and Labor, and special acknowledgment is due to Hon. O. P. Austin, chief of the bureau, for furnishing the figures for many substances which are not reported in the Monthly Summary. The complete statement of production in the United States is given on an early page in this volume.

IMPORTS. (a)

Year.	Aluminum. Crude. Lb.	Kg.	Value.	Value per Lb.	Ammonium Sulphate. Lb.	Metric Tons.	Value.	Value per Lb.
1900	256,559	116,374	$44,455	$0.172	24,024,188	10,897	$591,937	$0.025
1901	564,803	251,657	104,168	0.186	31,711,085	14,384	728,085	0.023
1902	745,217	338,028	215,032	0.290	35,535,556	16,119	858,036	0.024
1903	498,655	226,190	139,298	0.279	29,104,817	13,199	765,220	0.026
1904	515,416	234,293	128,350	0.249	39,859,690	18,077	1,058,981	0.027
1905	530,429	240,584	106,108	0.200	15,512,399	7,038	416,048	0.027
1906	770,713	349,195	154,292	0.200	31,797,291	14,423	894,663	0.028
1907	872,474	395,754	181,352	0.208	70,440,992	31,960	1,828,236	0.026
1908	465,317	210,785	80,268	0.173	76,475,104	34,698	1,982,830	0.026

Year.	Antimony. Lb.	Metric Tons.	Value.	Value per Lb.	Antimony Ore. Lb.	Metric Tons.	Value.	Value per Lb.
1900	3,632,843	1,648	$285,749	$0.079	6,035,734	2,738	$78,581	$0.013
1901	3,674,923	1,667	255,346	0.069	1,731,756	786	24,256	0.014
1902	5,742,703	2,605	347,899	0.061	1,639,043	743	29,476	0.018
1903	5,125,515	2,325	279,957	0.054	2,673,142	1,213	51,489	0.019
1904	4,056,299	1,840	235,401	0.058	2,487,602	1,129	50,362	0 020
1905	5,737,891	2,603	431,774	0.075	1,976,694	897	52,868	0.027
1906	7,900,194	3,593	1,417,816	0.179	2,247,131	1,019	128,347	0.057
1907	8,662,683	3,928	1,423,276	0.164	2,780,186	1,261	180,903	0.065
1908	8,114,651	3,954	572,979	0.071	3,280,922	1,488	106,930	0.033

Year.	Asbestos. Crude Value.	Mfd. Value.	Total Value.	Asphaltum. Long Tons.	Metric Tons.	Value.	Value per L. T.
1900	$631,796	$24,155	$655,951	113,557	115,374	$404,921	$3 57
1901	667,087	24,741	691,828	132,079	134,192	516,515	3.85
1902	729,421	33,013	762,434	139,944	142,183	439,570	3.09
1903	657,269	32,058	689,327	167,554	170,235	514,051	3 06
1904	700,572	51,290	751,862	119,575	121,489	510,524	4.27
1905	776,362	70,117	846,479	86,748	88,136	382,667	4.41
1906	536,500	.....	.....	97,274	98,830	388,010	3.93
1907	1,104,109	200,371	1,304,480	127,902	129,948	518,074	4.05
1908	1,068,342	147,548	1,215,869	131,862	133,971	587,698	4.45

Year.	Arsenic. (b)				Barytes.				Bauxite.			
	Lb.	Metric Tons.	Value.	Value per Lb.	Long Tons.	Metric Tons.	Value.	Value per L. T.	Long Tons.	Metric Tons.	Value.	Value per L. T.
1900....	(f)	.....	.....	.....	(f)	.....	.....	.....	8,656	8,795	$32,967	$3.81
1901....	(f)	.....	.....	.....	(f)	.....	.....	.....	17,866	18,153	66,107	3.70
1902....	(f)	.....	.....	.....	(f)	.....	.....	.....	15,790	16,043	54,410	3.45
1903....	7,391,566	3,241	$256,097	$0.036	6,344	6,446	$22,777	$3.59	14,889	15,127	49,684	3.34
1904....	6,391,566	2,900	226,481	0.036	6,689	6,796	27,463	4.11	15,475	15,723	49,577	3.20
1905....	6,444,083	2,924	219,198	0.034	7,879	8,005	36,796	4.67	11,726	11,914	46,517	3.96
1906....	7,639,507	3,464	336,609	0.044	4,293	4,362	37,296	8.69	17,809	18,094	63,221	3.55
1907....	9,922,870	4,500	553,440	0.056	28,350	28,804	174,225	6.15	25,065	25,466	93,208	3.72
1908....	9,592,881	4,558	417,137	0.056	(n) 12,196	12,390	58,822	4.83	21,679	22,033	87,823	4.05

Year.	Chloride of Lime.				Cement.			
	Lb.	Metric Tons.	Value.	Va lue per Lb.	Barrels. (c)	Metric Tons.	Value.	Value per Bbl.
1900....	132,520,478	60,111	$1,524,205	$0.012	2,386,684	433,037	$3,330,453	$1.40
1901....	120,611,346	54,709	1,673,190	0.014	944,892	170,431	1,305,692	1.38
1902....	112,374,478	50,973	1,456,435	0.013	1,904,790	361,932	2,581,883	1.29
1903....	113,285,240	51,586	912,843	0.008	2,317,951	420,569	3,027,111	1.30
1904....	87,909,168	39,876	707,174	0.008	1,046,404	189,910	1,382,913	1.32
1905....	104,919,462	47,604	843,285	0.008	846,577	153,644	1,102,041	1.30
1906....	105,221,371	47,718	863,490	0.008	2,205,712	400,115	2,950,268	1.33
1907....	112,090,783	50,833	939,248	0.008	2,006,228	363,929	2,637,424	1.31
1908....	74,602,059	33,848	621,713	0.008	839,246	152,313	1,189,560	1.42

Year.	Chrome Ore.				Bismuth.				Coal, Anthracite.			
	Long Tons.	Metric Tons.	Value.	Value per L.T.	Lb.	Kg.	Value.	Value per Lb.	Long Tons.	Metric Tons.	Value.	Value per L.T.
1900....	17,572	17,823	$305,001	$17.38	.....	.....	.....	.....	118	120	$649	$4.65
1901....	20,112	20,434	363,108	18.04	.....	.....	.....	.....	286	291	1,844	6.45
1902....	39,570	40,203	582,597	14.72	.....	.....	.....	.....	73,006	74,174	323,517	4.43
1903....	22,932	23,299	302,025	13.17	147,324	66,826	$235,199	$1.59	151,023	153,439	675,623	4.47
1904....	24,227	24,615	348,527	14.38	147,712	67,002	268,837	1.82	72,526	73,686	220,665	3.04
1905....	54,434	55,305	725,301	13.32	148,589	67,459	318,007	2.14	34,262	34,810	107,394	3.13
1906....	43,441	44,136	557,594	12.84	254,733	115,000	318,452	1.25	32,357	32,875	105,190	3.25
1907....	41,999	42,671	491,925	11.71	259,881	117,882	325,015	1.25	9,896	10,054	40,966	4.14
1908....	27,876	28,320	345,960	12.40	164,793	73,002	257,397	1.56	16,483	16,747	73,777	4.47

Year.	Coal, Bituminous.				Total Coal.		Coke.			
	Long Tons.	Metric Tons.	Value.	Value per L.T.	Long Tons.	Value.	Long Tons.	Metric Tons.	Value.	Value per L.T.
1900....	1,909,258	1,939,806	$5,019,553	$2.63	1,909,366	5,020,102	103,175	104,826	$371,341	$3.60
1901....	1,919,962	1,950,681	5,291,429	2.75	1,920,248	5,293,273	172,729	173,893	266,078	3.67
1902....	2,478,375	2,518,029	7,012,674	2.84	2,551,381	7,339,791	107,437	109,156	423,774	4.05
1903....	3,295,379	3,348,105	9,329,221	2.83	3,446,402	10,004,844	127,479	129,519	437,625	3.43
1904....	1,556,149	1,581,047	3,915,613	2.52	1,628,675	4,136,278	161,476	164,060	648,520	4.01
1905....	1,616,581	1,644,478	3,908,877	2.42	1,652,843	4,016,271	181,376	184,278	796,544	4.39
1906....	1,712,150	1,739,544	4,129,555	2.41	1,744,507	4,234,745	114,703	116,538	558,419	4.87
1907....	2,116,122	2,149,980	5,398,167	2.55	2,126,018	5,439,133	132,536	134,656	594,137	4.48
1908....	1,487,816	1,511,621	4,059,786	2.73	1,504,299	4,133,563	129,591	131,624	603,964	4.65

Year.	Cobalt Oxide.				Copper, Ore and Matte.			
	Lb.	Kg.	Value.	Value per Lb.	Long Tons.	Metric Tons.	Value.	Value per L.T.
1900	54,073	24,527	$68,651	$1.64	54,329	55,201	$5,195,010	$92.23
1901	71,969	32,645	134,208	1.86	96,047	97,584	14,692,645	152.99
1902	79,984	36,281	151,115	1.89	181,566	184,470	8,695,780	47.89
1903	73,350	33,272	145,264	1.98	284,912	289,471	3,177,582	11.15
1904	42,352	19,211	86,925	2.05	268,234	272,527	4,308,410	16.06
1905	70,048	31,802	139,377	1.99	296,251	300,991	5,765,238	19.46
1906	41,084	18,652	83,167	2.02	208,702	212,041	6,796,696	32.56
1907	42,794	19,421	73,028	1.71	291,957	297,096	9,048,270	31.32
1908	1,550	701	3,095	2.00	288,022	292,630	6,978,513	24.20

Year.	Copper, Ingots, Old, etc.				Cryolite.			
	Lb.	Metric Tons.	Value.	Value per Lb.	Long Tons.	Metric Tons.	Value.	Value per L. T.
1900	65,796,808	31,206	$10,557,870	$0.153	5,437	5,524	$72,763	$13.38
1901	73,826,406	33,488	11,812,216	0.160	5,383	5,469	70,886	13.17
1902	103,129,568	46,778	13,051,159	0.126	6,188	6,287	85,640	13.83
1903	136,707,995	62,011	17,262,148	0.126	7,708	7,831	102,879	13.35
1904	142,344,433	64,567	18,374,959	0.129	959	974	13,708	14.30
1905	160,619,385	72,876	22,103,741	0.137	1,600	1,623	22,482	14.05
1906	176,558,390	80,069	30,416,578	0.172	1,505	1,529	29,683	19.72
1907	192,901,267	87,523	38,658,754	0.200	1,438	1,461	28,920	20.10
1908	162,224,144	73,604	22,851,134	0.141	1,124	1,142	16,445	14.63

Year.	Emery Grains.				Emery Rock.			
	Lb.	Metric Tons.	Value.	Value per Lb.	Long Tons.	Metric Tons.	Value.	Value. per L. T.
1900	661,482	300	$26,520	$0.040	11,392	11,574	$202,980	$17.82
1901	1,116,729	506	43,207	0.039	12,441	12,640	240,856	19.35
1902	1,665,737	756	60,079	0.036	7,166	7,281	151,959	21.21
1903	3,595,239	1,630	109,272	0.030	10,385	11,059	188,985	17.36
1904	2,281,193	1,035	109,772	0.048	7,054	7,167	131,493	18.64
1905	3,209,915	1,456	143,729	0.045	11,073	11,250	185,689	16.77
1906	4,655,168	2,113	215,357	0.043	13,840	14,061	286,386	20.69
1907	4,282,228	1,942	186,156	0.043	11,235	11,415	211,184	18.80
1908	1,845,366	838	89,702	0.049	8,077	8,205	145,668	18.05

Year.	Phosphates, Crude.				Pig Iron.			
	Long Tons.	Metric Tons.	Value.	Value. per L. T.	Long Tons.	Metric Tons.	Value.	Value per L. T.
1900	137,086	139,272	$791,189	$5.77	52,565	53,406	$1,907,361	$36.28
1901	175,765	178,577	872,503	4.97	62,930	63,937	1,792,014	28.48
1902	137,336	139,584	646,264	4.70	619,354	629,264	10,935,831	17.66
1903	132,965	134,992	679,112	5.11	599,574	609,167	11,173,302	18.64
1904	130,214	132,297	745,744	5.73	79,590	80,772	1,765,107	22.20
1905	56,021	56,917	273,289	4.88	212,465	215,864	5,185,784	24.41
1906	23,281	23,653	147,547	6.34	379,828	385,905	11,851,210	31.20
1907	25,876	26,290	163,944	6.34	489,475	497,305	13,418,982	27.42
1908	26,734	27,161	175,365	6.56	92,202	93,677	2,886,339	31.35

Year.	Fuller's Earth.		Gold.		Iron Ore.			
	Long Tons.	Value.	In Coin and Bullion.	In Ore.	Long Tons.	Metric Tons.	Value.	Value per L. T.
1900	.....	.....	$45,703,256	$21,045,328	879,831	893,908	$1,303,196	$1.48
1901	.....	.....	33,237,629	21,524,251	966,950	982,421	1,659,273	1.72
1902	.....	.....	22,710,957	21,482,360	1,165,470	1,184,118	2,583,077	2.22
1903	15,267	$120,671	44,054,902	21,212,794	980,440	996,127	2,261,008	2.31
1904	9,126	78,006	75,646,128	9,157,106	487,613	495,415	1,101,284	2.26
1905	13,001	105,997	38,564,328	11,729,077	845,651	859,181	2,062,161	2.44
1906	13,238	108,696	139,705,887	15,873,493	1,060,390	1,077,356	2,967,434	2.80
1907	14,648	122,221	130,605,413	12,792,659	1,229,168	1,248,835	3,937,483	3.20
1908	10,963	92,413	38,346,267	11,930,026	776,898	789,326	2,224,248	2.86

Year.	Scrap, Iron and Steel.			Bar Iron.		
	Long Tons.	Metric Tons.	Value.	Long Tons.	Metric Tons.	Value.
1900	34,431	34,982	$663,231	19,685	19,094	$1,058,761
1901	20,130	20,452	339,827	20,792	21,126	1,093,736
1902	109,510	111,262	1,606,720	28,844	29,307	1,296,236
1903	82,921	84,248	1,273,941	43,592	44,290	1,904,469
1904	13,461	13,676	189,506	20,905	21,247	917,254
1905	23,731	24,111	370,328	37,294	37,891	1,522,434
1906	19,091	19,397	248,106	35,793	36,366	1,590,592
1907	27,652	28,094	368,842	39,746	40,382	1,774,441
1908	5,090	5,171	61,981	19,671	19,980	837,585

Year.	Rails.			Hoop, Band or Scroll.			Ingots, Blooms, Slabs, Billets, etc.		
	Long Tons.	Metric Tons.	Value.	Long Tons.	Metric Tons.	Value.	Long Tons.	Metric Tons.	Value.
1900	1,448	1,471	$56,129	165	167	$12,409	12,709	12,913	$1,332,896
1901	1,905	1,935	67,052	2,974	3,021	116,841	8,164	8,295	2,340,112
1902	63,522	64,538	1,576,679	3,362	3,416	131,052	289,318	293,965	7,943,818
1903	95,555	97,083	2,159,273	1,525	1,550	74,898	261,570	265,932	7,331,299
1904	37,776	38,390	808,775	2,135	2,169	60,934	(m) 10,807	10,980	1,537,531
1905	17,278	17,554	409,907	4,772	4,848	137,612	(m) 14,641	14,875	2,072,606
1906	4,943	5,022	137,104	10,231	10,395	256,836	(m) 21,337	21,678	3,010,589
1907	3,752	3,812	104,958	1,508	1,532	82,706	(m) 19,334	19,643	3,004,178
1908	1,719	1,752	53,128	1,110	1,127	75,920	11,212	11,391	1,437,514

Year.	Sheet, Plate and Taggers Iron or Steel.			Tin Plates, Terne Plates and Taggers Tin.			Wire Rods.		
	Long Tons.	Metric Tons.	Value.	Long Tons.	Metric Tons.	Value.	Long Tons.	Metric Tons.	Value.
1900	5,143	5,226	$426,541	60,386	61,356	$4,617,813	21,092	21,430	$1,212,594
1901	5,626	5,716	443,880	77,395	78,638	5,294,789	16,804	17,073	964,744
1902	7,156	7,270	545,739	60,115	61,080	4,023,421	21,382	21,725	1,033,074
1903	11,557	11,741	540,272	47,360	48,118	2,999,252	20,836	21,169	1,028,977
1904	4,165	4,232	302,500	70,552	71,782	4,354,761	15,313	15,558	707,779
1905	2,236	2,272	242,955	65,740	66,792	4,090,523	17,616	17,898	800,027
1906	3,231	3,283	325,276	56,982	57,894	3,883,225	17,799	18,084	876,270
1907	3,749	3,809	367,140	57,773	58,697	4,462,522	17,076	17,349	851,571
1908	2,628	2,669	377,549	58,492	59,426	3,651,576	11,208	11,387	543,170

Year.	Wire and Articles Made from.			Total Iron Imports. (e)	Lead in Ore and Base Bullion.		
	Long Tons.	Metric Tons.	Value.		Short Tons.	Metric Tons.	Value.
1900	1,848	1,877	$409,087	$20,443,911	(g)114,397	103,780	$3,975,695
1901	4,129	4,192	585,354	20,404,122	111,867	101,486	4,807,762
1902	3,469	3,525	606,724	41,468,826	105,186	95,425	4,424,511
1903	5,018	5,098	728,430	41,258,864	103,394	93,790	3,596,635
1904	3,956	4,019	624,892	21,621,970	104,127	94,464	3,517,691
1905	3,978	4,042	705,465	26,401,283	92,657	84,081	3,565,282
1906	6,610	6,716	1,079,868	34,827,132	72,371	65,640	3,490,750
1907	(f)	(f)	1,551,415	38,789,992	70,538	64,019	3,579,990
1908	(f)	(f)	1,003,973	19,957,261	109,286	99,171	4,384,904

Year.	Lead in Pigs and Old.			Lead, Sheet, Pipe, Shot, Etc.			Other Lead Mfrs.	Total Lead.
	Short Tons.	Metric Tons.	Value.	Lb.	Metric Tons.	Value.		
1900	.....	.....	.....	27,945	13	$1,393	$5,854	$3,964,942
1901	604	548	$33,882	56,735	26	2,773	4,654	4,849,071
1902	2,529	2,294	132,500	224,208	102	7,765	18,918	4,533,694
1903	3,023	2,742	164,528	17,008	8	810	8,071	3,770,044
1904	8,724	7,914	461,316	69,581	32	2,441	7,755	3,989,203
1905	5,720	5,812	367,106	54,779	25	2,638	4,580	3,939,606
1906	11,763	10,669	910,417	346,177	157	17,250	20,681	4,401,167
1907	9,277	8,414	846,166	734,418	333	39,210	12,736	4,426,156
1908	2,759	2,504	182,503	42,376	19	2,026	46,486	4,567,407

Yr.	White Lead.				Litharge.				Red Lead.			
	Lb.	Metric Tons.	Value.	Value per Lb.	Lb.	Metric Tons.	Value.	Value per Lb.	Lb.	Metric Tons.	Value.	Value per Lb.
1900	456,872	207	$26,366	$0.062	77,314	35	$2,852	$0.032	549,551	249	$25,532	$0.046
1901	384,673	174	21,226	0.056	49,306	22	1,873	0.038	485,467	220	19,370	0.040
1902	506,423	230	25,320	0.050	88,115	40	2,908	0.033	1,075,839	488	37,383	0.035
1903	453,284	206	24,595	0.054	42,756	19	1,464	0.034	1,152,715	523	40,846	0.035
1904	587,338	266	33,788	0.058	44,541	20	1,500	0.034	836,077	379	30,115	0.036
1905	597,510	271	34,722	0.058	117,757	53	4,139	0.035	704,402	320	26,553	0.038
1906	647,636	294	41,233	0.064	87,230	40	3,737	0.043	1,093,619	497	50,741	0.046
1907	584,309	265	37,482	0.064	90,475	41	4,396	0.048	679,171	308	35,959	0.053
1908	540,311	245	30,451	0.056	96,184	44	3,327	0.035	645,073	292	28,155	0.044

Year.	Orange Mineral.				Magnesite.			
	Lb.	Metric Tons.	Value.	Value per Lb.	Long Tons.	Metric Tons.	Value.	Value per L. T.
1900	1,068,793	485	$61,885	$0.059	.....	.....	.....	.....
1901	977,644	443	52,409	0.053	30,350	30,835	.....	.....
1902	997,494	452	49,060	0.049	45,157	45,880	$373,928	8.28
1903	756,742	343	36,407	0.048	49,684	50,479	461,399	9.29
1904	766,469	348	37,178	0.049	35,106	35,668	286,828	8.17
1905	628,003	285	31,106	0.049	66,405	67,506	638,619	9.46
1906	770,342	350	42,519	0.055	80,711	82,002	863,492	10.70
1907	615,015	279	37,793	0.061	88,400	89,814	875,359	9.90
1908	485,407	220	26,645	0.055	75,442	76,648	736,763	9.80

Year.	Manganese Ore.				Mica.	Nickel. (h)	Nickel Ore and Matte.		
	Long Tons.	Metric Tons.	Value.	Value per L. T.			Long Tons.	Metric Tons.	Value.
1900	256,252	260,352	$2,042,361	$7.97	$319,560	(f)	(f)	.....	.....
1901	165,720	168,372	1,486,573	8.97	335,054	.....	(f)	.....	.....
1902	235,576	239,345	1,931,282	8.20	466,332	.....	(f)	.....	.....
1903	146,056	148,393	1,278,108	8.75	317,969	$207,954	11,936	16,191	$1,285,935
1904	108,519	110,255	901,592	8.31	269,808	206,021	8,549	8,685	915,470
1905	257,033	261,146	1,952,407	7.60	403,755	335,211	13,451	13,666	1,626,920
1906	221,260	224,800	1,696,043	7.67	1,042,608	86,336	15,156	15,398	1,816,631
1907	209,021	211,236	1,793,143	8.59	915,259	90,153	16,888	17,158	2,153,971
1908	178,203	181,054	1,350,223	7.59	264,755	101,398	16,322	16,582	2,396,217

Year.	Oil, Mineral.			Platinum, Unmanufactured.				Platinum Mfres.
	Gal.	Value.	Value per Gal.	Lb. Troy.	Kg.	Value.	Value per Lb. Troy.	
1900	3,039,094	$274,766	$0.091	9,246	3,450	$1,728,777	$187.00	$36,714
1901	2,294,684	151,913	0.066	7,496	2,797	1,673,713	223.30	24,482
1902	3,578,393	207,310	0.058	8,670	3,235	1,950,362	224.96	37,618
1903	4,266,974	261,199	0.061	9,540	3,561	1,921,772	201.44	1,727,330
1904	4,846,681	277,399	0.057	8,648	3,230	1,812,242	209.55	105,636
1905	13,725,720	672,127	0.049	8,681	3,240	1,985,107	228.67	188,156
1906	21,045,316	1,061,076	0.050	13,928	5,198	3,601,021	258.54	187,639
1907	29,553,234	1,615,011	0.055	7,515	2,805	2,509,926	333.99	175,651
1908	40,839,522	2,218,999	0.054	4,155	1,551	1,096,615	263.92	134,119

Yr.	Potassium Salts.											
	Chlorate.			Chloride.			Chromate and Bichromate.			Nitrate.		
	Lb.	Value.	Value per Lb.	Lb.	Value.	Value per Lb.	Lb.	Value.	Value per Lb.	Lb.	Value.	Value per Lb.
		$	$		$	$		$	$		$	$
1900	1,243,612	68,772	0.055	130,175,481	1,976,604	0.015	111,761	7,758	0.069	10,545,392	276,664	0.026
1901	811,127	61,348	0.076	143,189,337	2,316,577	0.015	430,996	29,224	0.068	9,656,193	253,286	0.026
1902	1,209,148	60,429	0.050	140,980,460	2,141,553	0.015	231,009	15,161	0.066	10,505,474	299,416	0.028
1903	468,042	19,308	0.041	169,337,673	2,550,478	0.015	41,229	2,784	0.067	13,835,068	367,721	0.026
1904	95,889	4,209	0 044	174,865,872	2,832,554	0.016	26,053	1,817	0.069	14,184,287	376,931	0.027
1905	42,510	2,876	0.067	214,207,064	3,326,748	0.016	59,650	4,225	0.070	9,911,534	304,596	0.027
1906	45,873	3,103	0.068	223,203,387	3,858,895	0.017	30,098	2,102	0.080	11,326,256	371,595	0.033
1907	12,980	959	0.074	252,303,441	4,175,353	0.017	18,171	1,307	0.072	18,291,390	574,977	0.031
1908	17,607	1,447	0.082	214,338,887	3,415,326	0.016	216,080	15,453	0.072	16,118,160	470,116	0.029

Year.	Potassium Salts. All Other.		Precious Stones.			Pyrites. (i)			
	Lb.	Value.	Uncut.	Cut, not Set.	Jewelry.	Long Tons.	Metric Tons.	Value.	Value per L. T.
1900	54,904,088	$1,407,303	$3,751,219	$9,612,127	(f)	332,517	337,837	$1,095,598	$3.30
1901	72,489,913	1,636,856	6,637,360	17,166,049	(f)	398,969	405,353	1,407,244	3.53
1902	91,857,009	1,820,585	8,282,760	18,494,288	(f)	437,319	444,316	1,623,430	3.71
1903	70,205,850	1,593,380	10,374,377	15,428,319	$954,456	427,319	434,156	1,636,450	3.83
1904	74,720,241	1,678,699	10,316,615	16,934,090	803,952	413,585	420,202	1,533,564	3.73
1905	82,935,632	1,891,081	10,206,350	26,699,670	801,566	515,732	520,926	1,780,800	3.47
1906	30,302,735	763,513	11,937,542	32,201,949	988,766	597,347	606,903	2,138,746	3.58
1907	91,299,496	2,220,685	8,740,278	23,706,975	1,069,373	656,479	666,981	2,637,485	4.01
1908	69,382,278	1,721,626	2,287,440	11,660,442	699,506	668,115	678,804	2,624,339	3.93

Year.	Salt.				Silver.		Sodium Nitrate.			
	Short Tons.	Metric Tons.	Value.	Value per Sh. T.	In Coin and Bullion.	In Ore.	Long Tons.	Metric Tons.	Value.	Value per L. T.
1900	207,933	188,636	$633,192	$3.05	$14,695,965	$25,404,378	182,108	185,022	$4,935,520	$27.12
1901	194,967	176,872	670,648	3.44	12,957,987	18,188,795	208,654	211,992	5,997,595	28.82
1902	188,775	167,481	654,990	3.47	8,502,614	17,900,321	205,245	208,529	5,996,205	29.21
1903	157,201	142,494	489,179	3.11	7,935,844	16,038,664	272,947	277,314	8,700,806	31.88
1904	167,295	151,810	515,822	3.08	11,865,805	14,221,237	228,012	231,660	9,333,613	32.41
1905	158,449	143,783	491,079	3.10	16,472,911	19,466,224	321,231	326,371	11,206,548	34.89
1906	170,505	154,648	502,583	2.95	20,402,738	23,825,103	372,222	378,178	14,115,206	37.92
1907	153,435	139,166	452,227	2.95	17,652,679	28,259,681	364,610	370,444	14,844,675	40.71
1908	156,008	142,043	440,484	2.81	14,169,524	28,054,606	310,713	315,684	11,385,393	36.68

Year.	Sodium Hydroxide (Caustic).			Soda Ash and Carbonate.			All Other Sodium Salts.	
	Lb.	Value.	Value. per Lb.	Lb.	Value.	Value per Lb.	Lb.	Value.
1900	8,403,749	$150,530	$0.018	73,815,425	$613,379	$0.008	20,484,938	$259,802
1901	3,812,847	94,303	0.025	31,415,788	276,261	0.009	14,491,559	189,543
1902	3,334,697	77,482	0.020	31,889,252	284,634	0.009	17,151,682	233,745
1903	2,970,426	73,647	0.025	25,313,370	228,041	0.009	14,272,646	268,738
1904	2,570,984	64,405	0.025	23,631,832	205,496	0.009	10,399,711	281,527
1905	2,245,789	56,515	0.025	15,754,979	146,812	0.009	11,257,629	247,413
1906	1,209,053	35,262	0.022	6,800,288	71,013	0.010	.....	.....
1907	1,297,070	37,894	0.029	6,198,136	66,521	0.011	8,481,979	258,262
1908	874,813	26,079	0.029	3,515,933	38,372	0.011	.....	296,777

Year.	Sulphur.									
	Crude.				Flowers.			Refined.		
	Long Tons.	Metric Tons.	Value.	Value per L. T.	Long Tons.	Metric Tons.	Value.	Long Tons.	Metric Tons.	Value.
1900	166,457	169,120	$2,918,610	$17.53	628	638	$17,437	243	247	$ 6,279
1901	174,162	176,949	3,256,951	18.70	748	761	20,201	268	272	6,308
1902	176,951	179,782	3,360,562	19.00	738	750	19,954	14	15	369
1903	188,888	191,910	1,649,756	19.32	1,354	1,883	52,690	189	192	7,254
1904	128,885	130,947	2,463,779	19.12	1,332	1,353	39,133	204	207	9,776
1905	83,201	84,582	1,522,005	18.29	572	581	16,037	778	709	19,960
1906	72,404	73,562	1,282,873	17.72	1,100	1,118	29,565	709	720	17,928
1907	20,399	20,725	355,944	17.45	1,458	1,481	41,216	606	616	14,589
1908	19,620	19,993	318,577	16.24	793	804	22,562	692	700	17,227

Year.	Talc.				Tin.			
	Short Tons.	Metric Tons.	Value.	Value per Sh. T.	Lb.	Metric Tons.	Value.	Value per Lb.
1900	79	72	$ 1 070	$13.54	69,989,502	31,747	$19,458,586	$0.278
1901	2,386	2,164	27,015	11.74	74,560,487	33,820	19,024,761	0.255
1902	2,859	2,594	35,336	12.35	85,043,353	38,575	21,263,337	0.250
1903	1,790	1,623	19,685	11.00	83,133,847	37,702	22,265,367	0.268
1904	3,268	2,964	36,370	11.13	83,168,557	37,718	22,356,896	0.270
1905	4,000	3,630	48,225	12.06	89,227,698	40,507	26,316,023	0.294
1906	5,643	5,118	67,818	12.02	101,027,188	45,816	37,446,508	0.371
1907	10,060	10,221	126,301	12.56	82,548,838	37,436	32,075,091	0.389
1908	7,429	6,738	97,296	13.08	82,503,190	37,433	23,932,560	0.290

Year.	Zinc.								
	Blocks, Pigs and Old.				Oxide (j).		Sulphide.		Mfrs.
	Lb.	Metric Tons.	Value.	Value, per Lb.	Lb.	Value.	Lb.	Value.	Valu
1900.....	2,013,196	913	$97,772	$0.048	2,657,514	.....	.....	.....	$36,836
1901.....	775,881	352	30,920	0.040	3,327,976	.....	.....	.....	42,643
1902.....	1,238,091	561	46,713	0.038	3,434,466	.....	1,247,936	$32,879	37,191
1903.....	728,614	330	30,900	0.042	3,653,076	$188,496	1,229,806	33,077	18,928
1904.....	933,474	423	44,455	0.048	2,809,905	165,110	1,228,875	31,382	11,918
1905.....	1,042,081	473	51,052	0.048	3,779,311	196,220	1,235,360	33,308	12,390
1906.....	4,407,481	1,999	253,310	0.057	4,494,014	288,065	1,286,469	40,112	17,385
1907.....	3,555,890	1,613	210,322	0.059	5,311,318	323,551	1,570,073	51,435	16,282
1908.....	1,762,627	799	85,885	0.049	4,635,101	262,876	1,048,109	46,713	7,474

(a) From Summary of Commerce and Finance of the United States. (b) Includes arsenic sulphide. (c) Barrels of 400 lb. (e) Not including iron ore. (f) Not reported. (g) Includes pig and old. (h) Includes nickel oxide, alloys in which nickel is the principal constituent and manufactures of nickel. (i) Containing more than 25 per cent sulphur. (j) Includes white pigments containing zinc but not lead, dry and in oil. (m) Includes bar of steel and steel forms not elsewhere specified. The high value is due to the value of "high-speed" steel. (n) Crude.

EXPORTS OF DOMESTIC PRODUCTS. (a)

Year..	Aluminum and Mfrs. of.	Asbestos and Mfrs. of.	Cement.			
			Bbl. (i)	Metric Tons.	Value.	Value per Bbl.
1900.....	$281,821	$124,971	100,400	18,216	$225,306	$2.24
1901.....	183,579	113,316	373,934	67,393	679,296	1.82
1902.....	116,052	130,437	340,821	61,838	526,471	1.54
1903.....	157,187	158,360	285,463	51,748	433,984	1.52
1904.....	166,876	223,096	774,940	140,898	1,104,086	1.42
1905.....	290,777	.....	1,016,236	185,345	1,387,706	1.37
1906.....	364,251	259,760	583,299	105,811	944,886	1.62
1907.....	304,938	200,371	900,550	163,360	1,450,841	1.61
1908.....	330,092	296,890	846,528	153,630	1,249,229	1.47

Year.	Coal.								Coke.	
	Anthracite.				Bituminous.					
	Long Tons.	Metric Tons.	Value.	Value per L. T.	Long Tons.	Metric Tons.	Value.	Value per L. T.	Long Tons.	Value.
1900..	1,654,610	1,681,084	$7,092,489	$4.29	6,262,909	2,363,631	$14,431,590	$2.31	376,999	$1,358,968
1901..	1,993,307	2,025,200	8,937,147	4.48	5,390,086	3,476,327	13,085,763	2.53	384,330	1,516,898
1902..	907,977	922,505	4,301,946	4.73	5,218,969	5,302,472	13,927,063	2.66	392,491	1,785,188
1903..	2,008,857	2,040,999	9,780,044	4.86	6,303,241	6,404,093	17,410,385	2.76	416,385	2,091,875
1904..	2,228,392	2,264,046	11,077,570	4.97	6,345,126	6,446,648	17,160,538	2.74	523,090	2,311,401
1905..	2,229,983	2,265,663	11,104,654	4.98	6,959,265	7,070,613	17,867,964	2.56	599,054	2,243,010
1906..	2,216,969	2,252,441	10,896,200	4.91	(k) 7,704,850	7,828,128	(k)19,787,459	2.57	765,190	2,753,551
1907..	2,698,072	2,741,241	13,217,985	4.90	(m)10,454,647	10,621,950	(m)26,982,111	2.54	874,689	3,206,793
1908..	2,752,358	2,796,394	13,524,595	4.92	(m) 9,100,819	9,246,431	(m)23,361,914	2.53	622,228	2,161,032

Year.	Copper.									
	In Ore and Matte (b).				Ingots, Bars, Plates and Old.				Mfrs.	Total Except Ore.
	Long Tons.	Metric Tons.	Value.	Value per L. T.	Lb.	Metric Tons.	Value.	Value per Lb.	Value.	
1900..	10,007	10,168	$1,332,829	$133.18	337,973,751	153,304	$55,285,047	$0.164	$2,257,563	$57,542,610
1901..	19 613	19,924	2,536,549	129.40	194,249,828	88,111	31,692,563	0.164	1,842,336	33,354,899
1902..	18,035	18,321	1,326,131	73.53	354,668,849	160,876	43,392,800	0.122	2,092,798	45,485,398
1903..	12,291	12,488	855,367	69.59	310,729,524	140,920	41,170,059	0.132	2,339,729	43,509 788
1904..	18,927	19,230	1,202,537	63.54	554,550,030	251,497	71,488,116	0.129	3,328,818	74,816,934
1905..	37,888	38,291	1,531,429	40.63	534,907,619	242,699	80,693,232	0.151	4,184,070	84,890,302
1906..	47,619	48,330	1,760,140	36.96	454,752,018	206,239	84,728,400	0.186	4,284,011	89,013,011
1907..	99,141	100,727	2,452,562	24.74	508,929,401	230,799	94,912,185	0.186	5,888,170	100,800,355
1908..	63,149	64,158	1,254,172	19.87	661,876,127	300,302	87,393,200	0.132	3,162,303	90,555,503

Year.	Gold.		Iron							
			Ore.				Pig.			
	In Coin and Bullion. (c)	In Ore. (d)	Long Tons.	Metric Tons.	Value.	Value per L. T.	Long Tons.	Metric Tons.	Value.	Value Per L. T.
1900	$54,064,607	$69,926	51,460	52,283	$154,756	$3.01	286,687	291,404	$4,654,582	$16.23
1901	56,717,350	1,012,589	64,703	65,748	163,465	2.54	81,211	82,510	1,257,699	15.65
1902	35,722,835	307,756	88,445	89,860	294,168	3.32	27,487	27,927	502,947	18.30
1903	43,765,360	581,474	80,611	81,901	255,728	3.17	20,379	20,705	384,334	18.86
1904	120,226,424	985,403	213,865	217,287	458,823	2.14	49,025	49,809	764,543	15.60
1905	46,099,580	694,887	208,017	211,345	530,457	2.55	49,221	50,009	762,899	15.50
1906	46,068,451	640,707	265,240	269,484	771,831	2.91	83,317	84,650	1,506,774	18.08
1907	54,809,688	345,993	278,208	282,659	763,422	2.74	73,703	74,879	1,508,938	20.43
1908	80,778,091	437,365	309,099	314,043	1,012,924	3.29	46,696	47,441	789,318	16.92

Year.	Iron, Bar.				Iron, Band, Hoop and Seroll.			Billets, Ingots and Blooms.			
	Long Tons.	Metric Tons.	Value.	Value per L. T.	Long Tons.	Metric Tons.	Value.	Long Tons.	Metric Tons.	Value.	Value per L. T.
1900	13,298	13,512	$558,576	$42.04	2,976	3,024	$137,437	107,385	109,103	$2,915,371	$27.15
1901	17,708	17,993	674,671	38.16	1,561	1,586	74,056	28 614	29.072	708,887	24.78
1902	22,249	22,605	869,519	39.08	1,674	1,701	82,322	2,409	2,447	74,938	31.11
1903	19,380	19,690	796,631	41.11	1,241	2,175	101,839	5,445	5,532	141,924	26.07
1904	29,582	30,055	1,133,128	34.93	3,425	3,489	162,039	314,324	319,353	6,150,035	19.56
1905	32,025	32,537	1,255,418	39.20	4,426	4,497	182,431	237,738	241,542	4,701,909	19.79
1906	56,024	56,920	2,575,905	45.98	5,405	5,491	242,776	192,616	195,698	4,094,659	21.26
1907	24,190	24,577	1,092,631	45.17	8,587	8,724	395,758	79,991	81,271	2,013,319	25.17
1908	8,224	8,355	362,909	44.12	4,334	4,402	223,073	112,177	113,390	2,674,524	23.84

Year.	Iron, Nails and Spikes, Cut.				Iron, Nails and Spikes, All Other.				Iron, Plates and Sheets.		
	Lb.	Metric Tons.	Value.	Value per Lb.	Lb.	Metric Tons.	Value.	Value per Lb.	Long Tons.	Met. Tons.	Value.
1900	25,005,308	11,342	$626,497	$0.025	65,444,287	29,681	$1,816,813	$0.028	9,331	9,481	$600,600
1901	20,835,944	9,452	450,331	0.021	46,295,262	21,001	1,152 368	0.025	6,909	7,020	452,695
1902	16,122,775	7,312	339,227	0.021	64,565,650	29.287	1,456,768	0.022	3,434	3,489	229,887
1903	19,912,563	9,031	424,985	0.021	75,654,532	34.310	1,698,500	0.024	4,782	4,858	273,618
1904	20,772,049	9,422	416,339	0.020	80,279,746	36.403	1,949,908	0.024	4,728	4,804	247,694
1905	17,674,099	8,019	352,405	0.020	89,976,088	40,506	2,118,836	0.024	8,004	8,132	460,995
1906	16,951,893	7,688	340,526	0.020	116,310,428	52,747	2,731,021	0.024	17,054	17,327	1,139,526
1907	15,521,208	7,042	354,802	0.023	111,670,147	50,642	3,014,863	0.027	40,651	41,301	2,902,025
1908	15,721,898	7,133	364,202	0.023	71,427,124	32,407	1,813,784	0.025	44,100	44,805	2,985,538

Year.	Steel, Sheets and Plates.			Iron Rails.				Steel Rails.			
	Long Tons.	Metric Tons.	Value.	Long Tons.	Metric Tons.	Value.	Value per L. T.	Long Tons.	Metric Tons.	Value.	Value per L. T.
1900	45,534	46,264	$1,638,478	5,374	5,460	$119,206	$22.18	356,445	361,945	$10,895,416	$30.58
1901	23,923	24,303	959,471	901	915	32,357	35.93	318,055	323,044	8,628,781	27.14
1902	14,866	15.104	725.547	211	214	4.639	22.02	67,455	68,534	1,902,396	28.09
1903	13,312	13.525	657.713	181	184	8,808	48.67	30,656	31,146	937,779	30.59
1904	50.477	51.278	2,064.241	1,405	1,427	23,870	17.00	414.845	421.482	10.661.222	25.72
1905	67,093	68,166	2,889,084	*Nil.*	.....	.....	.....	295.023	299.473	7,310,029	24.78
1906	93,601	95,099	4,081,915	*Nil.*	.....	.....	.....	328.036	333.285	8,903,411	27.14
1907	82,045	83,358	4,262,582	*Nil.*	.....	.....	.....	338,906	344,328	10,411,072	30.72
1908	60,893	61,865	3,422,031	*Nil*	.....	.....	.....	196,510	199,654	6,021,549	30.62

Year.	Structural Iron and Steel.				Wire.				Steel Wire Rods.			
	Long Tons.	Metric Tons.	Value.	Value per L. T.	Long Tons.	Metric Tons.	Value.	Value per L. T.	Long Tons.	Metric Tons.	Value.	Value per L. T.
1900	67,714	68,797	$3,570,769	$52.73	78,014	79,262	$4,604,047	$59.77	10,652	10,822	$505,529	$47.37
1901	54,005	54,869	3,031,861	56.10	88,238	89,650	4,805,008	54.36	8,165	8,296	271,552	33.26
1902	53,859	54,721	2,828,400	52.52	97,843	99,414	5,140,702	52.54	24,613	25,007	831,067	33.76
1903	30,641	31,131	1,788,556	58.37	108,521	110,258	5,528,726	50.94	22,360	22,718	713,718	31.92
1904	55,514	56,402	2,777,768	50.04	118,581	120,478	5,935,093	50.05	20,073	20,394	695,448	34.64
1905	84,234	85,582	4,357,186	51.73	142,601	144,883	7,061,442	49.52	6,514	6,618	277,651	42.62
1906	112,555	114,356	6,140,861	54.56	174,014	176,798	8,770,042	50.40	5,896	5,990	221,679	37.60
1907	138,442	140,657	7,784,618	56.23	161,228	163,808	9,164,829	56.84	10,653	10,823	465,757	43.72
1908	116,878	118,746	6,289,610	53.80	136,167	138,344	7,270,794	53.35	7,412	7,530	277,694	37.39

Year.	Lead and Mfrs. of.	Nickel. (e)	Petroleum products. (In Thousands of Units.)*								
			Crude.			Naphtha.			Illuminating Oil.		
			M Gals.	M Value.	Value per Gal.	M Gals.	M Value.	Value per Gal.	M Gals.	M Value.	Value per Gal.
1900	$459,574	$1,382,727	138,161	$7,341	$0.053	18,570	$1,681	$0.081	739,163	$54,663	$0.074
1901	625,234	1,521,291	127,008	6,038	0.050	21,685	1,742	0.079	827,479	53,491	0.065
1902	696,010	924,579	145,234	6,331	0.042	19,683	1,393	0.071	778,801	49,079	0.063
1903	491,362	703,550	126,512	6,782	0.054	12,973	1,519	0.118	691,537	51,356	0.074
1904	616,126	2,130,933	111,176	6,351	0.057	24,989	2,322	0.093	761,358	58,384	0.077
1905	511,690	2,894,700	126,185	6,086	0.048	28,420	2,215	0.078	881,450	54,901	0.062
1906	600,057	3,493,643	148,045	7,731	0.052	27,545	2,488	0.090	878,284	54,858	0.063
1907	686,096	2,845,663	126,306	6,334	0.050	34,625	3,676	0.106	905,924	59,635	0.066
1908	599,640	3,297,988	149,190	6,520	0.044	43,890	4,543	0.103	1,129,005	75,988	0.067

Year.	Petroleum Products. (In Thousands of Units.)*									
	Lubricating Oil.			Residue, Etc. (g)			Paraffin.			
	M Gals.	M Value.	Value per Gal.	M Gals.	M Value.	Value per Gal.	M Lb.	M Metric Tons.	M Value.	Value per Lb.
1900	71,211	$9,933	$0.139	19,750	$845	$0.042	157,108	71.2	$8,186	$0.052
1901	75,306	10,260	0.136	27,596	1,255	0.046	151,694	68.8	7,960	0.052
1902	82,201	10,872	0.133	38,316	922	0.024	175,269	79.5	8,398	0.048
1903	95,622	12,690	0.133	9,753	282	0.029	204,120	92.6	9,596	0.047
1904	89,738	12,389	0.138	34,904	1,174	0.034	174,582	79.2	8,273	0.047
1905	113,730	14,312	0.126	70,728	2,128	0.030	160,836	73.0	7,873	0.049
1906	151,260	18,690	0.124	64,645	1,971	0.030	173,504	72.9	8,463	0.049
1907	152,029	19,210	0.126	75,775	2,528	0.033	207,504	94.1	10,209	0.049
1908	147,769	18,971	0.128	77,552	2,793	0.036	141,667	64.2	6,923	0.049

Year.	Crude Phosphates.				Quicksilver.			Silver.	
	Long Tons.	Metric Tons.	Value.	Value per L. T.	Lb.	Metric Tons.	Value.	In Coin and Bullion (c)	In Ore (d)
1900	619,995	629,915	$5,217,560	$8.38	778,191	353	$425,812	$65,705,909	$515,755
1901	729,539	741,212	5,839,245	8.01	843,938	383	475,609	55,526,975	111,383
1902	802,086	814,919	6,193,372	7.73	1,013,434	459	575,099	49,228,303	44,651
1903	785,259	797,823	6,109,230	7.78	1,344,615	610	719,119	40,531,095	79,347
1904	842,484	855,964	6,521,555	7.74	1,611,365	731	847,108	49,975,370	159,875
1905	934,940	949,899	7,465,592	7.91	1,009,446	458	497,470	54,133,721	3,379,381
1906	904,214	918,681	7,373,945	8.16	484,151	219	243,914	57,012,104	266,674
1907	1,018,212	1,034,503	8,387,176	8.24	384,913	174	192,094	61,202,024	423,842
1908	1,188,411	1,207,425	9,294,009	7.82	224,692	102	124,960	51,554,414	283,257

Year.	Zinc Ore.				Zinc Pigs, Bars, Plates and Sheet.			
	Long Tons.	Metric Tons.	Value.	Value per L. T.	Lb.	Metric Tons.	Value.	Value per Lb.
1900	37,555	38,158	$1,133,633	$30.19	44,802,577	20,322	$2,217,693	$0.050
1901	39,425	40,056	1,167,684	29.62	6,780,221	3,071	288,906	0.043
1902	49,762	50,558	1,449,104	29.12	6,473,135	2,936	300,557	0.046
1903	35,188	35,751	987,000	28.05	3,041,911	1,380	163,379	0.053
1904	32,063	32,576	905,782	28.25	20,145,942	9,204	1,094,490	0.053
1905	27,630	28,072	848,451	30.71	11,031,815	5,005	682,254	0.062
1906	24,750	25,146	733,300	29.63	9,340,455	4,236	583,526	0.062
1907	18,171	18,462	579,490	31.89	1,126,753	511	75,194	0.067
1908	23,311	23,683	877,745	37.61	5,280,344	2,396	250,254	0.047

Year.	Zinc Oxide.			
	Lb.	Metric Tons.	Value.	Value per Lb.
1900	11,391,666	5,167	$496,380	$0.044
1901	9,122,263	4,138	393,259	0.043
1902	10,716,364	4,861	433,722	0.040
1903	14,429,885	6 544	578,215	0.041
1904	16,313,826	7,399	628,494	0.039
1905	22,559,625	10,236	810,203	0.036
1906	31,156,616	14.129	1,149,297	0.037
1907	26,512,920	12,023	1,069,924	0.040
1908	4,635,101	2,103	262,876	0.057

RE-EXPORTS OF FOREIGN PRODUCTS. (a)

Year.	Antimony.			Antimony Ore.			Asphaltum, Crude.		
	Lb.	Metric Tons.	Value.	Short Tons.	Metric Tons.	Value.	Long Tons.	Metric Tons.	Value.
1900	23,520	10.7	$2,352	*Nil.*	.....	.....	629	639	$10,044
1901	*Nil.*	.....	.....	25	22.1	$1,536	2,209	2,244	18,078
1902	37,184	16.9	2,710	104	94.6	4,602	2,930	2,977	23,564
1903	79,917	36.0	4,478	*Nil.*	.....	.....	1,605	1,631	13,894
1904	31,077	14.0	1,734	214	194.0	10,775	1,887	1,917	26,272
1905	*Nil.*	.....	.....	*Nil.*	.....	.....	1,081	1,098	18,190
1906	24,892	11.2	4,939	*Nil.*	.....	.....	1,765	1,793	22,324
1907	47,999	21.8	9,064	6	5	273	8,288	8,421	31,749
1908	1,763	0.8	125	4.8	4.3	663	4,263	4,290	23,919

Year.	Cement.			Chemicals.					
				Salts of Potassium. (f)			Chloride of Lime.		
	Bbl. (i)	Metric Tons.	Value.	Lb.	Kg.	Value.	Lb.	Kg.	Value.
1900	39,540	7,174	$63,890	808,701	366,824	$43,524	148,116	67,185	$1,987
1901	43,691	7,927	72,761	633,100	287,182	43,446	13,916	6,312	312
1902	32,594	5,913	48,797	1,266,145	574,323	59,789	198,794	90,172	2,997
1903	25,362	4,601	32,156	1,299,905	589,637	33,264	836,411	379,696	7,609
1904	39,711	7,186	54,486	1,262,222	572,544	33,358	1,434	650	13
1905	31,874	5,782	40,583	3,053,191	1,386,149	63,652	100	102	3
1906	16,216	2,941	19,487	2,264,175	1,027,935	77,043	*Nil.*	.....	.....
1907	20,697	3,754	30,435	2,675,248	1,285,892	75,470	*Nil.*	.....	.....
1908	9,552	1,734	11,455	1,046,689	570,445	34,505	*Nil.*	.....	.....

Year.	Chemicals. (Continued.)											
	Nitrate of Sodium.			Caustic Soda.			Soda Ash and Carbonate.			Sodium Salts, All Other.		
	Long Tons.	Metric Tons.	Value.	Lb.	Kg.	Value.	Lb.	Kg.	Value.	Lb.	Kg.	Value.
1900....	3,089	3,139	$112,550	1,139,954	517,080	$24,228	78,017	35,388	$1,126	270,307	122,610	$2,788
1901....	2,482	2,519	101,489	1,001,940	452,482	21,511	369,521	167,614	5,184	133,400	60,510	3,396
1902....	3,675	3,734	144,650	1,343,132	609,246	28,704	62,653	28,419	931	115,491	52,385	1,626
1903....	4,417	4,488	184,657	1,116,254	506,378	23,227	30,030	13,622	464	42,540	19,294	437
1904....	6,076	6,173	279,864	1,115,600	506,036	23,608	40,351	18,303	593	1,778,616	806,780	25,312
1905....	8,991	9,135	420,613	1,087,772	493,348	22,728	32,221	14,628	473	16,748	7,604	177
1906....	6,660	6,767	324,915	(i)	.....	.....	2,486	1,128	41	1,032,372	468,180	21,624
1907....	7,189	7,274	370,048	(i)	.....	.....	3,100	1,406	53	742,201	336,662	16,099
1908....	9,955	10,113	514,799	(i)	.....	.....	4,645	2,104	77	834,207	378,726	18,255

Year.	Coal, Bituminous.			Copper.						Graphite.	
				Ore and Matte.			Pigs, Bars, Ingots, Old and All Unmanufactured.				
	Long Tons.	Metric Tons.	Value.	Long Tons.	Metric Tons.	Value.	Lb.	Metric Tons.	Value.	Long Tons.	Value.
1900....	6,740	6,848	$19,740	964	979	$170,191	1,281,782	581	$212,264	3	$115
1901....	3,796	4,403	10,627	9,891	10,050	1,406,648	12,888,083	5,846	2,145,468	Nil.	.....
1902....	7,559	7,680	22,153	14,446	14,657	2,239,912	11,629,877	5,275	1,604,522	12	834
1903....	88,468	89,883	453,613	5,750	5,232	852,726	2,093,103	949	261,413	63	4,223
1904....	7,250	7,366	21,910	Nil.	.....	.....	1,088,672	494	140,695	8	455
1905....	3,945	4,008	10,974	Nil.	.....	.....	1,718,584	780	272,945	5	91
1906....	2,541	2,582	13,062	71	72	19,791	1,567,782	711	309,605	3	362
1907....	1,947	1,978	12,199	.....	.....	.....	995,555	451	199,828	1	41
1908....	4,759	4,832	16,313	2	2	50	718,541	326	93,148	.....	.....

Year.	Iron and Steel.											
	Pig Iron.			Scrap.			Bar Iron.			Rails.		
	Long Tons.	Metric Tons.	Value.	Long Tons.	Metric Tons.	Value.	Long Tons.	Metric Tons.	Value.	Long Tons.	Metric Tons.	Value.
1900....	151	153	$6,579	9,079	9,224	$131,241	48	49	$2,447	Nil.	.....	.....
1901....	189	191	6,148	3,331	3,384	51,663	67	68	7,569	Nil.	.....	.....
1902....	250	254	6,286	1,542	1,567	25,020	22	22	1,875	297	302	$7,184
1903....	1,863	1,893	33,996	262	266	2,862	16	16	2,108	739	751	17,560
1904....	1,646	1,672	25,910	190	193	2,367	7	7	765	96	98	2,305
1905....	1,010	1,026	29,047	4,270	4,338	80,623	22	22	2,556	31	31	1,132
1906....	6,750	6,858	236,957	5,111	5,193	101,886	61	62	7,207	Nil.	.....	.....
1907....	2,921	2,968	86,420	157	160	3,378	38	39	3,959	Nil.	.....	.....
1908....	1,827	1,855	52,079	228	232	3,597	26	26	1,271	Nil.	.....	.....

Year.	Iron and Steel. (Continued.)									Lead and Mfrs.
	Steel, Ingots, Blooms, Etc.			Sheets, Plates, Rods, Wire.			Tin and Terne Plates, Taggers Tin.			
	Long Tons.	Metric Tons.	Value.	Long Tons.	Metric Tons.	Value.	Long Tons.	Metric Tons.	Value.	
1900	2	2	$1,342	209	213	11,599	464	470	$37,395	$3,843,881
1901....	2	2	1,059	190	193	17,272	118	120	8,519	4,190,525
1902....	106	108	6,774	236	240	14,221	98	100	7,471	3,553,144
1903....	60	61	5,316	55	56	5,532	2	2	184	2,917,957
1904....	40	41	6,208	108	110	6,482	81	82	5,306	2,880,907
1905....	86	87	15,570	161	164	8,019	26	26	3,014	2,441,166
1906	196	199	14,104	318	323	27,631	0.4	0.4	28	2,307,345
1907....	292	297	25,974	14	14	1,220	42.2	42.9	1,813	2,416,082
1908....	33	34	9,822	66	66	3,441	4.7	4.7	351	3,068,427

Year.	Salt.			Sulphur—Crude.			Tin in Blocks, Pig and Granulated.		
	Lb.	Metric Tons.	Value.	Long Tons.	Metric Tons.	Value.	Long Tons.	Metric Tons.	Value.
1900.....	3,548,724	1,610	$3,907	590	599	$13,495	495	503	$335,377
1901.....	3,699,411	1,678	7,155	207	210	5,086	939	954	562,350
1902.....	2,310,759	1,048	4,544	1,253	1,273	28,024	479	486.	286,897
1903.....	7,804,215	3,585	26,636	967	982	22,658	512	520	317,805
1904.....	2,089,234	948	2,814	2,493	2,533	58,887	519	527	322,234
1905.....	611,912	278	893	1,713	1,741	36,858	557	567	375,763
1906.....	1,462,413	663	1,129	403	409	8,475	807	820	650,411
1907.....	1,166,049	529	1,686	301	306	5,759	502	571	492,415
1908.....	2,525,945	1,146	9,352	380	386	8,500	244	248	156,761

* For convenience in tabulating, the quantities of all petroleum products and their gross values have been divided by 1000

(*a*) From Summary of Commerce and Finance of the United States. (*c*) Total exports of coin and bullion; that is, includes both foreign and domestic. (*d*) Only approximately correct. The Bureau of Statistics reports only the value of silver ores exported, but a much larger amount of silver leaves the country in copper matte, which is classified as copper ore, and no record is kept of its silver contents. The gold in copper matte exported is not included in the exports of gold given in the above table. These figures include ore of both domestic and foreign origin. (*e*) Includes nickel oxide and nickel matte. (*f*) Includes chlorate, chloride, nitrate and all other salts of potassium. (*g*) Reported in barrels, but calculated to gallons, on a basis of 42 gallons to the barrel. (*i*) Barrel of 400 lb. (*k*) Does not include 5,125,103 tons of coal valued at $16,202,150, used for fuel on vessels for foreign trade. (*l*) Included in all other salts of soda. (*m*) Does not include 5,754,176 tons of coal valued at $18,523,594 used for fuel on vessels for foreign trade.

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